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FORM 6-K
SECURITIES AND EXCHANGE COMMISSION
Washington, D.C. 20549

Report of Foreign Private Issuer

Pursuant to Rule 13a-16 or 15d-16
of the Securities Exchange Act of 1934

For the month of January, 2001

SEC MAIL PROCESSING RECEIVED FEB 0 1 2002 WASH. D.C. 354 SECTION

Breakwater Resources Ltd.

(Translation of registrant's name into English)

95 Wellington Street West, Suite 2000, Toronto, Ontario M5J 2N7
(Address of principal executive offices)

Indicate by check mark whether the registrant files or will file annual reports under cover Form 20-F or Form 40-F.

Form 20-F X   Form 40-F.....

Indicate by check mark whether the registrant by furnishing the information contained in this Form is also thereby furnishing the information to the Commission pursuant to Rule 12g3-2(b) under the Securities Exchange Act of 1934.

Yes ..... No X

If "Yes" is marked, indicate below the file number assigned to the registrant in connection with Rule 12g3-2(b): 82- _______

CRGN

Signatures

Pursuant to the requirements of the Securities Exchange Act of 1934, the registrant has duly caused this report to be signed on its behalf by the undersigned, thereunto duly authorized.

BREAKWATER RESOURCES LTD.
(Registrant)

By: *(signed) E. Ann Wilkinson*
E. Ann Wilkinson
Corporate Secretary

Date: January 31, 2002

SECURITIES AND EXCHANGE COMMISSION
Washington, D.C. 20549

FORM 6-K

Report of Foreign Private Issuer

Pursuant to Rule 13-a-16 or 15d-16
of the Securities Exchange Act of 1934

BREAKWATER RESOURCES LTD.
(Exact name of registrant as specified in its charter)

EXHIBITS

INDEX TO EXHIBITS

The following documents are being filed with the Commission as exhibits to, and are incorporated by reference into and form part of, the report on Form 6-K.

Number	Exhibit
1.	News Release issued January 7, 2002.
2.	Independent Technical Report on the Mining Assets of Breakwater Resources Ltd. prepared by SRK Consulting.

EXHIBIT 1

BREAKWATER RESOURCES LTD.
95 Wellington Street West, Suite 2000
Toronto, Ont., M5J 2N7

Tel: (416) 363-4798
Fax: (416) 363-1315

NEWS RELEASE

BREAKWATER ANNOUNCES INCREASED MINERAL RESERVES AND RESOURCES AT EL MOCHITO AND EL TOQUI

January 7, 2002... Breakwater announces that a recent independent review of its operations has resulted in an increase in the mineral reserves and resources at its El Mochito mine in Honduras and its El Toqui mine in Chile.

In preparation for the filing of a prospectus to complete a previously announced rights offering, Breakwater engaged SRK Consultants to prepare an independent technical report as per National Instrument 43-101 ("NI 43-101"). SRK has reported that there are significant additions to the mineral reserves and resources at El Mochito and El Toqui. There were no material changes in mineral reserves and resources at any of Breakwater's other operations.

El Mochito

The following table summarizes the mineral reserves and resources of the El Mochito mine as of November 30, 2001 as estimated:

El Mochito Mineral Reserves and Resources

	November 30, 2001				December 31, 2000			
	Tonnes (000s)	Zn (%)	Pb (%)	Ag (g/t)	Tonnes (000s)	Zn (%)	Pb (%)	Ag (g/t)
Proven and Probable Reserves	3,356	6.8	1.9	78	2,846	7.2	2.0	76
Measured and Indicated Resources	5,937	7.8	2.7	115	4,709	7.8	2.1	88
Inferred Resources	793	7.3	4.3	102	804	8.4	5.9	235

Note: Measured and Indicated Resources include Proven and Probable Reserves

In the December 31, 2000 mineral reserve and resource calculations, Breakwater did not include mineral reserves and resources that have typically been mined from outside of the mineral reserve boundaries. Since 1995, 37% of the mill feed has come from outside the mineral reserve boundaries due primarily to mineral extraction adjacent to reserve outlines. These outlines were based on conservative assumptions. In addition, mining in areas of inferred resources was also considered mining from outside the reserves by Breakwater and was based on an overly conservative classification system. Based on SRK's recommendations this consistent over-extraction has been included as an extraction factor in the conversion of mineral resources to mineral reserves.

From January to November 2001, inclusive, 596,488 tonnes of ore were mined at El Mochito and when taken into account with the new reserve information would account for an increase in proven and probable mineral reserves of 1,106,488 tonnes for the period, or, 1.5 years of mining at the present production rate.

In summary, the large increase in the measured and indicated resources has come primarily from newly discovered mineral reserves and resources through diamond drilling and the application of the extraction factor.

El Toqui

The following table summarizes the mineral reserves and resources of the El Toqui mine as of November 30, 2001 as estimated:

El Toqui Mineral Reserves and Resources

	November 30, 2001			December 31, 2000		
	Tonnes (000s)	Zn (%)	Au (g/t)	Tonnes (000s)	Zn (%)	Au (g/t)
Proven and Probable Reserves	**2,506**	**8.0**	**1.83**	**2,185**	**8.4**	**0.3**
Measured and Indicated Resources	**2,910**	**8.5**	**1.60**	**2,978**	**9.1**	**0.4**
Inferred Resources	**6,362**	**8.0**	**0.41**	**4,334**	**7.7**	**1.1**

Note: Measured and Indicated Resources include Proven and Probable Reserves

The November 30, 2001 mineral reserves were reclassified by SRK. The greatest change in the mineral reserves and resources was a large increase in the inferred resources due to the addition of the Concordia poperty resources. SRK recommended that Breakwater reclassify the unclassified resources at its Concordia property to an inferred mineral resource. The higher gold grade is associated with the Aserradero deposit and the most recent results from the diamond drilling program.

Breakwater's present method of converting resources to reserves included a category called "possible reserves" which was not reported in accordance with the CIM 2000 definitions. On SRK's recommendations, Breakwater has re-classified a portion of these possible reserves as probable reserves. A total of 536,243 tonnes grading 8.0% zinc and 1.7 g/t gold that were originally classified as possible reserves was reclassified as probable reserves based on the CIM 2000 definitions. These reserves occur in extensions to areas that are very well known, for which mining plans exist and that have in the past consistently delivered very close to the planned tonnes and grade.

From January to November 2001, inclusive, 390,956 tonnes of mineral reserves were mined at El Toqui and when this tonnage is taken into account, with the new reserve information, it would account for an increase in proven and probable mineral reserves of 711,356 tonnes for the period, or, 1.7 years of mining at the present production rate.

This news release contained forward-looking statements. When used in this news release the words "anticipate", "believe", "intend", "estimate", "plans", "projects", "expect", "will", "budget", "could", "may", and similar expressions are intended to identify forward-looking statements. To the extent that this news release contains forward-looking statements regarding operating results or business prospects please be advised that the actual operating results and business performance of the Company may differ materially from that anticipated, projected or estimated in such forward-looking statements.

Should you require further information please contact:

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EXHIBIT 2

AN INDEPENDENT TECHNICAL REPORT ON THE MINING ASSETS OF BREAKWATER RESOURCES LTD.

Prepared for:

BREAKWATER RESOURCES LTD.
95 Wellington Street West, Suite 2000
Toronto, Ontario M5J 2N7
Canada

Prepared by:

SRK Consulting
357 Bay Street, Suite 602
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January 18, 2002

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SUMMARY

i. INTRODUCTION

Breakwater Resources Ltd. (BWR) has commissioned SRK Consulting (SRK) to prepare an independent technical report on the mining assets of BWR. The technical report will be included with a prospectus to be issued by BWR in early 2002 in support of a rights offering. The terms of reference were to prepare an independent technical report in compliance with National Instrument 43-101.

BWR is a mineral resource company engaged in the acquisition, exploration, development and mining of mineral properties in the Americas and in North Africa. The principal product is zinc concentrate. BWR also produces lead, copper and gold concentrates, with silver as a by-product. The concentrates are sold to smelters throughout the world. The principal markets for concentrate are in western Europe.

The scope of work undertaken by SRK involved an assessment of BWR's mines, taking into consideration geology, mineral resources, mineral reserves, mining plans, processing, tailings, environmental aspects and an economic analysis of the assets.

ii. DESCRIPTION OF THE MINING ASSETS

ii.i Bouchard-Hébert Mine

Effective May 1, 2000, BWR purchased the Bouchard-Hébert underground zinc/copper/gold/silver mine from Cambior Inc. The Bouchard-Hébert Mine is located 30 kilometres northeast of Rouyn-Noranda in Dufresnoy Township, Québec. The Bouchard-Hébert Mine facilities include an underground mine, a conventional mill and concentrator, which produces separate copper and zinc concentrates. Other facilities include a headframe and hoist building, a paste backfill plant and concentrate loadout facility. Hydro Québec provides electrical power. Access to the Bouchard-Hébert Mine is by a 747 metre deep, three-compartment production/service shaft.

The Bouchard-Hébert Mine is situated within a succession of rhyolitic flows and felsic pyroclastic rocks within the Abitibi Greenstone Belt in the Superior Province. Two volcanogenic massive sulfide (VMS) deposits have been identified to date including the Upper Zone (Mobrun Mine), which has been completely mined-out, and the 1100 lens, which is currently being mined. These two zones are approximately 250 metres across strike from each other. The upper part of the 1100 lens is 300 metres below the surface.

Table i presents a summary of the mineral resources and mineral reserves at the Bouchard-Hébert Mine.

Table i: Bouchard-Hébert Mine, Summary of Mineral Resources and Mineral Reserves

	30 November, 2001					31 December, 2000				
	Tonnes (000s)	Zn (%)	Cu (%)	Au g/t	Ag g/t	Tonnes (000s)	Zn (%)	Cu (%)	Au g/t	Ag g/t
Proven and Probable Reserves	3,219	4.9	0.6	1.0	33	4,414	4.9	0.7	1.1	36
Measured and Indicated Resources	3,402	4.9	0.6	1.0	33	4,665	4.8	0.7	1.1	36
Inferred Resources	-					-				

Measured and Indicated Resources include Proven and Probable Reserves.
No Inferred Resources.

At Bouchard-Hébert a mechanized longhole stope mining method is used with the sequence mining of primary and secondary stopes. The mine infrastructure consists of seven mining levels spaced at 60 metres. A network of ore and waste passes feeds a crushing station on the bottom mine level.

Milling capacity is 3,000 tonnes per day using conventional grinding and flotation to produce separate copper and zinc concentrates. Mill tailings not used for backfill in the mine (approximately 35% of the total tailings) are deposited in the tailings pond as a conventional slurry. Effluent from the tailings pond is sent to a retaining pond and then pumped to a treatment facility prior to seasonal discharge, via a polishing pond, into the environment.

Bouchard-Hébert currently employs 150 people, of which 100 are hourly employees. In addition to the mine's personnel, there are 32 contractors.

ii.ii Nanisivik Mine

In July, 1996 BWR acquired Nanisivik Mines Ltd. from AEC West Limited. In 1997, the zinc/lead/silver underground mine was sold to CanZinco, a wholly-owned subsidiary of BWR.

The Nanisivik Mine is located in Nunavut on northern Baffin Island, on the southern shore of Strathcona Sound, approximately 750 kilometres north of the Arctic Circle. Access to the property is by air or by ship. Ocean shipping is possible from June to October.

The Nanisivik Mine consists of an underground mine, a 2,200 tonne per day concentrator, a power plant, a townsite, recreational facilities and a concentrate storage facility and ship loader. The Nanisivik Mine operates an 11.2 megawatt diesel-electric generating plant to meet all power requirements.

The Main Zone deposit (considered to be Mississippi Valley Type) is about 3 kilometres long. It is oriented east-west, although it is sinuous in plan. The deposit is broadly 'T' shaped, with a flat-topped upper section that is typically about 100 metres wide and 20 metres high. In the mine area, dips are usually quite shallow and the main structure is faulting, much of it of the horst and graben type. The keel section of the deposit extends to about 80 metres below the upper section.

A summary of the mineral resources and mineral reserves at the Nanisivik Mine is presented in Table ii.

Table ii: Nanisivik Mine, Summary of Mineral Resources and Mineral Reserves

	30 November, 2001				31 December, 2000			
	Tonnes (000)	Zn (%)	Pb (%)	Ag (g/t)	Tonnes (000)	Zn (%)	Pb (%)	Ag (g/t)
Proven and Probable Reserves	772	7.43	0.4	30	2,868	6.9	0.4	28
Measured and Indicated Resources	2,012	7.15	0.4	30	4,152	6.3	0.4	24
Inferred Resources	110	4.62	0.4	30	359	5.1	0.4	19

Measured and Indicated Resources include Proven and Probable Reserves

Pb and Ag grades for November 30, 2001 resources and reserves are based on previous mining experience. Grades are expected to be 0.4% Pb and 30 g/t Ag. Pb is not a payable metal, while Ag adds approximately 1% to NSR value.

In the Main Lens the primary mining method is room-and-pillar. This zone has been largely mined out, but post pillars remain that will be recovered. First pass recovery has been approximately 80%. Stope backfill consists of low grade sulfide material sourced from surface stockpiles, development waste or shale from surface. The current short term strategy is to close operations in September, 2002 due to low metal prices.

The mill has a proven capability of processing 780,000 tonnes per year using conventional crushing, rod and ball mill grinding, differential lead and zinc flotation, and concentrate drying. Crushing is done underground using a jaw crusher and a cone crusher. Flotation tailings are pumped through a four kilometre pipeline to the West Twin Lake tailings disposal area.

BWR recently installed a dense media separation (DMS) plant at Nanisivik that came on-stream in July, 2001. The plant allows for blending of mine plan grade ore with run-of-mine resources, and rejecting gangue material from the DMS plant feed, resulting in an increase in mill head grade. The DMS plant is a means of combating low ore grades and mill feed dilution.

BWR employs 184 people at the Nanisivik Mine. Most employees work a rotation that allows for eight weeks at the site followed by four weeks off of the site. While on site, employees work a variety of five, six and seven day shift schedules, with most employees working either ten or twelve hours per day, depending upon operational requirements.

ii.iii Bougrine Mine

In September, 1997 BWR, through a wholly-owned subsidiary, Breakwater Tunisia S.A. ("BWR Tunisia"), acquired the Bougrine zinc/lead underground mine. After some improvements to the mill and the mine, operations commenced on May 2, 1998 and the site was considered to be in commercial production June 1, 1998.

The Bougrine Mine is located between the towns of Le Sers and Le Kef in Tunisia, 160 kilometres southwest of Tunis, the capital. Access is by paved road. As well as a 1,200 tonne per day concentrator, the site facilities include a security/mine rescue/ambulance station, a warehouse, dry facilities, offices, repair and maintenance shops and a concentrate load out facility.

The Bougrine zinc/lead deposit lies in the "Zone de Dômes" of the Tunisian Atlas, a part of the alpine-age Atlas mountain range extending through part of Morocco, Algeria and Tunisia along the Mediterranean coast. The "Zone de Dômes" is characterized by Triassic salt diapers. Using a 6% zinc-equivalent cutoff, the flat lens-shaped zone extends for about 500 metres along strike and 500 metres down-dip. Maximum true width is close to 40 metres and average mineable width is 10 metres. The overall deposit consists of two main styles: stratabound mineralization in the F2 zone, and crosscutting mineralization in the limestone F3 zone. While the F2 zone is larger (originally 3.3 million tonnes with an average grade of 12% Zn), the F3 is smaller but of much higher grade (0.75 million tonnes with 22% Zn). The F2 in long section looks like a donut, fairly circular in shape, with the F3 zone cutting through its centre.

Table iii presents a summary of mineral resources and mineral reserves at the Bougrine Mine.

Table iii: Bougrine Mine, Summary of Mineral Resources and Mineral Reserves

	30 November, 2001			31 December, 2000		
	Tonnes (000s)	Zn (%)	Pb (%)	Tonnes (000s)	Zn (%)	Pb (%)
Proven and Probable Reserves	1,684	10.94	2.1	1,975	11.20	2.1
Measured and Indicated Resources	1,695	13.68	2.6	1,986	14.10	2.6
Inferred Resources	604	8.30	1.4	604	8.30	1.4

Measured and Indicated Resources include Proven and Probable Reserves
1 January, 2001 to 30 November, 2001 production includes 291,345 tonnes grading 12.7% Zn and 2.35% Pb from within reserve outlines, while an additional 53,078 tonnes grading 6.77% Zn and 1.15% Pb were mined outside of the reserve limits, and therefore, not subtracted from 31 December, 2000 reserves.

The main mining method is modified room-and-pillar with delayed unconsolidated backfill between sublevels. In the case of the F2 zone, this could also be described as a modified AVOCA method where intact slices are kept in the mineralized body to increase overall stability. Pillar spacing is approximately 20 metres along strike. The system allows for an extraction of approximately 80% of the in-situ resources. The geometry of the F2 zone with its 45° dip requires a balance between additional dilution and loss, as the minimum inclination of the slot raises and the footwall contact of the stopes is 60°. The material is extracted by conventional drifting and slashing on sublevels with 15 metre spacing between sublevels. Longhole blasting is undertaken to extract the material between the sublevels. Backfill is then placed via the upper sublevel, which requires that some short access drifts be developed in the hanging wall sequence.

Grinding is accomplished in a rod mill in open circuit, and a ball mill in closed circuit with two stages of cycloning to produce the ground feed to flotation. Conventional, differential flotation is employed to produce saleable lead and zinc concentrates. The flotation tailings are thickened in the tailings impoundment with the overflow (tailings effluent) recycled to the mill process and the thickener underflow pumped to an impoundment area adjacent to the mill. The tailings effluent mostly evaporates and only a limited quantity is available for recycling to the mill. There is zero discharge of tailings effluent to the environment.

The proximity of the mineralized body to a salt dome results in the process water from the mine being highly saline. This high salinity causes poor selectivity in the flotation circuit when this water is used, and consequently impacts the concentrate grade. A desalination plant, designed to provide better process water quality and thus allow better product quality, was constructed during 2000 and commenced operation in late 2001.

BWR Tunisia's workforce consists of 301 people, 10 being expatriate personnel and 291 being local personnel. Of the total 301 personnel, 289 are at the mine site and 12 are in Tunis.

ii.iv El Mochito Mine

In March 1990, BWR acquired American Pacific Mining Corporation, Inc. ("AMPAC") by way of an amalgamation of AMPAC with a wholly-owned subsidiary of BWR now named Santa Barbara Mining Company, Inc. In 1998 the El Mochito zinc/lead/silver underground mine was transferred to American Pacific Honduras S.A. de C.V., a wholly-owned subsidiary. Except for a short period in 1987, the mine has been in continuous production since 1948.

The El Mochito Mine is located in northwest Honduras, near the town of Las Vegas. The closest major city is San Pedro Sula, the commercial centre of the country, approximately 88 kilometres northeast of the mine. The concentrates are trucked to BWR's concentrate/storage shed on tidewater at Puerto Cortés approximately 115 kilometres northeast of the mine. Access is by paved roads.

The El Mochito Mine is comprised of an underground zinc/lead/silver mine and a 2,000 tonne per day concentrator. The El Mochito property includes 53 exploitation concessions totalling 10,835 hectares of mineral rights.

Zinc/lead/silver mineralization at El Mochito occurs in sedimentary rocks of Cretaceous age, and belongs to the economically important class of high-temperature replacement zinc/lead deposits in carbonates. The replacement deposits can take two shapes, some follow the essentially flat bedding of their host rock ("mantos") while others cut across the rocks ("chimneys" or "pipes").

A summary of mineral resources and mineral reserves at the El Mochito Mine is presented in Table iv.

Table iv: El Mochito Mine, Summary of Mineral Resources and Mineral Reserves

	30 November, 2001				31 December, 2000			
	Tonnes (000s)	Zn (%)	Pb (%)	Ag (g/t)	Tonnes (000s)	Zn (%)	Pb (%)	Ag (g/t)
Proven and Probable Reserves	3,356	6.8	1.9	78	2,846	7.2	2.0	76
Measured and Indicated Resources	5,937	7.8	2.7	115	4,709	7.8	2.1	88
Inferred Resources	793	7.3	4.3	102	804	8.4	5.9	235

Measured and Indicated Resources include Proven and Probable Reserves

El Mochito utilizes a combination of mining methods dependent upon the size, geometry and geotechnical considerations of the various zones. Within the Nacional zone, BWR is presently mining an area of highly-variable grade and geometry by "post-pillar" methods, with hydraulic backfill, which yields 80% mining recovery. Within the remnant, higher-grade fringe areas and pillars of the San Juan zone, conventional shrinkage and cut-and-fill stoping is practiced. Mining is currently following a plan to recover higher grade mineralization due to low metal prices.

The mill is a conventional, differential sulfide flotation mill capable of processing 2,000 tonnes per day of ore, producing separate zinc and lead concentrates. The process consists of crushing, grinding, flotation, concentrate dewatering and tailings disposal.

The flotation tailings are pumped to the mine backfill facility where the tailings, upon demand, are subjected to cycloning to produce hydraulic backfill for the mine or by-passed to the tailings impoundment area. No tailings effluent is recycled to the mill, but it is treated and discharged.

The El Mochito Mine operates under the Honduras Law of the Environment (1993). This law has a very limited section referring specifically to mining, for which the standards are for the most part World Bank standards. BWR believes the mine meets relevant North American standards as the relevant guidelines for its mining practices.

The number of personnel currently employed at the El Mochito Mine is 718. This includes 25 expatriate staff, 140 clerical and local staff, 454 weekly employees and 99 temporary employees. There are also 86 security guards and 135 contract miners.

The El Mochito Mine has a long history of replenishing mineral resources and mineral reserves, and has excellent exploration potential adjacent to mine workings and elsewhere on the property.

ii.v El Toqui Mine

In August 1997, BWR purchased all of the outstanding common shares of Sociedad Contractual Minera El Toqui ("El Toqui"), which owns the El Toqui zinc/gold underground mine in Chile. The El Toqui Mine has been in production since 1987 at a rate of approximately 400,000 tonnes per year. The El Toqui Mine is located in Chile's Region XI, approximately 1,350 kilometres south of Santiago and 120 kilometres northeast of Coyhaique.

The El Toqui property covers approximately 1,200 square kilometres of mountainous terrain, and includes the currently producing Doña Rosa zinc/gold mine, the former producing San Antonio and Mallin-Monica zinc/lead mines, and an 1,100 tonne per day concentrating plant. The property also includes mine and maintenance offices, laboratory and service buildings and houses as well as a ship loader at Puerto Chacabucco. The concentrate produced by the plant is transported to a deep-water port for export. El Toqui owns and operates a dedicated hydroelectric power generating plant as well as a diesel generating plant.

The areas covering the El Toqui claims are mainly characterized by zinc, gold and copper mineralization in vein, stockwork and disseminated form. The host rocks are volcanic and sedimentary types, from the Jurassic-Cretaceous period. The El Toqui property can be divided into two sectors in terms of mineralization, namely zinc and zinc/gold mineralization. Zinc/gold mineralization at El Toqui occurs as manto deposits hosted by an 8 metre to 12 metre thick bed of fossiliferous limestone, known as the Main Manto unit.

The El Toqui deposits all dip gently to the south, and are amenable to room and pillar mining, with 11 metre rooms around 8 metre pillars. The Doña Rosa Mine is accessed via an adit. Mining is carried out in a single horizon from flat lying mineralization.

Table v presents a summary of mineral resources and mineral reserves at the El Toqui Mine.

Table v: El Toqui, Summary of Mineral Resources and Mineral Reserves

	30 November, 2001			31 December, 2000		
	Tonnes (000s)	Zn (%)	Au (g/t)	Tonnes (000s)	Zn (%)	Au (g/t)
Proven and Probable Reserves	2,506	8.0	1.8	2,185	8.4	0.3
Measured and Indicated Resources	2,910	8.5	1.6	2,978	9.1	0.4
Inferred Resources	6,362	8.0	0.4	4,334	7.7	1.1

Measured and Indicated Resources include Proven and Probable Reserves

The operations are mechanized utilizing electric-hydraulic drill jumbos, a roofbolting jumbo, 8 cubic yard scooptrams and 30 tonne capacity trucks. All ore and waste rock haulage is done by a contractor, with the loading done by El Toqui personnel.

Crushing is performed in three stages (jaw and cone crushers with vibrating screens) to reduce the ore to minus seven millimetres. Grinding utilizes three parallel ball mill grinding circuits, each ball mill circuit is in closed circuit with cyclones. Conventional differential flotation including rougher and cleaner flotation circuits produce saleable gold/lead concentrate and zinc concentrate. The concentrates are separately thickened and filtered on vacuum and pressure filters before being conveyed to a storage area in the mill. The flotation tailings are pumped to a tailings impoundment area from which the tailings effluent is recycled to the mill.

The El Toqui Mine operates under various Chilean permits and authorities related to the environment. Mill tailings are discharged to the Confluencia tailings impoundment where they are separated via hydro-cyclone into coarse and fine fractions. The coarse fraction is utilized to construct the tailings dams in a centerline configuration in accordance with Chilean regulations. The fine fraction reports to the interior of the pond where it is deposited subaqueously. Reclaim water is pumped back to the mill for mineral processing while any excess is discharged via two sedimentation ponds to the Toqui River.

El Toqui currently has a workforce of 239 people, consisting of 197 hourly personnel and 42 staff personnel. Of the 239 personnel, 237 are at the minesite, and 2 are in the Santiago office.

The El Toqui Mine has a history of replenishing mineral resources and mineral reserves, and has excellent exploration potential adjacent to mine workings and elsewhere on the property.

ii.vi Langlois Mine

On May 1, 2000 BWR purchased the Langlois zinc/copper underground mine from Cambior Inc. The Langlois Mine is located in north-western Québec, approximately 48 kilometres northeast of the town of Lebel-sur-Quévillon and 213 kilometres north of Val d'Or. The mine is accessed via a gravel road.

The underground mine facilities include a headframe, a paste backfill plant, mechanical and electrical shops, a service building, a zinc/copper concentrator and a tailings pond. The mine produces zinc and copper concentrates, which are sold and forwarded to smelters for further processing. From such processing, gold and silver also result as by-products.

The mine is equipped with a 905 metre deep four-compartment shaft. There are two hoists; a 3.0 metre diameter double drum hoist for skipping and a 2.4 metre diameter double drum service hoist. Two seven tonne skips are used to hoist the blasted material to surface. At present, there is a crusher station on the bottom level of the mine. However, due to problems with severe ore-pass wall erosion and consequent dilution, a new loading facility was established on the Level 11.

The Langlois Mine contains four zinc-rich orebodies consisting of zones of massive sulfide, primarily pyrite and sphalerite, occurring within a thick, highly deformed felsic volcanic sequence injected by numerous barren mafic dikes. Each massive sulfide body is relatively thin (1 to 8 metres), but with considerable vertical and lateral extensions (> 500 metres in either direction). The massive sulfide zones trend easterly with a near vertical dip, sub-parallel to the regional structural fabric. The zones are stacked across the felsic sequence along a narrow corridor slightly oblique to the main structural trend. From southwest to northeast the zones are: Zone 5 (small uneconomic lens near surface), Zone 4, Zone 3 and Zone 97. In longitudinal section, each massive sulfide zone portrays an elongated lensoid shape, whose long axis plunges moderately towards the southeast, parallel to the plunge of the regional stretching lineation. In addition, the center of gravity of each lens becomes progressively deeper moving along the stacking corridor toward the northeast. Consequently, the top of Zone 97 is located at approximately 300 metres below surface. Ore production at the Langlois Mine has come exclusively from two zones, namely Zones 3 and 4. Zone 97 was discovered in 1994 but was not fully defined until recently.

A summary of mineral resources and mineral reserves is presented in Table vi.

Table vi: Langlois Mine, Summary of Mineral Resources and Mineral Reserves

	30 November, 2001					31 December, 2000				
	Tonnes (000s)	Zn (%)	Cu (%)	Au g/t	Ag g/t	Tonnes (000s)	Zn (%)	Cu (%)	Au g/t	Ag g/t
Proven and Probable Reserves	2,903	11.2	0.7	0.1	53	3,892	10.2	0.6	0.1	49
Measured and Indicated Resources	4,862	11.0	0.7	0.1	53	3,942	11.8	0.7	0.1	47
Inferred Resources	1,547	8.1	0.5	0.1	37	1,111	9.6	0.6	0.1	35

Measured and Indicated Resources include Proven and Probable Reserves

The mining method initially implemented in 1996 at the Langlois Mine was transverse longhole stoping. The level spacing was 60 metres and blocks were 20 metres along strike and 4 to 5 metres thick. Once terminated, the stopes were filled with high-density fill with 78% solids. The mining method was plagued from the beginning with excessive dilution. In 1997, a corporate decision was made to stop operations and convert the 60 metre high stopes into smaller 15 metre or 30 metre stopes depending on the width of the mineralization.

The revised mining method saw the block sizes reduced to 20 metre heights and 20 metre lengths (15 metres between sublevels). Where widths were over 3 metres, sublevels were spaced at 30 metres. The mine is largely trackless, apart from some sublevels that do not have ramp access. In general, where the mineralized widths were over 3 metres, a ramp access to the sublevel was made. The current mine design and plan are to recover higher grade, lower tonnage reserves.

The Langlois mill processed approximately 1,800 tonnes per day, five days per week, however, it has a capacity of 2,500 tonnes per day. Copper and zinc concentrates are produced by differential flotation, with payable gold and silver recovered in the copper concentrate. Approximately 60% of the tailings are used in paste backfill, and the remainder are sent to the tailings pond. Most of the tailings pond effluent overflow is recycled as mill process water, with a portion of the tailings pond effluent overflow released to the Wedding River after treatment with caustic lime to maintain pH levels acceptable to the regulations.

During the current temporary closure period (which began December, 2000) the mine is being kept on a care and maintenance program. The underground workings continue to be dewatered and all water is treated with lime before being sent to the tailings pond.

When the mine was in full production there were a total of 162 workers, 19 contractors and 59 staff. The hourly paid employees are unionized.

ii.vii Caribou Mine

BWR acquired the Caribou Mine in the fall of 1990. The underground zinc/lead/silver mine and concentrator operated for seven months in 1990. On October 26, 1990, operations were suspended due to poor recoveries and falling metal prices, and the facilities were placed on care and maintenance. The Caribou open pit mine was acquired in October 1995. It had not previously been in production. Ore from the Caribou underground mine and open pit mine was processed in the mill from July 1997 to August 1998, but the metallurgical recovery fell short of anticipated levels. As metal prices were declining, the operation was placed on care and maintenance again.

The Caribou Mine consists of an underground mine and mill located in Restigouche County in northeastern New Brunswick, 50 kilometres west of Bathurst, New Brunswick, and an open pit mine (formerly known as the Restigouche property) located approximately 80 kilometres west of Bathurst and 30 kilometres from the Caribou mill.

A 3,000 tonne per day concentrator complex is located at the Caribou underground mine site. Other surface facilities include a headframe and production hoisting system, a shop-warehouse complex, a compressor building and compressors, a mine dry, assay laboratory and an administration building. Electric power is obtained from New Brunswick Power.

The Caribou Mine is comprised of two deposits, the Caribou deposit and the Restigouche deposit. At the Caribou Mine, the mineralization delineated to date consists of six discrete massive sulfide lenses, overlapping in part, and distributed in an echelon fashion in, and conformable within, the Mine Sequence.

The Caribou open pit, or Restigouche deposit, is tabular in form, with the long axis striking north 15° west, and with a dip of less than 35° to the west. The tabular, massive sulfide body extends 425 metres along strike and is up to 120 metres wide and 45 metres thick.

Table vii presents a summary of the mineral resources and mineral reserves at the Caribou Mine.

Table vii: Caribou Mine, Summary of Mineral Resources and Mineral Reserves

	30 November, 2001				**31 December, 2000**			
	Tonnes (000s)	**Zn (%)**	**Pb (%)**	**Ag g/t**	**Tonnes (000s)**	**Zn (%)**	**Pb (%)**	**Ag g/t**
Proven and Probable Reserves	5,057	6.5	3.4	90	5,057	6.5	3.4	90
Measured and Indicated Resources	5,152	7.4	3.9	95	5,152*	7.4	3.9	95
Inferred Resources	4,163	6.7	3.2	98	4,163	6.7	3.2	98

Measured and Indicated Resources include Proven and Probable Reserves
*Previously stated by BWR as 5,264,000 tonnes.

An updated 2000 re-opening plan indicates that the Caribou Mine can be restarted for a capital cost prior to start-up of approximately $11.8 million (including mill modifications), excluding working capital. The re-opening plan calls for production from the Caribou underground mine at the rate of 1,650 tonnes per day and from the Caribou open pit mine at the rate of 1,350 tonnes per day. The reserves from the open pit will be depleted in 21 months and the underground reserves accessed from the open pit will be developed in time to produce 500 tonnes per day. At this time, production from the Caribou underground will be increased to 2,500 tonnes per day.

The mining method proposed will be the same as used in the previous operating period. This is an AVOCA type method with the use of development waste and stockpiled surface waste as backfill. BWR plans to mine the Caribou open pit in two stages. The first stage is expected to be the resumption of the open pit operation with the second stage being the development of the underground deposit commencing immediately upon depletion of the open pit reserves.

From the compilation of pilot plant test results and data in November, 1998, the operating period of 1997-98 and a 2000 Lakefield Research study, it was concluded that a number of process flowsheet modifications are required to obtain the desired response to treatment by Caribou mineralized material. The new flowsheet developed at Lakefield Research is less complex than the existing flowsheet. While the plant has been redesigned to handle higher circulating loads, the actual circulating loads will be less onerous than in the past due to the simplified flowsheet and by ensuring that the fine grinds, proven to be required to process Caribou materials, are achieved.

BWR and CanZinco Ltd. (CanZinco) have an agreement with the Province of New Brunswick whereby the liability of CanZinco for reclamation work, rehabilitation work, other environmental work or environmental liabilities of any nature related to exploration and development activities at the Caribou Mine that took place prior to October 29, 1993 be limited to $3 million. The Province of New Brunswick is responsible for liabilities exceeding the $3 million.

The Caribou Mine employed 199 personnel at the time the operation was placed on care and maintenance.

iii. MINERAL RESOURCES AND MINERAL RESERVES

Table viii on the next page presents the mineral resources and mineral reserves for each mine, classified by BWR's Qualified Person as of 30 November, 2001 according to the "*CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines*" (August, 2000). Accordingly, the Resources have been classified as Measured, Indicated or Inferred and the Reserves have been classified as Proven and Probable based on the Measured and Indicated Resources.

Table viii: Consolidated Mineral Resources and Mineral Reserves for BWR's Mining Assets

	30 November, 2001						31 December, 2000					
	Tonnes (000's)	Zn (%)	Pb (%)	Cu (%)	Ag (g/t)	Au (g/t)	Tonnes (000's)	Zn (%)	Pb (%)	Cu (%)	Ag (g/t)	Au (g/t)
PROVEN AND PROBABLE RESERVES												
Bouchard-Hébert	3,219	4.9		0.6	33	1.0	4,414	4.9		0.7	36	1.1
Nanisivik	772	7.4	0.4		30		2,868	6.9	0.4		28	
Bougrine	1,684	10.9	2.1				1,975	11.2	2.1			
El Mochito	3,356	6.8	1.9		78		2,846	7.2	2.0		76	
El Toqui	2,506	8.0				1.8	2,185	8.4				0.3
Langlois	2,903	11.2		0.7	53	0.1	3,892	10.2		0.6	49	0.1
Caribou	5,057	6.5	3.4		90		5,057	6.5	3.4		90	
Total	**19,497**	**7.6**					**23,237**	**7.5**				
MEASURED AND INDICATED RESOURCES												
Bouchard-Hébert	3,402	4.9		0.6	33	1.0	4,665	4.8		0.7	36	1.1
Nanisivik	2,012	7.2	0.4		30		4,152	6.3	0.4		24	
Bougrine	1,695	13.7	2.6				1,986	14.1	2.6			
El Mochito	5,937	7.8	2.7		115		4,709	7.8	2.1		88	
El Toqui	2,910	8.5				1.6	2,978	9.1				0.4
Langlois	4,862	11.0		0.7	53	0.1	3,942	11.8		0.7	47	0.1
Caribou	5,152	7.4	3.9		95		5,152*	7.4	3.9		95	
Total	**25,970**	**8.4**					**27,584**	**8.2**				
INFERRED RESOURCES												
Bouchard-Hébert	-						-					
Nanisivik	110	4.6	0.4		30		359	5.1	0.4		19	
Bougrine	604	8.3	1.4				604	8.3	1.4			
El Mochito	793	7.3	4.3		102		804	8.4	5.9		235	
El Toqui	6,362	8.0				0.4	4,334	7.7				1.1
Langlois	1,547	8.1		0.5	37	0.1	1,111	9.6		0.6	35	0.1
Caribou	4,163	6.7	3.2		98		4,163	6.7	3.2		98	
Total	**13,579**	**7.6**					**11,375**	**7.5**				

Measured and Indicated Resources include Proven and Probable Reserves.
*Previously stated by BWR as 5,264,000 tonnes.

iv. ECONOMIC ANALYSIS

BWR has developed an internal long-range plan (LRP) technical model for its operating mines (Bouchard-Hébert, Bougrine, El Mochito, El Toqui, Nanisivik) and non-operating mine properties (Langlois, Caribou).

SRK is of the opinion that the economic modeling parameters used by BWR in its LRP are fair and reasonably reflect current metal and financial market trends.

All operating mines in the LRP exhaust their current mineral reserves with the exception of the Nanisivik Mine. Due to low market prices BWR has elected to cease mining operations at Nanisivik prior to complete depletion of its ore reserves.

BWR plans to mine currently-delineated proven and probable reserves that total 11.1Mt of ore through the term of the LRP. Production rates, ore grades and metallurgical recoveries over this period have been reviewed by SRK and found to be within those historically achieved at each respective operation. Based upon current market forecasts, BWR should expect a run-of-mine (RoM) zinc equivalent grade of 8.89% (Table ix).

Table ix: BWR LRP, Summary of Technical Inputs (all Operating Mines 2002 - 2008)

Model Parameter	Zinc	Lead	Copper	Gold	Silver
Ore Milled (all mines)	11,136Mt				
RoM Grade	7.29%	0.64%	0.17%	0.70g/t	32.66g/t
Zn Equivalent	8.89%				
Contained Metals	811.5kt	71.5kt	18.7kt	249koz	11,694koz
Metallurgical Recovery	89%	90%	79%	54%	92%
Metal In Concentrates	1,591Mlb	141.8Mlb	32.5Mlb	134koz	10,758koz
Payable Metals	1,347Mlb	133Mlb	30.5Mlb	116koz	7,403koz

Projected revenues are in line with historical achievements and SRK is of the opinion that the values presented in the model are reasonable.

BWR projects unit operating costs for combined operations to be US$26.93/t-milled and US$26.64/t-milled in 2001 and 2002, respectively. El Mochito and Nanisivik presently represent the highest unit operating cost mines, while Bouchard-Hébert and El Toqui are currently the lowest unit cost producers.

Operating cost projections for each mine reflect historical achievements. SRK is of the opinion that these operating cost projections are achievable given that historical mine production rates will be maintained during the period of this analysis, thus keeping fixed costs under control (on a unit cost basis).

In BWR's LRP, the operating mines are planned to exhaust their currently defined mineral reserves, and for modeling purposes are assumed to close when the present mineral reserves run out. The actual timing of the closures will ultimately depend on the success of each operation to replenish reserves. Several of BWR's mines have a history of identifying additional resources and subsequently converting these resources to reserves, particularly El Mochito and El Toqui. However, as a condition of National Instrument 43-101, only proven and probable reserves can be considered in a model such as the LRP. As such, capital expenditure profiles call for completion of the underground mine development programs, mine and mill equipment requirements and closure and reclamation costs. Also included in the capital cost estimate are estimated salvage value credits. Ongoing exploration programs account for a small portion of planned capital expenditures

Mining operations are expected to continue operating under historic operating parameters, and given projected market prices an undiscounted cash flow of US$13.3 million is projected over the 2-year review period of 2001 to 2002.

Cash outflow in 2001 is projected to be US$6.2 million. Although revenues roughly equal expenses in 2001, it is the estimate of capital requirements that results in a negative cash flow for the year. Receivables are projected to exceed payables in 2002, which is projected to result in a cash flow of approximately US$19.5 million in 2002.

In its analysis, SRK has completed its review of BWR's LRP and concludes that projections and results presented by BWR are fair and reasonable. The net present value of the LRP cash flow, for fiscal years 2002 to 2008, at a discount rate of 10%, is projected to be US$55.2 million.

v. MATERIAL ISSUES AND OPPORTUNITIES

SRK finds no material issues associated with technical-economic aspects of the BWR model and concludes that the results shown are fair and reasonable. During the course of SRK's review, a number of risks and opportunities at BWR's mines were identified. These have been discussed fully with BWR management, together with potential strategies to mitigate risks and pursue opportunities.

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AN INDEPENDENT TECHNICAL REPORT ON THE MINING ASSETS OF BREAKWATER RESOURCES LTD.

1. INTRODUCTION AND TERMS OF REFERENCE

Breakwater Resources Ltd. (BWR) has commissioned SRK Consulting (SRK) to prepare an independent technical report on the mining assets of BWR. The technical report will be included with a prospectus to be issued by BWR in early 2002 in support of a rights offering.

The terms of reference were to prepare an independent technical report in compliance with National Instrument 43-101.

1.1 Scope of Work

The scope of work undertaken by SRK involved an assessment of the following aspects of the mining assets of BWR:

- Geology
- Mineral Resources
- Conversion of Mineral Resources to Mineral Reserves
- Life of Mine (LoM) Plans
- Rock Mechanics
- Metallurgy and Processing Plants
- Tailings/Waste Disposal
- Environmental – Including Water Management, Mine Closure and Salvage Value
- Infrastructure – Capital Expenditures
- Economic Analysis – Cash Flow Model.

1.2 Basis of the Technical Report

In summary, this technical report has been based on:

- Inspection visits to surface and underground operations, processing facilities, surface structures and associated infrastructure at each of the mining assets during 2001;
- Full access to key mine and head office personnel for discussion and enquiry;
- A review and, where appropriate, modification of BWR's estimates and classification of Mineral Resources and Mineral Reserves, including the methodologies applied by BWR in determining such estimates and classifications, for each of the Mining Assets including check calculations where appropriate;

- A review and where appropriate, modification of BWR's LoM Plans and supporting documentation and the associated technical-economic parameters, including assumptions regarding future operating costs, capital expenditures and saleable concentrates for the mining assets;
- An examination of documentation made available by BWR in respect of the Material Properties in support of the operational planning and in particular the LoM plans and one-year budgets;
- A desk-top review of the data pertaining to BWR's Minesite Exploration Properties, including Mineral Resource estimations and classifications, where appropriate.

SRK's approach in undertaking a review of the Mineral Resource and Mineral Reserve estimations and classifications is detailed in Section 4. In summary, SRK has generated audited Mineral Resource and Mineral Reserve statements based on a review of the Qualified Person's reports, LoM plans and the methodologies applied for estimation and classification of Mineral Resources and Mineral Reserves.

Given the extensive operating history of the Mining Assets, geological investigations, reconciliation studies, independent check assaying and, in certain instances, independent audits, SRK has not found it necessary to independently verify the underlying data, including sampling and assay data.

1.3 Limitations and Reliance on Information

SRK's opinion contained herein and effective 1 January 2002, is based on information provided to SRK by BWR throughout the course of SRK's investigations as described in Section 1.2, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time.

The achievability of LoM plans, budgets and forecasts are inherently uncertain. Consequently actual results may be significantly more or less favourable.

This report includes technical information, which requires subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

1.4 Disclaimers and Cautionary Statements for US Investors

In considering the following statements SRK notes that the term "ore reserve" for all practical purposes is synonymous with the term "Mineral Reserve".

The United States Securities and Exchange Commission (the "SEC") permits mining companies, in their filings with the SEC, to disclose only those mineral deposits that a company can economically and legally extract or produce from. Certain items are used in this report, such as "resources," that the SEC guidelines strictly prohibit companies from including in filings with the SEC.

Ore reserve estimates are based on many factors, including, in this case, data with respect to drilling and sampling. Ore reserves are determined from estimates of future production costs, future capital expenditures, future product prices and the exchange rate between the Canadian Dollar ("CDN$") and the United States Dollar ("US$"). The reserve estimates contained in this report should not be interpreted as assurances of the economic life of the Mining Assets or the future profitability of operations. Because ore reserves are only estimates based on the factors described herein, in the future these ore reserve estimates may need to be revised. For example, if production costs decrease or product prices increase, a portion of the resources may become economical to recover, and would result in higher estimated reserves.

The LoM Plans and the technical economic projections include forward-looking statements that are not historical facts and are required in accordance with the reporting requirements of the Ontario Securities Commission (OSC). These forward-looking statements are estimates and involve a number of risks and uncertainties that could cause actual results to differ materially.

SRK has been informed by BWR that there is no current litigation that may be material to any of BWR's mining assets, and that BWR is not aware of any pending litigation that may be material to any of BWR's mining assets.

1.5 Exchange Rates

For the purpose of this report the exchange rates, representing Financial Years ending 31 December, are CDN$0.65/US$1.00 for 2001, and CDN$0.64/US$1.00 for 2002 and thereafter.

Further, all assumed costs (unless otherwise stated) including operating, capital and environmental costs, are quoted in 1 December 2001 Canadian dollar terms.

1.6 Mineral Resource/Mineral Reserve Statements and LoM Plans

SRK's technical and economic appraisal presented in this technical report has been based on the latest LoM Plans and Mineral Resource and Mineral Reserve statements for the Mining Assets. Table 1.1 summarises the respective timing of the Mineral Resource and Mineral Reserve statements and the LoM plans for the Mining Assets.

Table 1.1: Summary of Mineral Resource and Mineral Reserve Statements and LoM Plans – Effective Date

MINE	Resource Statement	Resource Updates By SRK	LoM Plan
Bouchard-Hébert Mine	31-Dec-00	Subtract 2001 Production	31-Dec-00
Nanisivik Mine	30-Jun-01	Subtract 2001 Production	Oct-01
Bougrine Mine	31-Dec-00	Subtract 2001 Production	31-Dec-00
El Mochito Mine	30-Nov-01	Updated	31-Dec-00
El Toqui Mine	30-Nov-01	Updated	30-Nov-01
Langlois Mine	Jun-01	Unchanged	Jun-01
Caribou Mine	Mar-99	Unchanged	Dec-00

The Mineral Reserve statements and LoM Plans for the Mining Assets as reported in this technical report have been updated to reflect depletion from mining as at 30 November, 2001.

1.7 Short Term Strategies

The technical and economic appraisal reported herein has been based on the latest mineral resource and mineral reserve statements and LoM Plans which are formal documents prepared by each mine incorporating a long term view of zinc, lead, copper, silver and gold prices. At the time of this appraisal, the mines are investigating and some are implementing short term strategies (+/- 1 year) to address the low metals prices. Such strategies include deferring capital expenditures, mining to a higher cutoff grade, extending care and maintenance provisions and even premature closure, until metals prices return to more realistic prices. Given the assumed medium term metals prices as identified in Table 1.2, and for the purpose of this technical report, these short term strategies which have not been explicitly defined or determined in all cases were not considered as part of the LoM Plans and resource and reserve statements reported herein.

Table 1.2: Assumed Medium Term Metals Prices

METAL	PRICE
Zinc	US$0.50/lb
Lead	US$0.22/lb
Copper	US$0.68/lb
Silver	US$4.50/oz
Gold	US$275/oz

1.8 Qualifications of Consultant

The SRK Group comprises 500 staff, offering expertise in a wide range of resource engineering disciplines. The SRK Group's independence is ensured by the fact that it holds no equity in any project and that its ownership rests solely with its staff. This permits the SRK Group to provide its clients with conflict-free and objective recommendations on crucial judgement issues. The SRK Group has a demonstrated track record in undertaking independent assessments of resources and reserves, project evaluations and audits, technical reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide. The SRK Group has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs.

This technical report has been prepared based on a technical and economic review by a team of nine consultants sourced from the SRK Group's North American and UK offices. These consultants are specialists in the fields of geology, mineral resource and mineral reserve estimation and classification, underground and open pit mining, rock mechanics engineering, metallurgical processing, hydrogeology and hydrology, tailings management, infrastructure, environmental management and mineral economics.

Neither SRK nor any of its employees and associates employed in the preparation of this report has any beneficial interest in BWR or in the assets of BWR. SRK will be paid a fee for this work in accordance with normal professional consulting practice.

The individuals who have provided input to this technical report, who are listed below, have extensive experience in the mining industry and are members in good standing of appropriate professional institutions.

- Neal Rigby, C.Eng, MIMM, MAIME, PhD;
- Chris Page, P.Eng, PhD;
- Allan McCracken, B.Sc., C.Eng, MIMM;
- Michael Michaud, P.Geo., M.Sc;
- Andrew Bradfield, B.Sc., P.Eng.;
- Ken Reipas, B.Sc., P.Eng.;
- Louis Bernard, B.Sc., P.Eng.;
- Nick Michael, B.Sc., MAIME, MBA;
- Diana Sollner, B.Sc., P.Eng.

In compliance with National Instrument 43-101, the Qualified Person that made a personal inspection of each operating property of BWR is listed in Table 1.3.

Table 1.3: List of Qualified Persons

BWR Mine Site	Qualified Person
Bouchard-Hébert	Michael Michaud
Caribou	Michael Michaud
Langlois	Michael Michaud
Bougrine	Allan McCracken
El Mochito	Chris Page
El Toqui	Chris Page
Nanisivik	Chris Page

2. INFORMATION ON BWR

This section of the report describes the history, development and organizational structure of BWR. Historic metal production is summarized, and each Mining Asset is described.

2.1 History and Development of BWR

BWR was incorporated under the laws of the Province of British Columbia under the name "Gambier Exploration Ltd." on October 15, 1979. Effective June 23, 1981, the name was changed to "Breakwater Resources Ltd.". BWR was continued under the Canada Business Corporations Act effective May 11, 1992. By articles of amendment dated June 8, 1995, the then outstanding Common Shares were consolidated on a 1-for-400 basis and then immediately subdivided on a 20-for-1 basis (being an effective 1-for-20 consolidation).

The registered and principal office of BWR is situated at Suite 2000, 95 Wellington Street West, Toronto, Ontario M5J 2N7, Canada. The telephone number is (416) 363-4798 and the fax number is (416) 363-1315.

2.2 Business Overview

BWR is a mineral resource company engaged in the acquisition, exploration, development and mining of mineral properties in the Americas and in North Africa. The principal product is zinc concentrate. BWR also produces lead, copper and gold concentrates, with silver as a by-product. The concentrates are sold to smelters throughout the world. The principal markets for concentrate are in western Europe.

2.3 Organizational Structure

Table 2.1 sets forth the name of the principal subsidiaries of BWR and the jurisdiction of incorporation and the direct or indirect percentage ownership by BWR of each such subsidiary.

Table 2.1: Subsidiaries of BWR

Name of Subsidiary	Jurisdiction of Incorporation	Percentage Ownership
Breakwater Tunisia S.A.	Tunisia	100%
CanZinco Ltd.	Canada	100%
Consell Marketing Inc.	Barbados	100%
American Pacific Honduras S.A. de C.V.	Honduras	100%
Sociedad Contractual Minera El Toqui	Chile	100%
3064077 Canada Inc.	Canada	99%

Set forth below are the principal mining assets owned by BWR. Interest in the properties varies and may be subject to royalty or other obligations to third parties. In summary:

- BWR owns and operates the Bouchard-Hébert zinc/copper/gold/silver mine located near Rouyn-Noranda, Québec, Canada, which it purchased on May 1, 2000.

- BWR, through CanZinco Ltd. ("CanZinco"), owns and operates the Nanisivik zinc/silver mine located on Strathcona Sound, Baffin Island, Nunavut, Canada, which it purchased in July, 1996.

- BWR, through Breakwater Tunisia S.A., owns and operates the Bougrine zinc/lead mine which it purchased in September, 1997. The mine commenced production in May 1998 and is located in Tunisia.

- BWR, through American Pacific Honduras S.A. de C.V., owns and operates the El Mochito zinc/lead/silver mine located in Honduras that, except for a short period of time in 1987, has been in production since 1948. BWR acquired the mine in March, 1990 by amalgamating with American Pacific Honduras S.A. de C.V.

- BWR, through Sociedad Contractual Minera El Toqui, owns and operates the El Toqui zinc/gold mine that, except for a brief time in 1998, has been in production since 1987, and is located in Chile. BWR purchased the mine in August, 1997.

- BWR owns the Langlois zinc/copper/gold/silver mine located near Lebel-Sur-Quévillon, Québec, Canada. BWR purchased the mine on May 1, 2000. The Langlois Mine is temporarily on care and maintenance pending higher metal prices.

- BWR, through CanZinco, also owns the Caribou zinc/lead/silver mine located in New Brunswick, Canada, which was placed on care and maintenance in August, 1998 for an indefinite period of time. BWR acquired the Caribou underground mine and processing plant in the fall of 1990, and acquired the Caribou open pit mine in October, 1995.

2.4 Metal Production

Table 2.2 sets forth BWR's production statistics for the periods ended December 31, except for 2001 which is to 30 November.

Table 2.2: BWR Metal Production

	2001YTD 30 Nov	2000**	1999	1998
Milled (tonnes)	**3,029,161**	3,028,922	2,236,626	1,895,577
Zinc (%)	**7.1**	7.7	8.6	7.8
Concentrate Production				
Zinc (tonnes)	**359,155**	392,655	330,971	253,644
Lead (tonnes)	**21,023**	16,755	17,386	12,198
Copper (tonnes)	**40,095**	26,351	-	-
Gold (tonnes)	**3,107**	3,010	2,992	1,352
Metal in Concentrates				
Zinc (tonnes)	**192,299**	208,996	173,733	133,734
Lead (tonnes)	**12,236**	11,021	11,027	7,798
Copper (tonnes)	**6,597**	4,532	-	-
Silver (ounces)	**2,684,555**	2,790,137	2,155,781	1,930,531
Gold (ounces)	**36,628**	20,289	4,938	2,308
Minesite Operating Costs				
per tonne milled (US$)*	**27.56**	27.63	27.81	32.13
Total Cash Costs				
per lb. payable zinc (US$)*	**0.36**	0.40	0.40	0.43

*2001 costs are first nine months results
**Reflects additional production from Bouchard-Hébert and Langlois mines, May to December only.

The concentrates produced by BWR are sold directly to smelters located in Europe, Canada, Asia and South America, and to trading companies who resell the concentrates to smelters located in various parts of the world. The majority of BWR's concentrates are sold under long term supply contracts, with the balance of the concentrates sold by tender and/or spot sales.

2.5 Property Descriptions

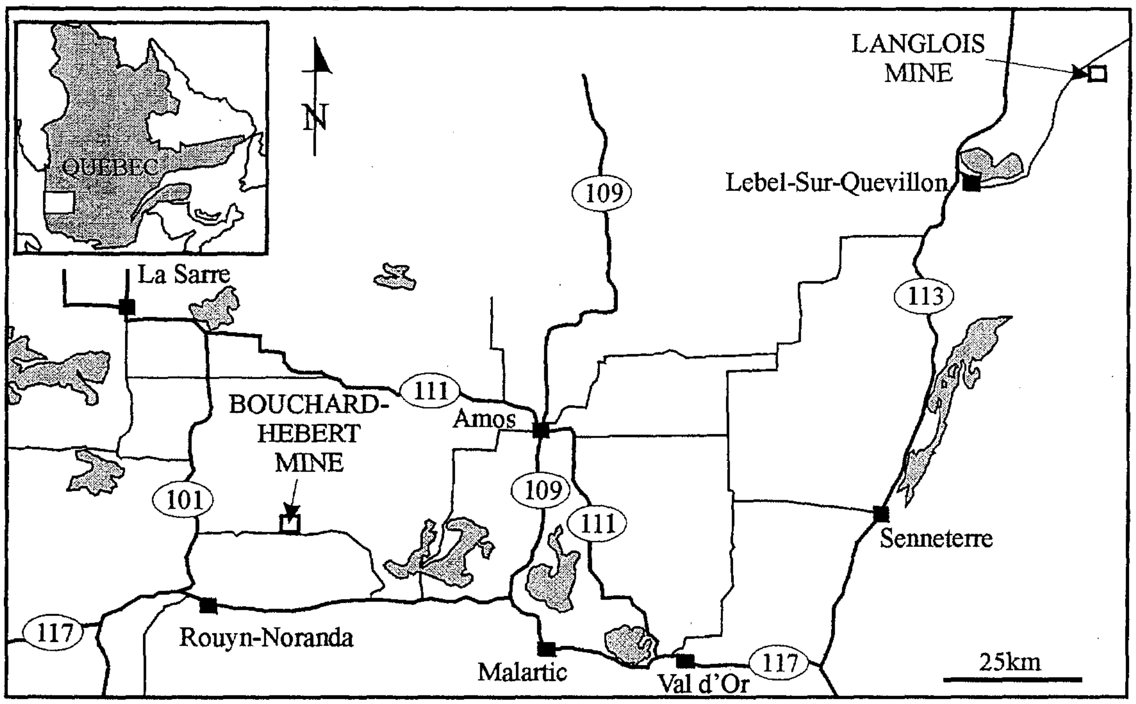
2.5.1 Bouchard-Hébert Mine

Effective May 1, 2000, BWR purchased the Bouchard-Hébert and Langlois zinc/copper/gold/silver mines from Cambior Inc. "(Cambior") for U.S.$40.3 million (Cdn.$63.5 million). The purchase price allocated to the Bouchard-Hébert Mine is U.S.$20.5 million (Cdn.$32.2 million which included approximately U.S.$2.9 million (Cdn.$4.4 million) of working capital. BWR owns 100% of the Bouchard-Hébert Mine, which is subject to a minor net operating profit royalty.

2.5.1.1 Location and Access

The Bouchard-Hébert Mine is an underground mine located 30 kilometres northeast of Rouyn-Noranda in Dufresnoy Township, Québec. A location map is presented in Figure 2.1. The property is accessible by a gravel road that links the mine with regional highway number 101.

Figure 2.1: Bouchard-Hébert Mine, Location Map



2.5.1.2 Description

The immediate mine property covers 107 hectares including two mining leases, one granted until 2007 and the other until 2015. In addition, surrounding the mining leases are 223 claims covering 8,269 hectares. The leases, upon expiry, may be renewed by formal application to the applicable governmental authorities. A map of the claims/lease boundaries is presented in Figure 2.2. A complete list of the claims and leases is presented in Appendix 1. SRK reviewed relevant project data and limited correspondence from the appropriate regulatory agencies to determine the validity and ownership of the mining asset. However, SRK did not conduct an in-depth review of mineral title and ownership and therefore has not provided an opinion on this matter.

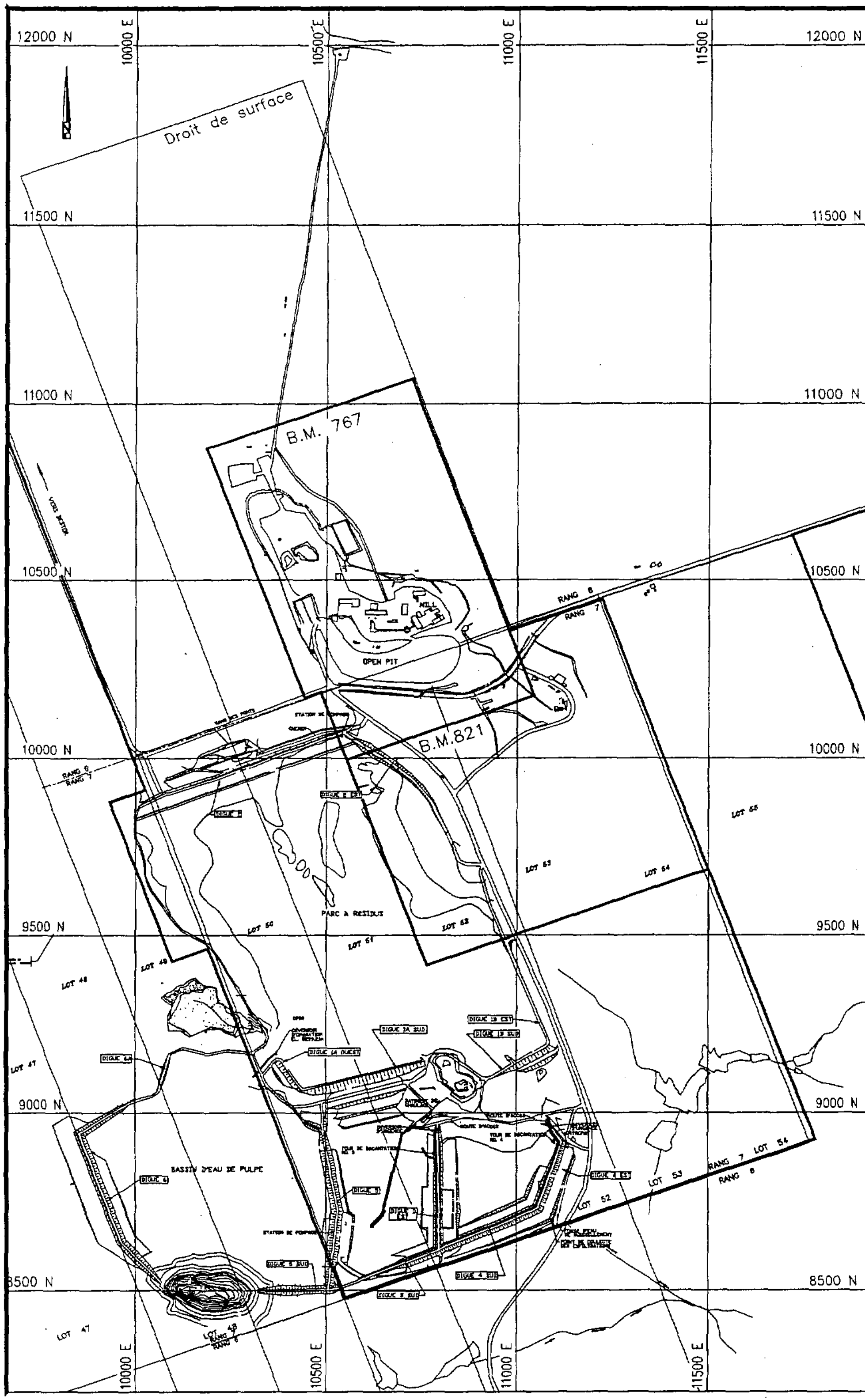


Figure 2.2: Bouchard-Hébert Mine, Claim/Lease Boundaries

The Bouchard-Hébert Mine facilities include an underground mine, a conventional mill and concentrator, which produces separate copper and zinc concentrates. A mine site plan is presented in Figure 2.3. Other facilities include a headframe and hoist building, which houses the compressors that provide air for underground activities and ancillary functions, a paste backfill plant and concentrate loadout facility. Hydro Québec provides electrical power.

Access to the Bouchard-Hébert Mine is by a 747 metre deep, three-compartment production/service shaft.

2.5.1.3 History

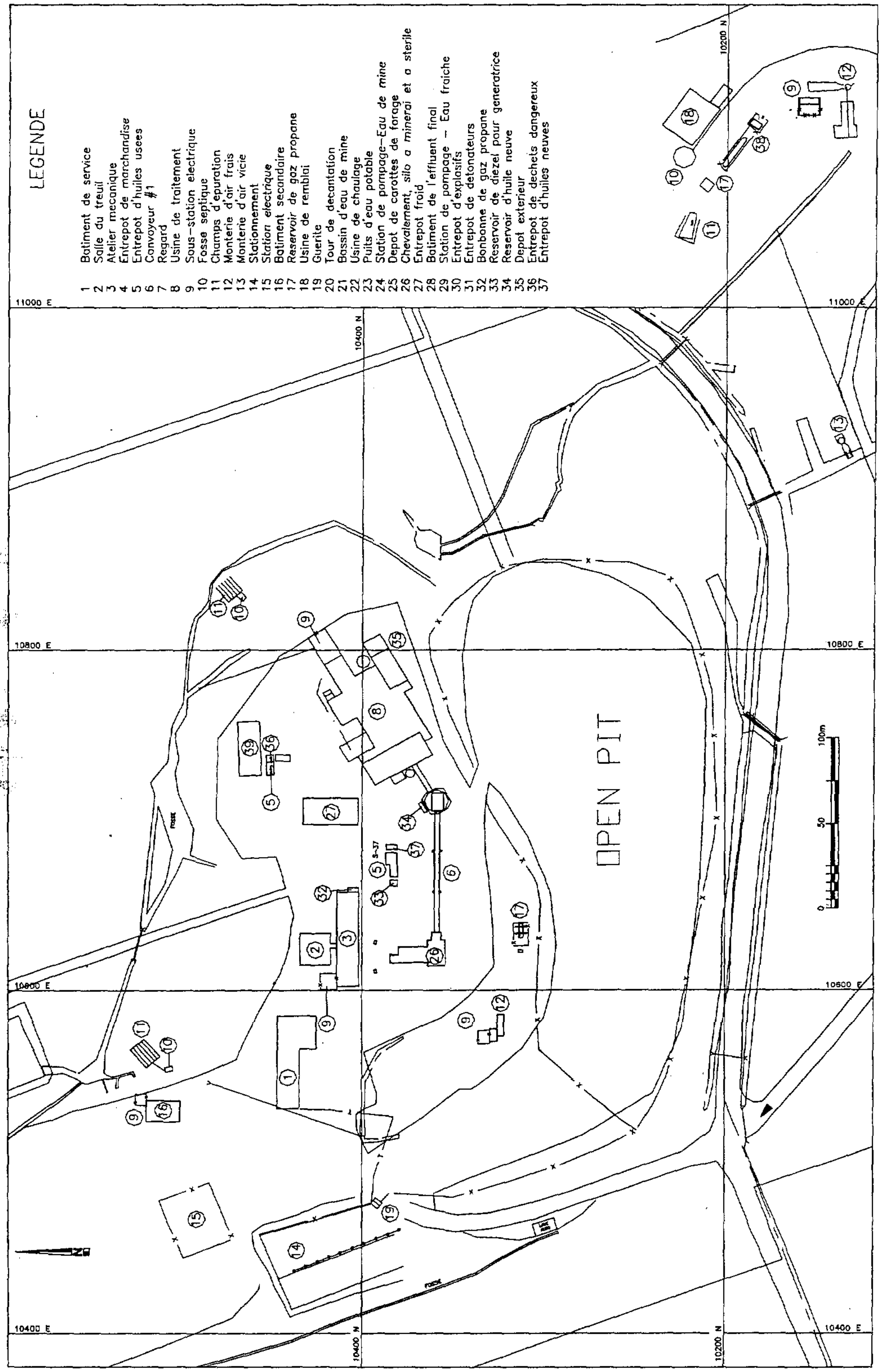
In 1955, Rio Algom conducted initial exploration on the Bouchard-Hébert property, which led to the discovery of the Main Lens of the Mobrun deposit (previously known as the Mobrun Mine, until Cambior renamed it the Bouchard-Hébert Mine in 1995). In 1984, Falconbridge Copper Corporation ("Falconbridge Copper") acquired the property subject to a net profit interest retained by Mobrun Copper, a wholly-owned subsidiary of Rio Algom and subsequently signed an option agreement with Audrey Resources Inc. ("Audrey Resources") in 1985. The option agreement allowed Audrey Resources the right to acquire a 70% interest in the property and become the project operator.

Open pit mining began in 1987 at a rate of 1,000 tonnes per day and the material was treated at Falconbridge Copper's Norbec mill. In 1987, Falconbridge Copper was restructured and renamed Minnova Inc. In 1988, the 1100 Lens was discovered at depth and 250 metres southeast of the Main Lens. As a result of the new discovery, Audrey Resources constructed an 1,100 tonne per day mill at the mine site. Mining operations were interrupted in 1992 due to reserve depletion. In total, over 1.5 million tonnes grading 2.42% zinc, 0.85% copper, 27.1 grams of silver per tonne and 2.4 grams of gold per tonne were mined from the Main Lens.

In September 1992, Cambior acquired 65% of Audrey Resources. In November 1992, an exploration and delineation program was initiated to delineate reserves for the 1100 Lens. In 1993, Metall Mining Corporation ("Metall") acquired Minnova and in 1994, Metall converted its remaining interest in the property to a 4% NSR, leaving Audrey Resources as the sole owner of the property. Metall was renamed Inmet Mining Corporation ("Inmet") in 1995. Cambior later purchased the 4% NSR from Inmet and commercial production from the 1100 Lens started in January 1995.

In June 1995, Cambior acquired the remaining 35% of Audrey Resources and the Mobrun Mine was renamed the Bouchard-Hébert Mine. Until December 31, 1998, the mine was 100% owned by Audrey Resources, a wholly-owned subsidiary of Cambior. In January 1999, Audrey Resources was liquidated into Cambior, and later dissolved.

Figure 2.3: Bouchard-Hébert Mine, Site Plan



BWR signed an agreement to purchase the Bouchard-Hébert and Langlois mines from Cambior on March 14, 2000. On March 31, 2000 Cambior encountered unforeseen mechanical problems with the semi-autogenous grinding ("SAG") mill at the Bouchard-Hébert Mine which resulted in a temporary shut down of the mill.

The required corrective action was determined and the repair work was initiated immediately. In the interim period a temporary crushing plant was installed in order to maintain production while the SAG mill was being repaired. The temporary system allowed for milling operations at approximately 68% of the normal capacity during May and June 2000 and 80% of capacity until the repaired SAG mill was put back into service on October 5, 2000. The lost production and property damage were covered by insurance, and BWR has received its portion of the claim.

2.5.2 Nanisivik Mine

In July, 1996, BWR acquired Nanisivik Mines Ltd. from AEC West Limited for $44.2 million. The purchase price included working capital of $39.4 million, comprised principally of zinc concentrate. In 1997, the mine was sold to CanZinco, a wholly-owned subsidiary of BWR.

2.5.2.1 Location and Access

The Nanisivik Mine is an underground mine located in Nunavut on northern Baffin Island on the southern shore of Strathcona Sound, approximately 750 kilometres north of the Arctic Circle. A location map is presented in Figure 2.4. Access to the property is by air or by ship. The Nanisivik/Arctic Bay airport is serviced by commercial carriers, which provide regular flights to Ottawa, Ontario. Heavier equipment and nonperishable foodstuffs are brought in by ship. Road access exists to the Inuit settlement of Arctic Bay 40 kilometres to the west. A mine site plan is presented in Figure 2.5.

Figure 2.4: Nanisivik Mine, Location Map

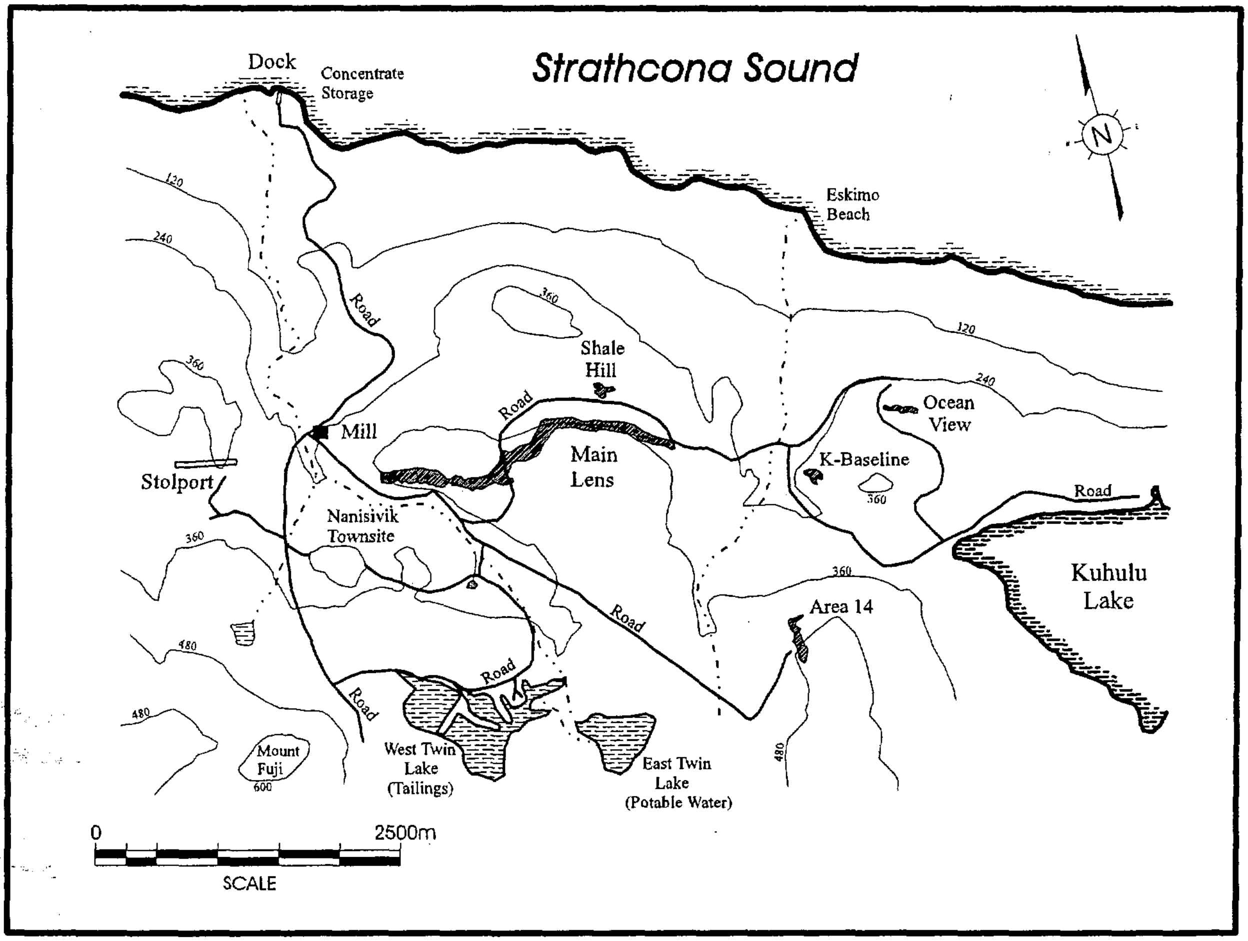


Figure 2.5: Nanisivik Mine, Site Plan

The climate at the mine is semi-arid arctic. Precipitation is low with a total of approximately 125 millimetres per year. The normal maximum temperature in the summer is 15°C and winter temperatures typically range from -20°C to -40°C. Shipping is possible from June to October when concentrate carriers with ice-breaking capabilities are used. The mine is different from many others in the world because of the presence of significant permafrost to below any potential mining depth.

2.5.2.2 Description

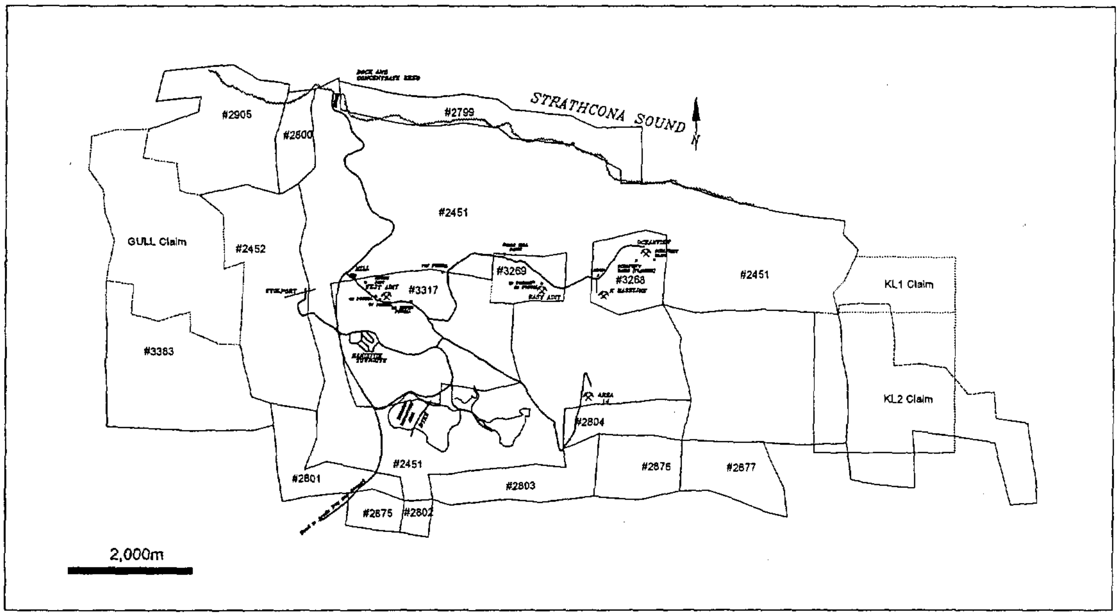
The Nanisivik Mine consists of an underground mine, a 2,200 tonne per day concentrator, a power plant, a townsite, recreational facilities and a concentrate storage facility and ship loader. The mine operates an 11.2 megawatt diesel-electric generating plant to meet all power requirements. The power plant exhaust gas is used to dry concentrates and, via heat exchangers, recovers heat for buildings and domestic hot water. The mine townsite facilities consist of a cafeteria, offices, school and daycare, recreation centre, swimming pool, gymnasium, ice hockey arena, Royal Canadian Mounted Police station, fire hall, nursing station, store, library and church. The Nanisivik Mine has accommodation ranging from bunkhouses to four bedroom detached houses. The harbour and port facilities are owned by the Federal Government of Canada and administered by the Canadian Coast Guard.

Mineral title to the Nanisivik Mine is held under mineral leases. Mineral leases are issued for 21-year periods with rights of renewal. Annual rentals are $1.00 per acre for the original period and $2.00 per acre for renewals. The leases have historically been renewed as required with the next renewal date being in 2009. In addition to the mineral title requirements, surface title is also required for certain operations. Essentially all of the surface title in the Nanisivik Mine area is controlled by the Federal Government of Canada. However, at mine start-up, the surface rights to one block of ground, called the Block Transfer, were transferred to the Nunavut Government (formerly the Government of the Northwest Territories). Within the Block Transfer, the mine negotiates land matters with the Nunavut Government, while outside that area, such negotiations are with the Federal Government of Canada.

The operation of the Nanisivik Mine is governed by an agreement signed June 18, 1974 (the "Master Agreement") between Nanisivik Mines Ltd. (as assignee of Mineral Resources International Limited ("MRI")) and the Department of Indian Affairs and Northern Development ("DIAND") which provided for the development and bringing into production of the Nanisivik Mine and, based upon the original mineral reserves and initial design capacity of the mine and concentrator, contemplated a mine life of 12 years. Nanisivik has been in compliance with the Master Agreement governing operations at the Nanisivik Mine except with respect to the goal for a specified percentage of the workforce that is to consist of northern residents and the requirement regarding the maximum amount of mineralized material which may be mined annually. BWR was advised by the former owner in connection with the acquisition of the Nanisivik Mine that these areas of non-compliance have continued for a number of years and are known to the responsible officials of the Federal Government of Canada. BWR believes that such non-compliance and future non-compliance will not have a material adverse impact on BWR's operations or financial condition.

A list of the claims and leases pertaining to the Nanisivik Mine is presented in Appendix 1. A map of the claim/lease boundaries is presented in Figure 2.6. SRK reviewed relevant project data and limited correspondence from the appropriate regulatory agencies to determine the validity and ownership of the Material Property. However, SRK did not conduct an in-depth review of mineral title and ownership and therefore has not provided an opinion on this matter.

Figure 2.6: Nanisivik Mine, Claim/Lease Boundaries



2.5.2.3 History

The area around the Nanisivik Mine was mapped for the Geological Survey of Canada ("GSC") in 1954 and the occurrence of galena and sphalerite in pyrite zones was noted. Based on the work of GSC, Texasgulf Inc. ("Texasgulf") commenced work in 1957 and staked much of the present mine property that year. Texasgulf carried out a significant amount of diamond drilling over the next ten years, but the work slowed down in the mid-1960s because of ongoing work on the Kidd Creek deposit. In 1969, Texasgulf refocused its efforts at Nanisivik, with the driving of a 650 metre adit into the east end of the main mineralized body. A 50-ton sample was shipped to Timmins, Ontario for metallurgical testing in 1970. In 1972, MRI approached Texasgulf and an agreement was concluded pursuant to which ownership of the Nanisivik deposit passed to MRI. MRI completed a feasibility study on the project that yielded positive economic projections. Based on this study, Nanisivik Mines Ltd. ("Nanisivik") was formed and ultimately became a wholly-owned subsidiary of AEC West Limited ("AEC"), formerly Conwest Exploration Company Limited.

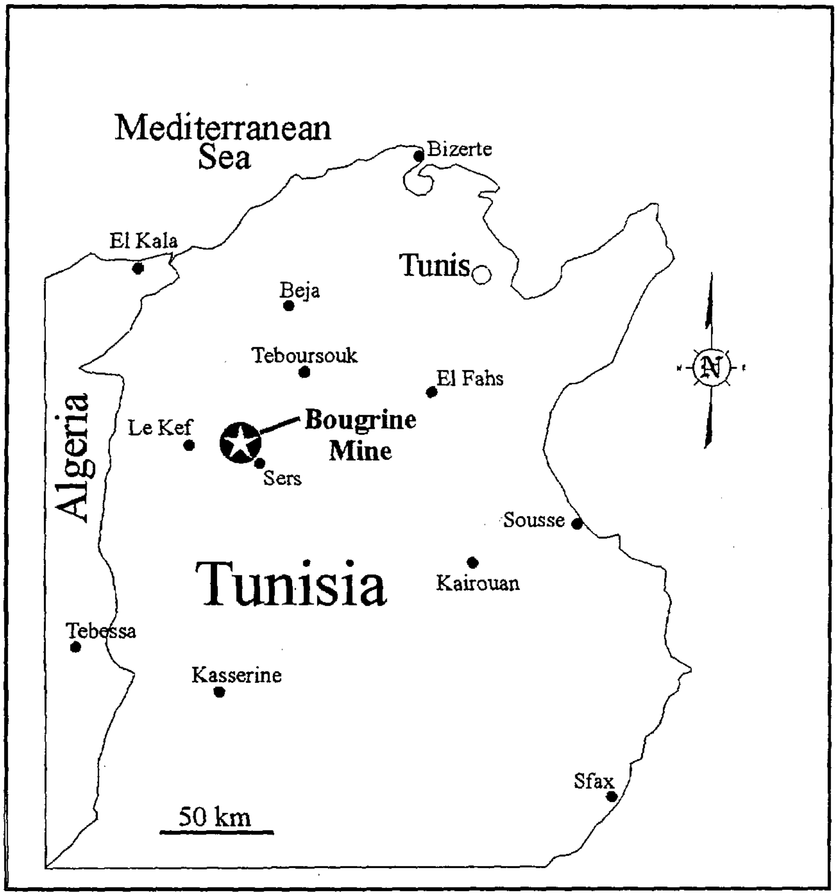
2.5.3 Bougrine Mine

In September 1997, BWR, through a wholly-owned subsidiary, Breakwater Tunisia S.A. ("BWR Tunisia"), acquired the Bougrine underground mine and related assets for $26.8 million. BWR spent $10.7 million in pre-production capital costs in 1997 and 1998 to ready the mine for production. In the mine, a new water pumping system and a new ventilation system were installed and the underground mobile mining equipment was rebuilt. In the mill, improvements were made to the grinding circuit, the material delivery system and other facilities. Operations commenced on May 2, 1998 and the site was considered to be in commercial production June 1, 1998.

2.5.3.1 Location and Access

The Bougrine Mine is located between the towns of Le Sers and Le Kef in Tunisia, 160 kilometres southwest of Tunis, the capital. Access is by paved road. A location map is presented in Figure 2.7.

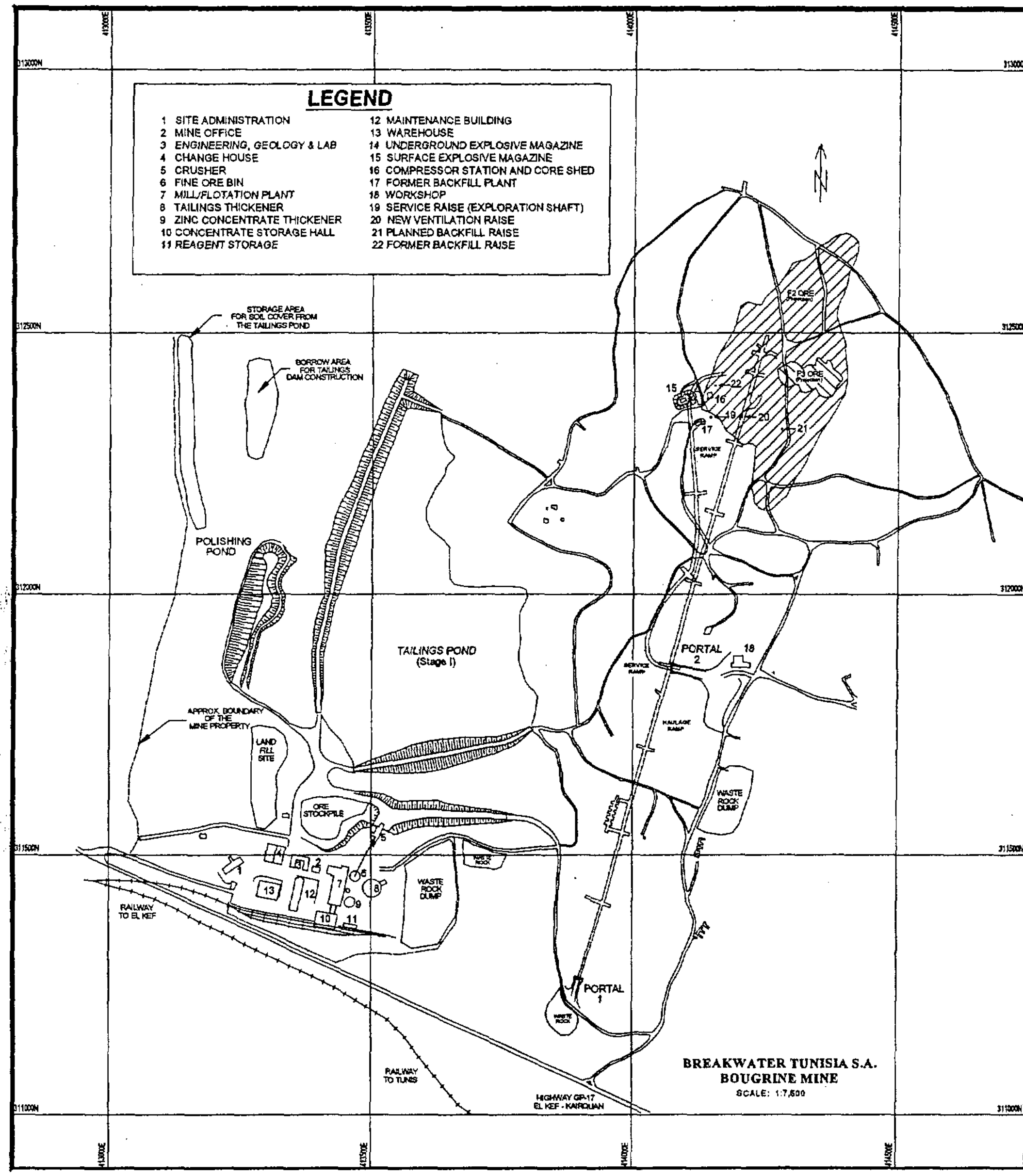
Figure 2.7: Bougrine Mine, Location Map



2.5.3.2 Description

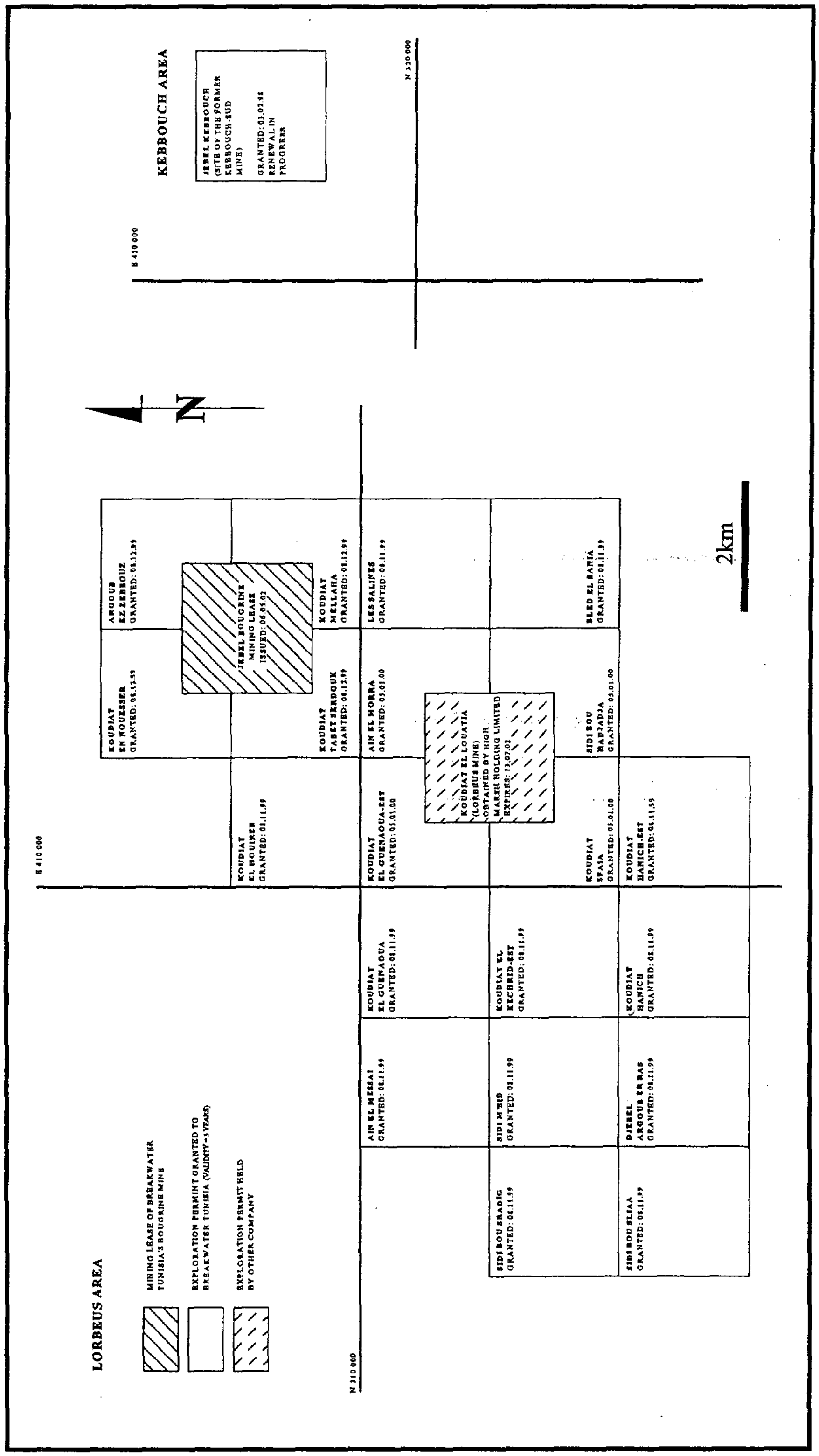
The Bougrine Mine is a zinc/lead underground mine. As well as a 1,200 tonne per day concentrator, the site facilities include a security/mine rescue/ambulance station, a warehouse, dry facilities, offices, repair and maintenance shops and a concentrate load out facility. A mine site plan is presented in Figure 2.8.

Figure 2.8: Bougrine Mine, Site Plan



The concentrate loading facilities at the port of La Goulette in Tunis include a concentrate storage building with a capacity for 12,000 tonnes of zinc concentrate and 3,000 tonnes of lead concentrate. Concentrates are loaded onto ships by conveyor and ship loader. The port land is owned by the government and has been leased by BWR for a 20-year period expiring in 2012. The leases, upon expiry, may be renewed by formal application to the applicable governmental authorities. BWR Tunisia holds a 50-year concession over the Bougrine mineralized body, which concession expires in March 2042. A plan of the mining lease and exploration permit boundaries is presented in Figure 2.9. Information on the mining lease and exploration permits is presented in Appendix 1.

Figure 2.9: Bougrine Mine, Mining Lease and Exploration Permit Boundaries



2.5.3.3 History

Zinc and lead deposits have been mined in the region since Roman times. Some small-scale development took place at Bougrine in the 1930's. A drilling program in the 1980's by ONM (an exploration division of the Tunisian government) resulted in the discovery of the F2 and F3 zones at depth. In the late 1980's a competition was held to find a foreign partner to develop a mine. Metalgesellschaft AG was awarded a license to develop the mine at Bougrine. Metalgesellschaft AG transferred its interest in the property to Metall Mining Corporation (subsequently renamed Inmet Mining Corporation) who, with a group of Tunisian banks, established Société Minière de Bougrine ("SMB"). Construction of the Bougrine facility began in 1992 with production commencing in June 1994. The total capital cost of the project was U.S.$81 million. Operations were suspended in October 1996 due to low zinc prices and financial constraints. During the operating period of June 1994 to October 1996, the mill processed 675,886 tonnes averaging 14.2% zinc and 2.5% lead.

2.5.4 El Mochito Mine

In March 1990, BWR acquired American Pacific Mining Corporation, Inc. ("AMPAC") by way of an amalgamation of AMPAC with a wholly-owned subsidiary of BWR now named Santa Barbara Mining Company, Inc. In 1998 the El Mochito Mine was transferred to American Pacific Honduras S.A. de C.V., a wholly-owned subsidiary. Except for a short period in 1987, the mine has been in continuous production since 1948.

2.5.4.1 Location and Access

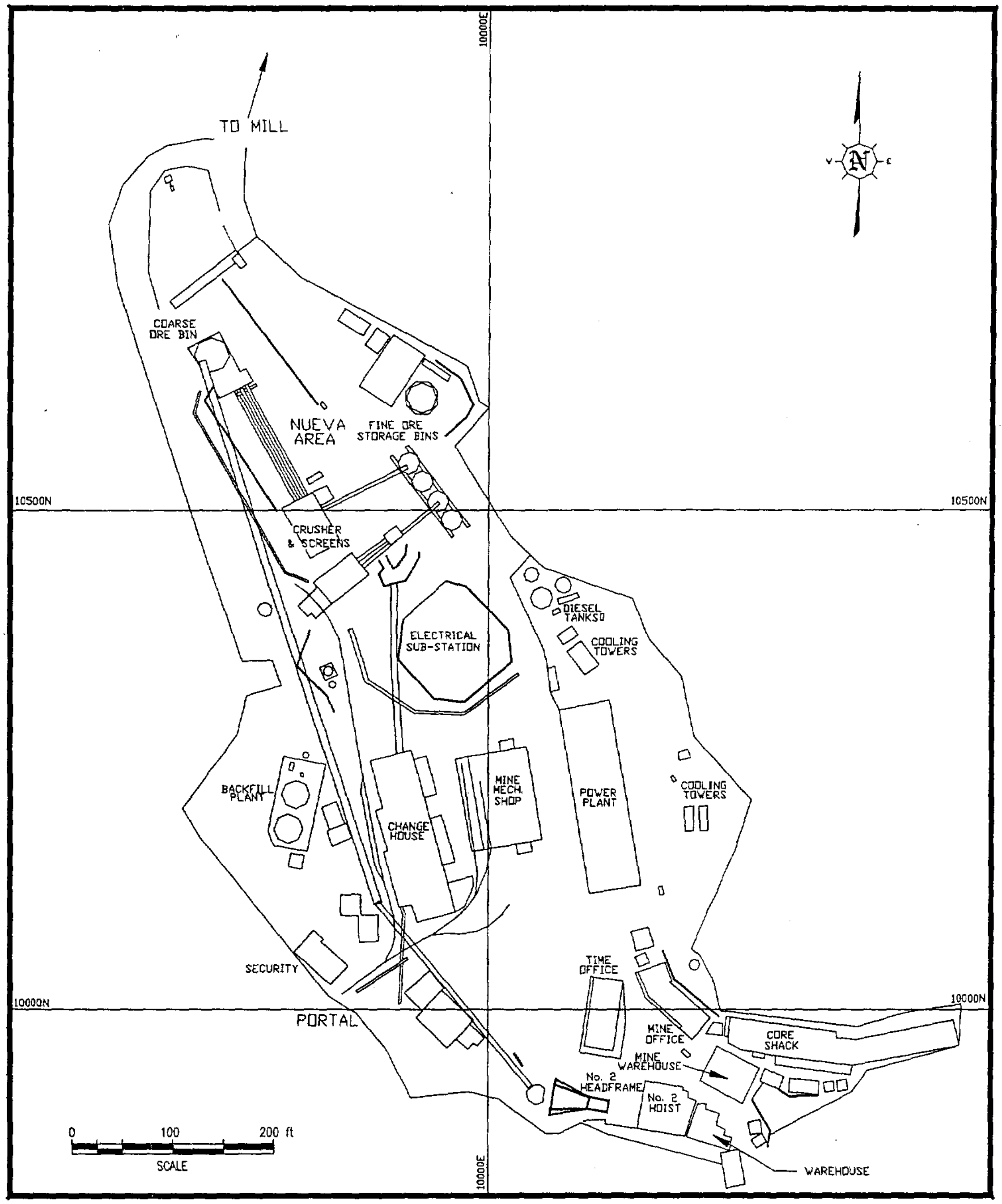
The El Mochito underground mine is located in northwest Honduras, near the town of Las Vegas. The closest major city is San Pedro Sula, the commercial centre of the country, 88 kilometres northeast of the mine. The concentrates are trucked to BWR's concentrate/storage shed on tidewater at Puerto Cortés approximately 115 kilometres northeast of the mine. Access is by paved roads. A location map is presented in Figure 2.10.

Figure 2.10: El Mochito Mine, Location Map

2.5.4.2 Description

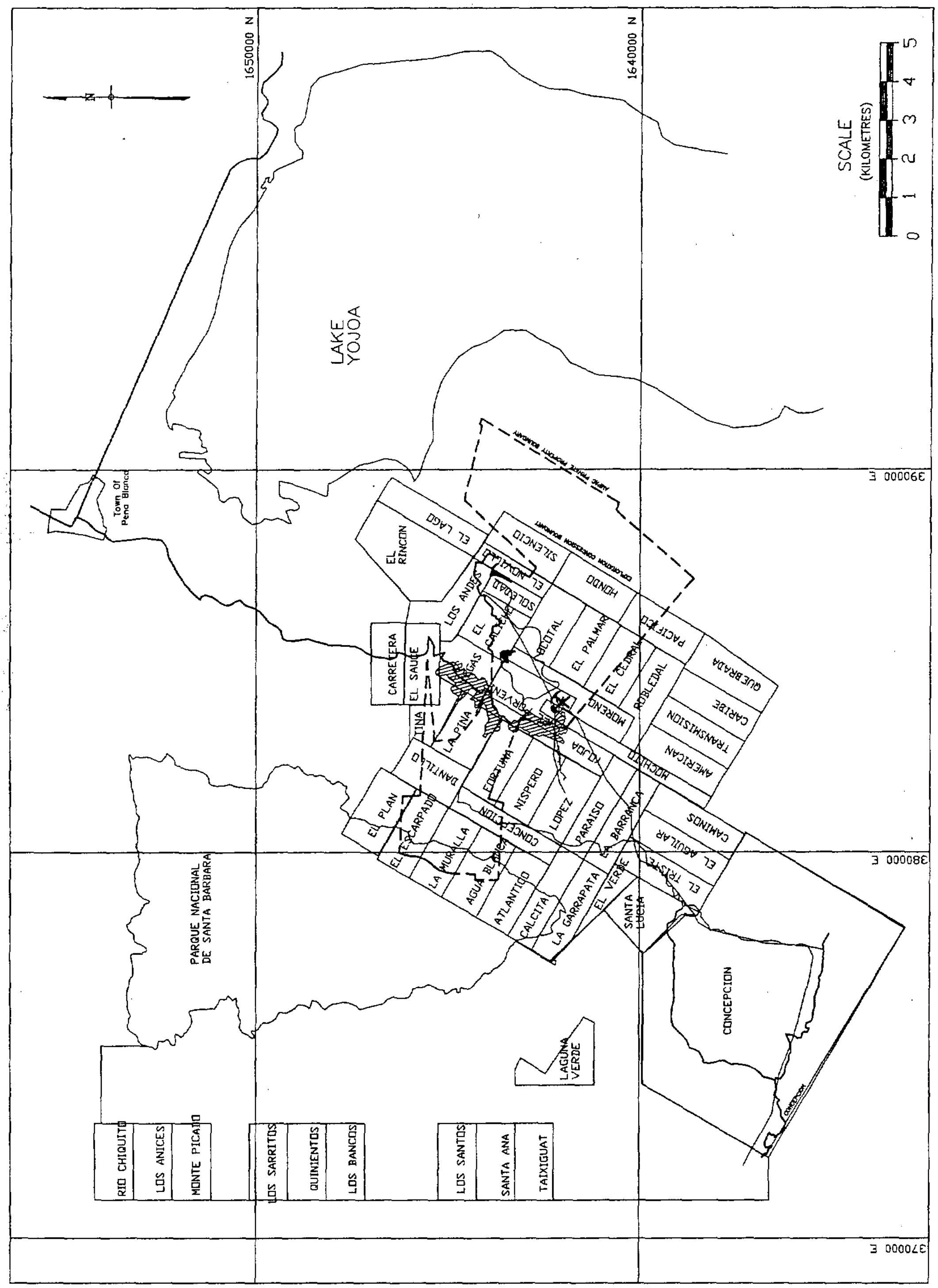
The El Mochito Mine is comprised of an underground zinc/lead/silver mine and a 2,000 tonne per day concentrator. A mine site plan is presented in Figure 2.11.

Figure 2.11: El Mochito Mine, Site Plan



The El Mochito property includes 53 exploitation concessions totalling 10,835 hectares of mineral rights. A map of the concession and mineral rights boundaries is presented in Figure 2.12. Information on the concessions and mineral rights is presented in Appendix 1.

Figure 2.12: El Mochito Mine, Concession and Mineral Rights Boundaries



2.5.4.3 History

The El Mochito Mine was originally discovered in 1938. In 1943, New York and Honduras Mining Company (now Rosario Resources Corporation ("Rosario")) purchased the property. Development work was accelerated after the war and in 1946 construction commenced on a small mill. Production began in 1948 with the initial zinc product being jig concentrates containing native silver, a bulk flotation concentrate and a silver product. Separation of the zinc/lead concentrates was uneconomic until 1960 when the amount of sulfide material produced from the deeper levels of the mine were increased. The mill has been expanded several times and presently has a capacity of 2,000 tonnes per day. In 1978, Amax, Inc. acquired Rosario. In April 1987, the El Mochito Mine was closed because of high taxes, labour problems and high operating costs.

In September 1987, AMPAC purchased the El Mochito Mine, a concentrate warehouse located at Puerto Cortés and the San Juancito property. The mine was re-opened in October 1987. In March 1990, BWR acquired AMPAC by way of an amalgamation of AMPAC with a wholly-owned subsidiary of BWR now named Santa Barbara Mining Company, Inc. In 1998 the mine was sold to American Pacific Honduras S.A. de C.V.

2.5.5 El Toqui Mine

In August 1997, BWR purchased all of the outstanding common shares of Sociedad Contractual Minera El Toqui ("El Toqui"), which owns the El Toqui underground mine in Chile, for $18.7 million, including $7.8 million for working capital, plus a net smelter return royalty. The royalty is 1% when the London Metal Exchange (the "LME") price of zinc exceeds U.S.$0.50 per pound and increases to a maximum of 3% when the LME price of zinc is equal to or greater then U.S.$0.60 per pound. The El Toqui Mine has been in production since 1987 at a rate of approximately 400,000 tonnes per year.

2.5.5.1 Location and Access

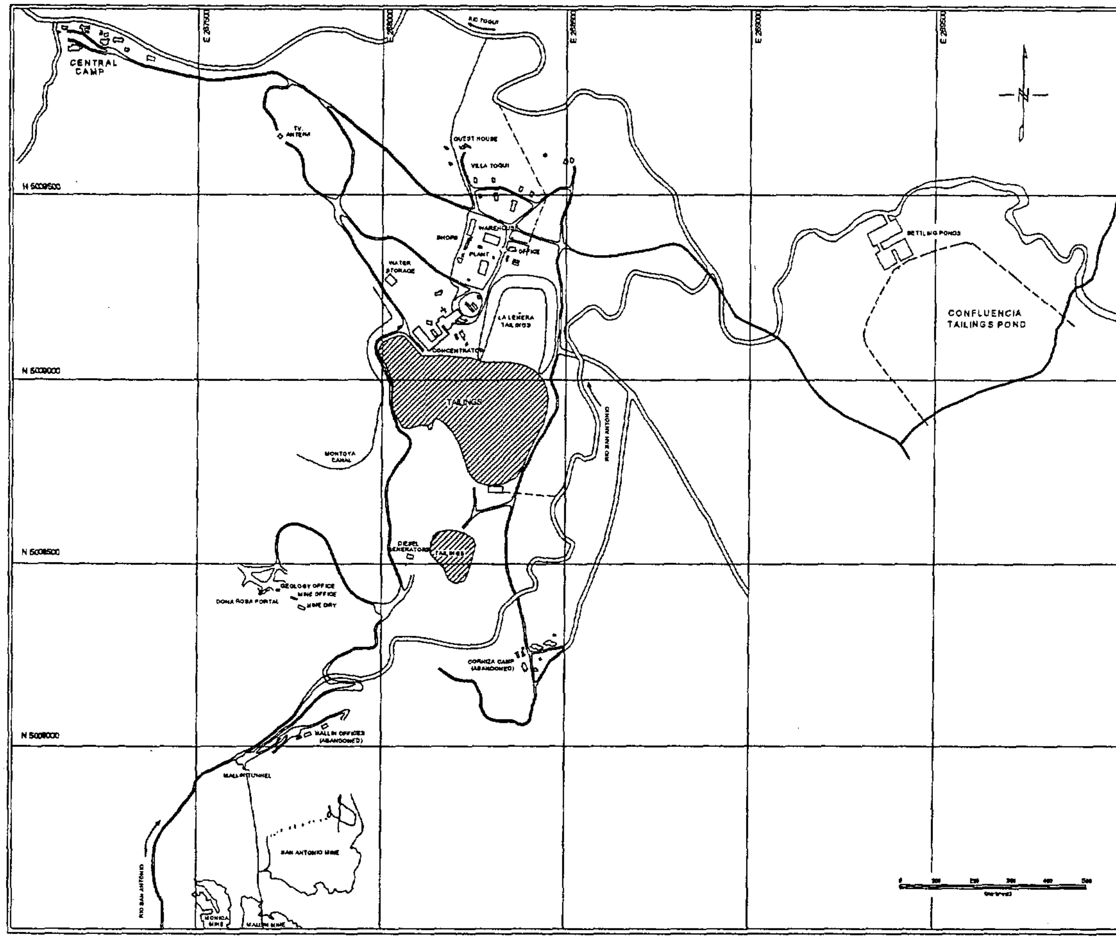
The El Toqui Mine is located in Chile's Region XI, approximately 1,350 kilometres south of Santiago and 120 kilometres northeast of Coyhaique. A location map is presented in Figure 2.13.

Figure 2.13: El Toqui Mine, Location Map

2.5.5.2 Description

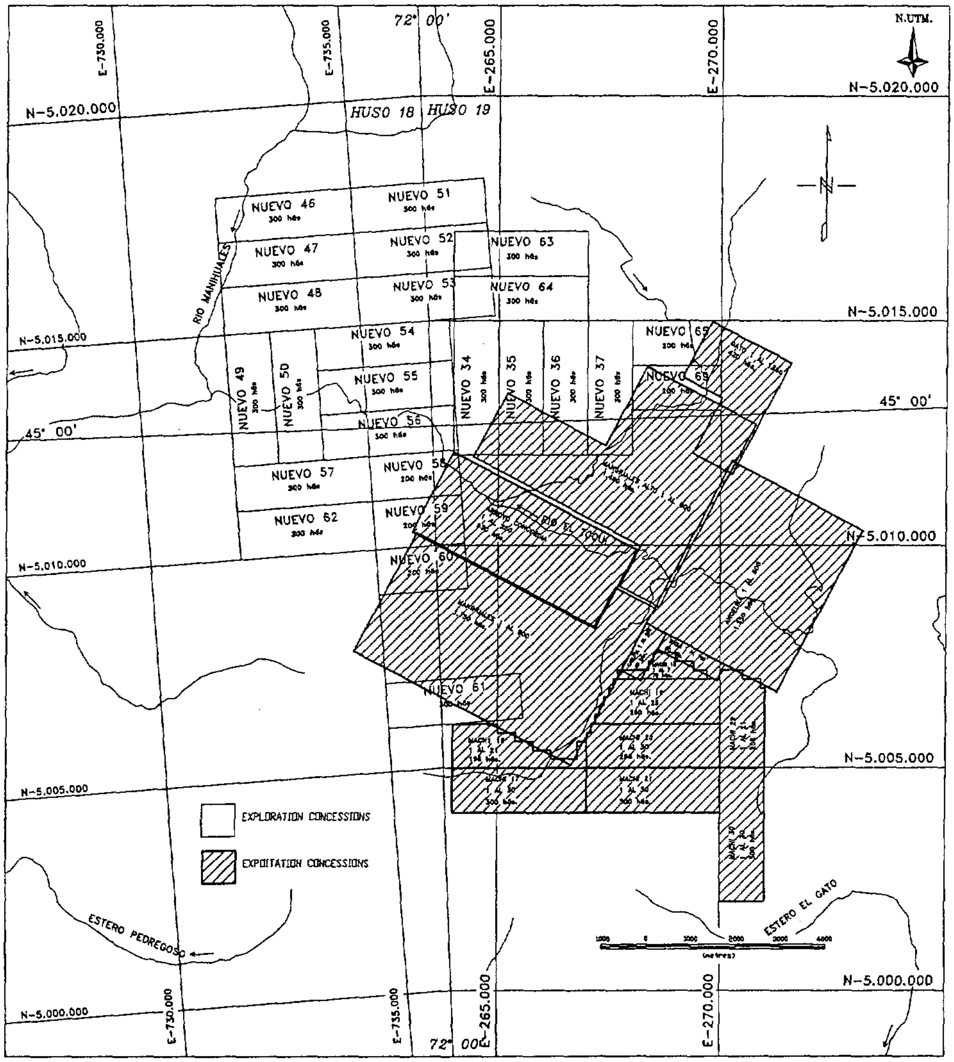
The El Toqui property covers approximately 1,200 square kilometres of mountainous terrain and includes the currently producing Doña Rosa zinc mine, the former producing San Antonio and Mallin-Monica zinc/lead mines and an 1,100 tonne per day concentrating plant. The El Toqui operation mines and processes material, which contains zinc, gold, silver, copper and lead. The concentrate produced by the plant is transported to a deep-water port for export. El Toqui owns and operates a dedicated hydroelectric power generating plant as well as a diesel generating plant. The property includes a concentrator, mine and maintenance offices, laboratory and service buildings and houses as well as a ship loader at Puerto Chacabucco. A mine site plan is presented in Figure 2.14.

Figure 2.14: El Toqui Mine, Site Plan



A plan of the El Toqui exploration and exploitation concessions is presented in Figure 2.15. Detailed information of the concessions is presented in Appendix 1.

Figure 2.15: El Toqui Mine, Exploration and Exploitation Concessions



2.5.5.3 History

The El Toqui deposit was discovered in the early 1970's when two small rich lead/silver veins, outcropping in a steep sided valley, were being worked. These high-grade veins were soon depleted and mining progressed to a series of manto-type zones. Metallgesellschaft AG owned the property for a number of years and then sold it to a Chilean entrepreneur. Lac Minerals subsequently acquired a majority interest in 1987, for U.S.$30 million, and in 1989 became the sole owner when it acquired the remaining 15% minority interest for U.S.$5.5 million. In 1994, Barrick Gold Corporation ("Barrick") acquired Lac Minerals which included El Toqui.

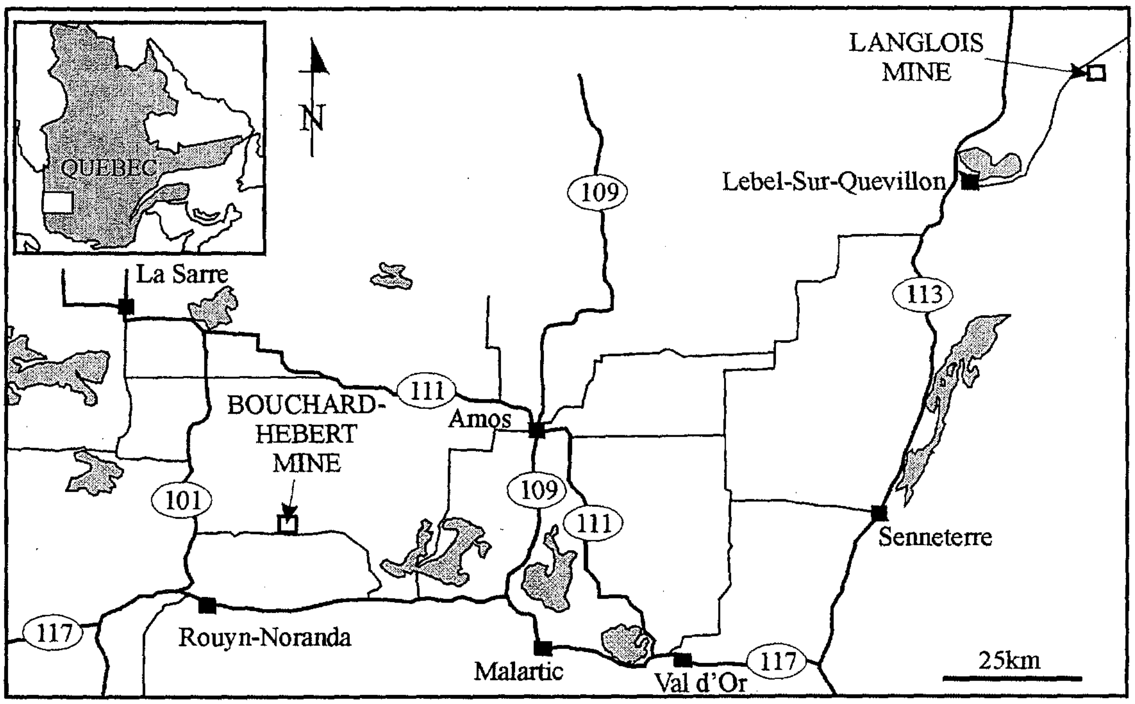
2.5.6 Langlois Mine

In May 2000, BWR purchased the Bouchard-Hébert and Langlois zinc/copper underground mines for U.S.$40.3 million (Cdn.$63.5 million). The purchase price allocated to the Langlois Mine was U.S.$19.8 million (Cdn.$31.2 million) which included approximately U.S.$1.1 million (Cdn.$1.6 million) of working capital.

2.5.6.1 Location and Access

The Langlois Mine is located in north-western Québec, approximately 48 kilometres northeast of the town of Lebel-sur-Quévillon and 213 kilometres north of Val d'Or. A location map is presented in Figure 2.16. Lebel-sur-Quévillon has a population of 3,500. The mine is accessed via a gravel road jointly maintained by BWR and a forest products company with operations in the area.

Figure 2.16: Langlois Mine, Location Map

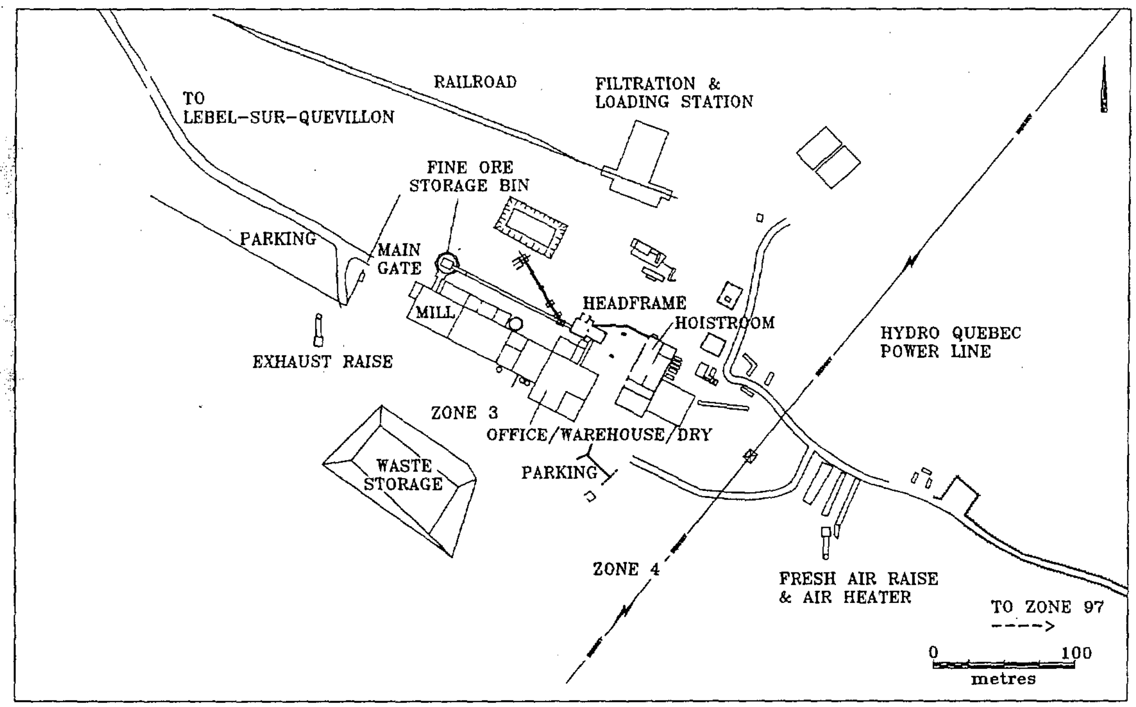


2.5.6.2 Description

The underground mine facilities include a headframe, a paste backfill plant, mechanical and electrical shops, a service building, a zinc/copper concentrator and a tailings pond. A mine site plan is presented in Figure 2.17. The mine produces zinc and copper concentrates, which are sold and forwarded to smelters for further processing. From such processing, gold and silver also result as by-products.

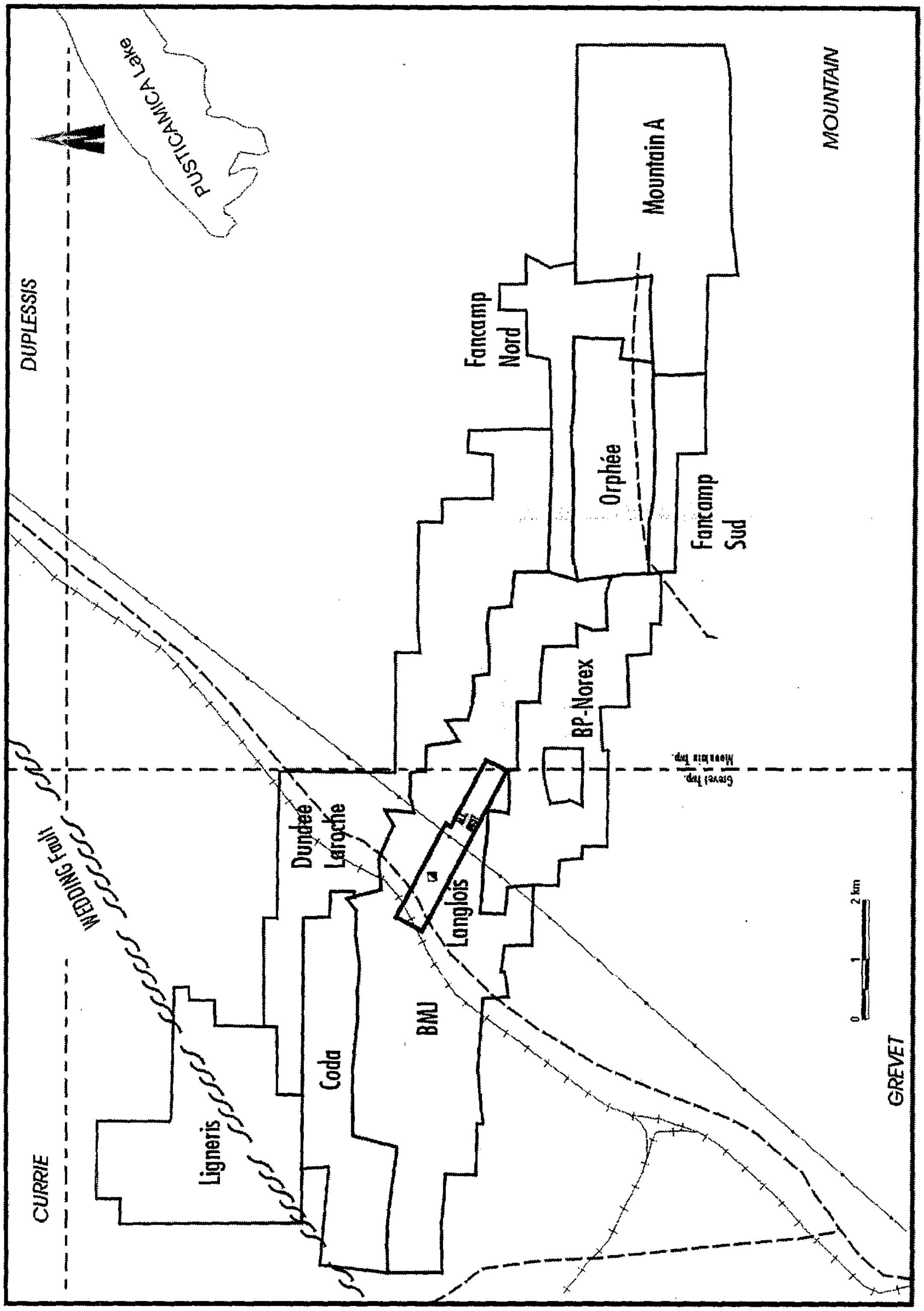
The mine is equipped with a 905 metre deep four-compartment shaft. There are two hoists; a 3.0 metre diameter double drum hoist for skipping and a 2.4 metre diameter double drum service hoist. Two seven tonne skips are used to hoist the blasted material to surface. At present, there is a crusher station on the bottom level of the mine. However, due to problems with severe ore-pass wall erosion and consequent dilution, a new loading facility was established on Level 11.

Figure 2.17: Langlois Mine, Site Plan



The property is held through a 133 hectare mining lease granted until the year 2015. In addition, there are 375 claims surrounding the lease covering 5,836 hectares in Grevet, Ruette and Mountain Townships. There are no royalties payable on mineral production from the Langlois Mine. The lease, upon expiry, may be renewed by formal application to the applicable governmental authorities. A map of the lease and claims is presented in Figure 2.18. Detailed lease and claim information is presented in Appendix 1.

Figure 2.18: Langlois Mine, Claim/Lease Boundaries



2.5.6.3 History

The deposit, originally known as the Grevet Project, was discovered in 1989 by Serem-Québec Inc. (50%) and VSM Exploration Inc. (50%). Cambior acquired its initial 50% interest in the Grevet Project in July 1992 with the acquisition of VSM. In September 1993, Cambior purchased the remaining 50% interest in the project from Serem to hold a 100% interest.

In 1994, Cambior commenced an underground exploration program designed to delineate mineable reserves. Due to the success of the underground exploration program, development work on the property commenced in the third quarter of 1994 and was completed in December 1995. Commercial production began at the mine in February 1996. Production at Langlois was halted in December 1996 due to high dilution problems in the mine. These problems were rectified by modifications to the mining method and production was resumed in July 1997.

The mine was purchased by BWR as of May 2000. During the months of June and July, the ore-pass system from the main production zone was out of service while a new system was being commissioned. This had a negative impact on mill throughput and grade. On November 28, 2000, BWR suspended operations at the Langlois Mine due to operating problems associated with the main ore-pass system and low zinc prices. The difficulties with the ore-pass system combined with a drop in metal prices and high treatment charges made it uneconomic to operate the mine until it is properly developed and operated at lower costs.

The temporary suspension of operations was expected to provide BWR with the time necessary to compile new geological data and design a long-term development and operating plan that would allow for production at lower operating costs. SRK was contracted to conduct a full feasibility study including the latest drill results of Zone 97 and a complete rework of the mine design and plan. The feasibility study was issued in August, 2001.

In the feasibility study, a decision was made to select a production rate of 450,000 tonnes/year and to increase the cutoff grade, creating a high-grade alternative with a mine life of eight years. The feasibility mine plan does not include mining of all of the mineable reserves, since it is a high-grade alternative. High-grade mining in the feasibility study does not isolate the lower grade reserves, which are currently excluded from the mining schedule. These reserves can be brought into production if metals prices increase sufficiently.

The feasibility operating plan incorporates several improvements to ensure reliability of production and to control costs, and provides a construction and development work schedule to prepare for recommencement of mining. The operating schedule for the underground mine will be five days per week, two 8-hour shifts per day.

At a zinc price of US$0.50/lb, the feasibility study indicates that the total net pre-tax cash flow is Cdn$60.9 million. The internal rate of return is 24.0% and the NPV at 10.0% is $20.9 million.

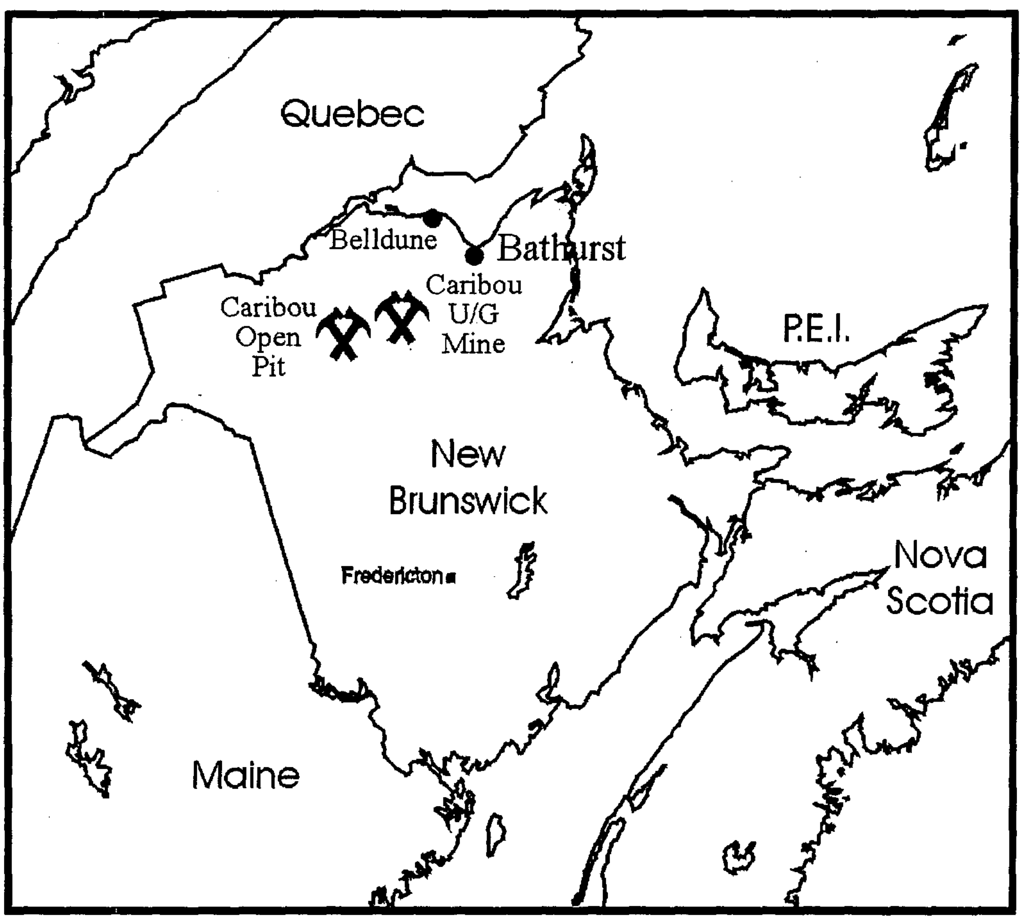
2.5.7 Caribou Mine

BWR acquired the Caribou Mine in the fall of 1990. The underground mine and concentrator operated for seven months in 1990. On October 26, 1990, operations were suspended due to poor recoveries and falling metal prices, and the facilities were placed on care and maintenance. The Caribou open pit mine was acquired in October 1995. It had not previously been in production. Ore from the Caribou underground mine and open pit mine was processed in the mill from July 1997 to August 1998, but the metallurgical recovery fell short of anticipated levels. As metal prices were declining, the operation was placed on care and maintenance again.

2.5.7.1 Location and Access

The Caribou Mine consists of an underground mine and mill located in Restigouche County in northeastern New Brunswick, 50 kilometres west of Bathurst, New Brunswick, and an open pit mine (formerly known as the Restigouche property) located approximately 80 kilometres west of Bathurst and 30 kilometres from the Caribou mill. Access is by way of Provincial Highway 180 to within four kilometres of the minesite and then by local mine road. Access to the open pit mine is also by way of Provincial Highway 180 and three kilometres by local mine road. A location map is presented in Figure 2.19.

Figure 2.19: Caribou Mine, Location Map



2.5.7.2 Description

The Caribou Mine property covers an area of 3,106 hectares and is held under a mining lease. A plan of the lease covering the underground mine and processing plant is presented in Figure 2.20. A plan of the lease covering the open pit mine is presented in Figure 2.21. Additional information on the Caribou Mine leases and claims is presented in Appendix 1. The Caribou underground mine production is subject to an NSR which escalates from 1% to 3% based on zinc prices ranging from U.S.$0.65 to U.S.$0.85 per pound and is capped at a maximum of $7.8 million. The Caribou underground mine production is also subject to a 10% net profit royalty. Net profit is calculated by deducting from revenue all operating, administration, depreciation, amortization and interest costs, including all costs incurred in prior years.

Figure 2.20: Caribou Mine, Mining Lease Map

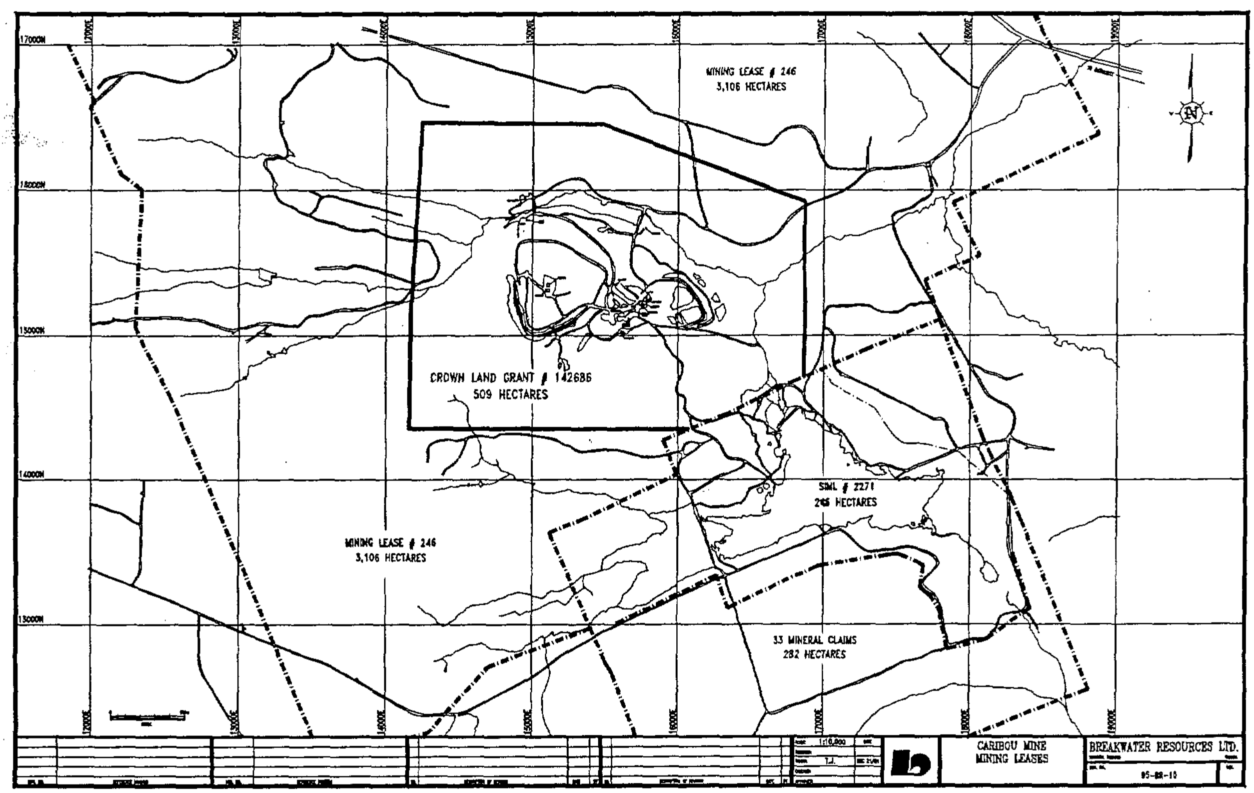
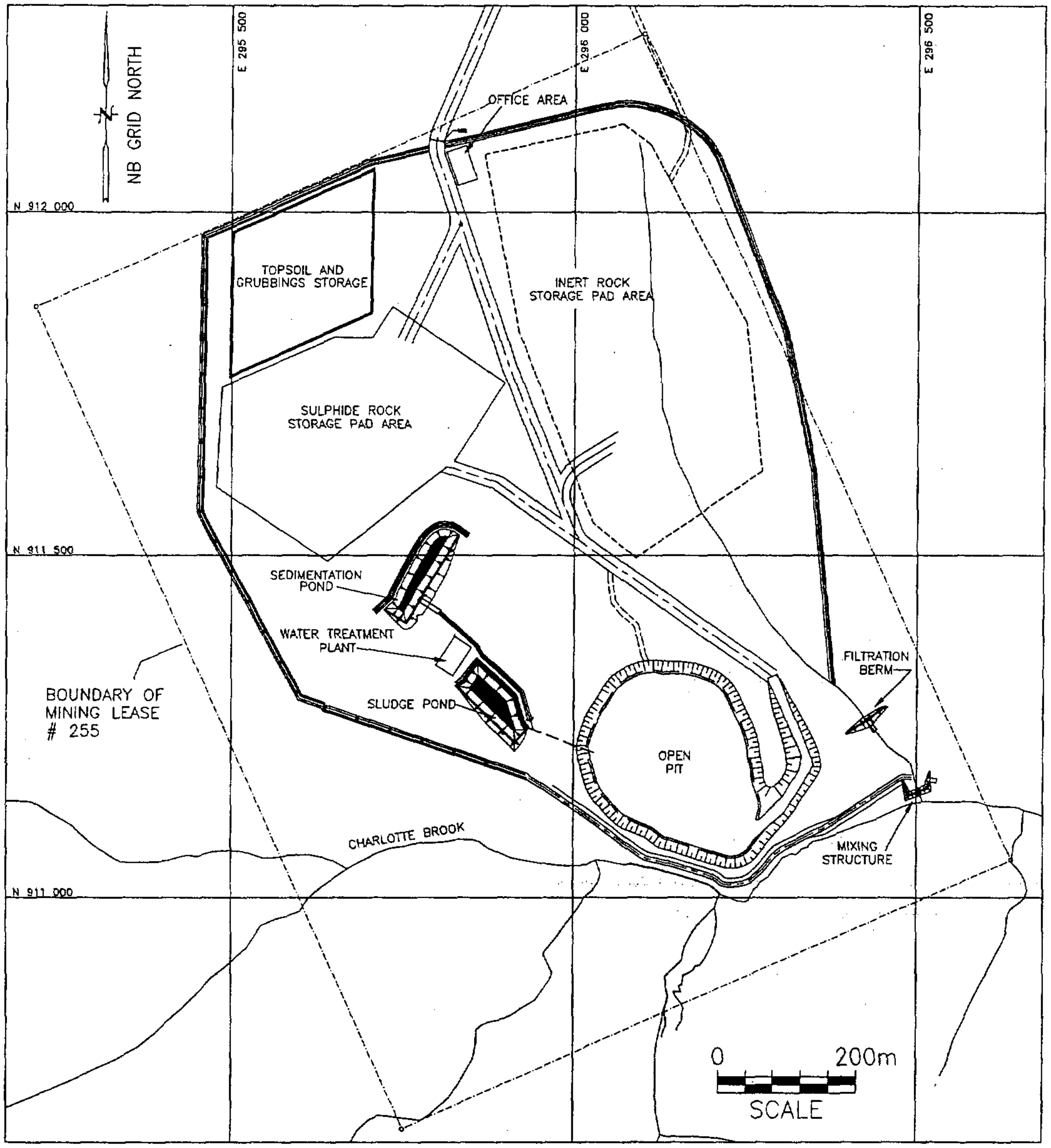
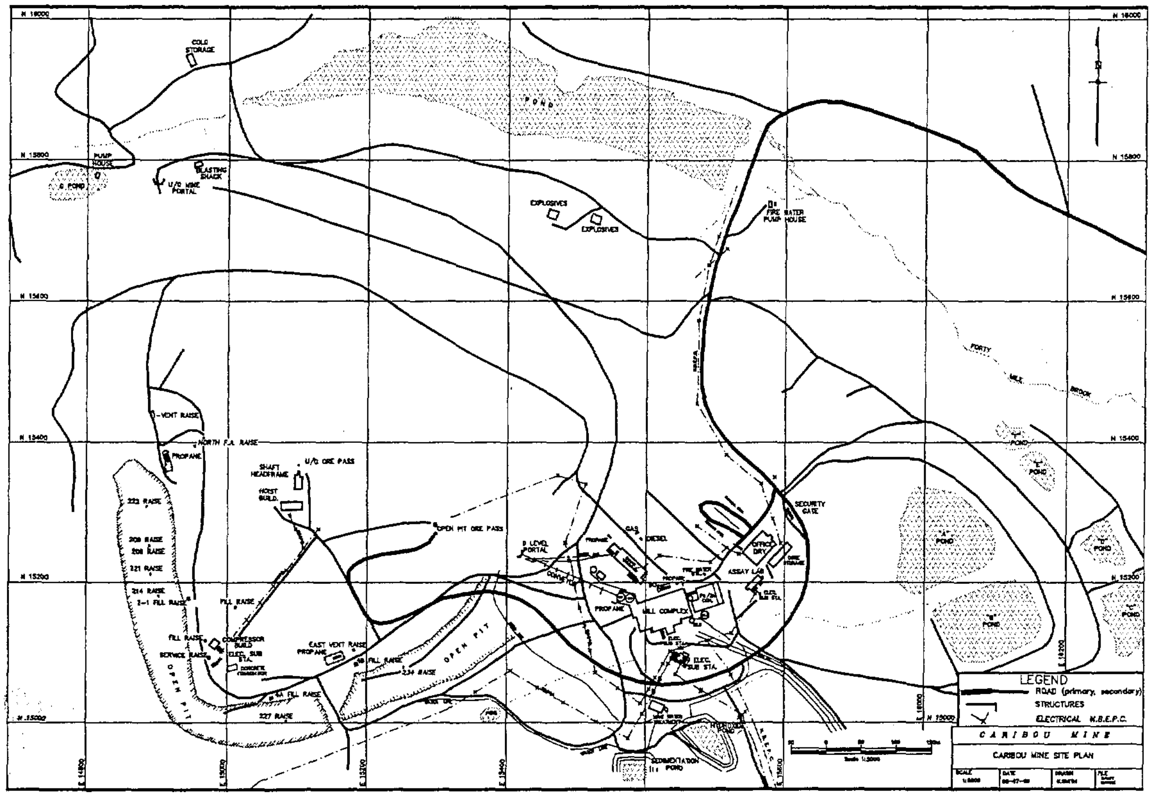


Figure 2.21: Caribou Mine, Mining Lease Covering Open Pit Operations



The Caribou open pit mine was acquired in October 1995 from Marshall Minerals Corp. for 1,500,000 Common Shares and a royalty payment on each tonne mined and hauled to the Caribou mill for processing. The amount of the royalty is $1.00 per tonne when the price of zinc is less than U.S.$0.55 per pound, escalating to a maximum of $4.00 per tonne at a price in excess of U.S.$0.70 per pound. The Caribou open pit zinc/lead/silver deposit is shallow and has higher lead and silver grades than the Caribou underground mine, but is too small to justify construction of its own processing plant. The Caribou open pit mine had not previously been in production. The Caribou open pit and underground mines are referred to collectively herein as the "Caribou Mine". A mine site plan is presented in Figure 2.22.

Figure 2.22: Caribou Mine, Site Plan



A 3,000 tonne per day concentrator complex is located at the Caribou underground mine site. Other surface facilities include a headframe and production hoisting system, a shop-warehouse complex, a compressor building and compressors, a mine dry, assay laboratory and an administration building. Mine services include a fresh water pumping and distribution system, fire protection and a 100-hectare tailings impoundment pond located 1.5 kilometres southeast of the concentrator. Electric power is obtained from New Brunswick Power. A 1.5 hectare deep-water concentrate loading and storage building and facilities site is located on the coast at Belledune, New Brunswick, 75 kilometres from the Caribou mill. This facility can receive concentrates from the Caribou Mine for shipment to smelters in Europe, North America and Asia.

2.5.7.3 History

The Caribou massive sulfide deposit was first discovered by Anaconda in 1955. In 1965, extensive underground drifting, sampling, pilot plant testing and drilling were initiated and a small high-grade secondary copper cap was developed into an open pit mine-mill operation until the depletion of the copper mineralization in 1974. In 1982, a silver/gold heap leach plant was constructed to process 60,000 tonnes of gossan materials, which were stockpiled during earlier pit operations.

In December 1986, East West Minerals NL ("East West"), an Australian public company, purchased the mine from Anaconda through a subsidiary, East West Caribou Mining Limited ("EWCM"). A feasibility study of the project was developed based on production of a bulk lead/zinc concentrate and in 1988 a concentrator with a design capacity of 2,000 tonnes per day was commissioned and operated until July 1989 when operations were suspended pending the formulation of a new mining plan.

On April 9, 1990, the Caribou concentrator recommenced milling operations. Pursuant to a series of transactions during the summer and fall of 1990, EWCM became wholly-owned by Bathurst Base Metals Inc. ("BBMI") and BBMI became wholly-owned by BWR. The mine and concentrator operated for seven months in 1990. On October 26, 1990, operations were suspended due to poor recoveries and falling metal prices. The mine/concentrator/tailings pond complex was placed on care and maintenance.

In late 1994, BWR initiated a metallurgical review of the Caribou underground mine mineralization with the objective of developing a process capable of producing separate saleable zinc and lead concentrates, and Lakefield Research performed a pilot plant test. In 1995, the decision was made to re-open the Caribou Mine. The plan for the Caribou Mine was developed based upon the combined mineral reserves of the Caribou underground and open pit mines and an increase of the mill capacity to 3,000 tonnes per day.

The addition of the Caribou open pit deposit to the mine plan facilitated the early start-up of the Caribou Mine allowing sufficient time to develop the underground mine to reach its planned capacity and use a more efficient system of material transfer from underground to the mill.

The Caribou underground mine and Caribou open pit operated as expected. Although the mill commenced production in July 1997, mechanical deficiencies and design shortcomings prevented the operation from attaining commercial production. A number of the mechanical problems and design deficiencies were resolved by the end of 1997 allowing BWR to focus on achieving the metallurgical performance targets contained in the feasibility study. Throughout 1998 the metallurgical performance of the Caribou mill improved steadily but fell short of the levels anticipated by the feasibility study. As a result, and also because of declining metal prices, a decision was made to extend a planned maintenance shutdown that commenced in August 1998. The operation was placed on care and maintenance pending additional technical and economic studies.

3. GEOLOGY

3.1 Introduction

This section describes the geology of the BWR mining assets. For each mining asset the nature and geometry of the orebodies being, or planned to be mined, their structural complexity and the variability of their commodity grades or qualities is discussed.

3.2 Bouchard-Hébert Mine

The Bouchard-Hébert Mine is situated within a succession of rhyolitic flows and felsic pyroclastic rocks comprised in the uppermost member of the Noranda Subgroup, in the Blake River Group. The Blake River Group consists of Archean felsic to mafic volcanic rocks in the southern part of the Abitibi Greenstone Belt in the Superior Province. The Blake River Group is bordered to the north and south by sedimentary rocks of the Kewagama and Cadillac groups respectively. Contacts between the Blake River Group and the adjacent sedimentary groups are characterized by large deformation zones.

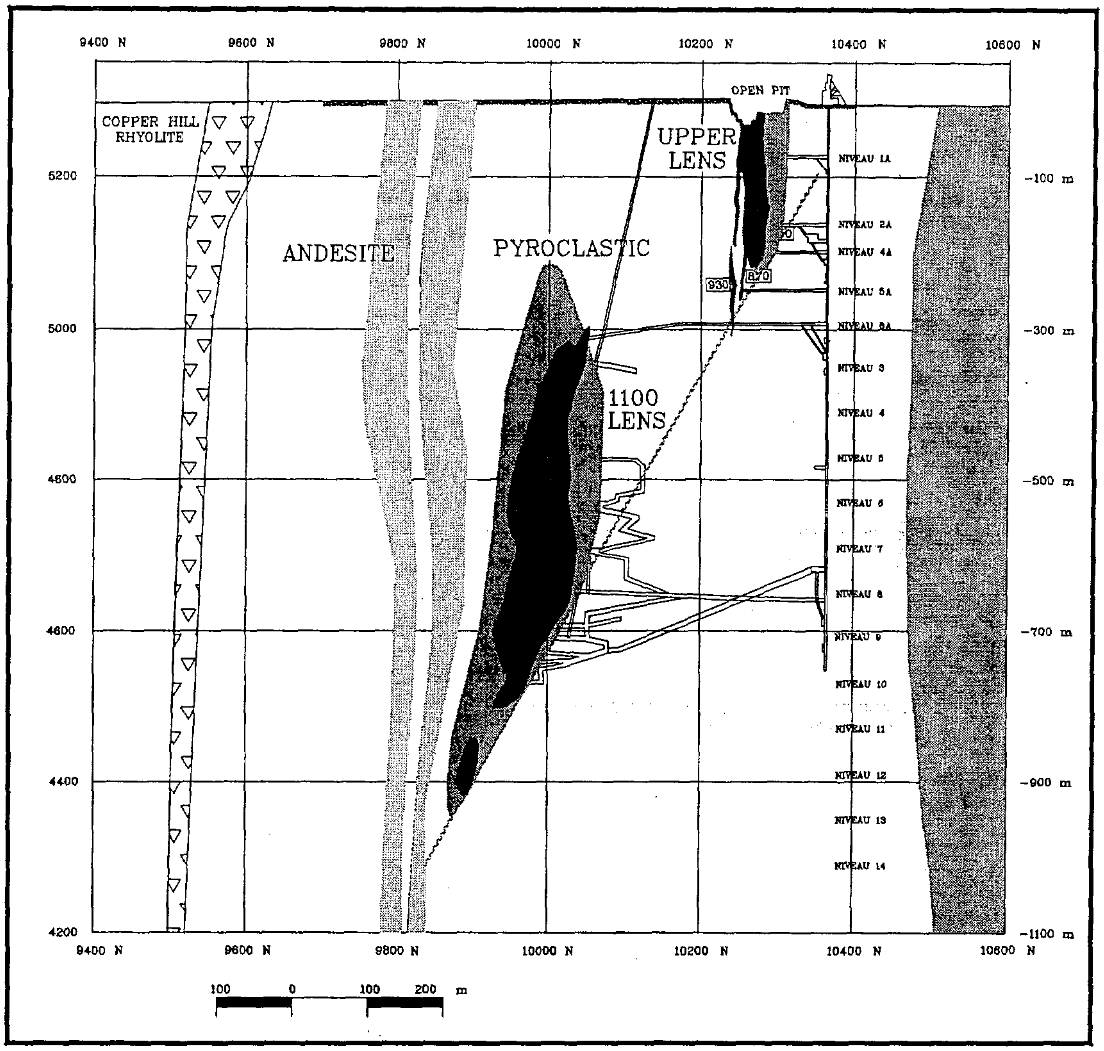
Structural analysis of the Blake River Group volcanic rocks have revealed large "Z" structures resulting from the combination of two former fold systems striking east/southeast to west/northwest and east to west. The whole domain is transected by two major fault zones: the Porcupine-Destor fault to the north and the Cadillac-Larder Lake fault to the south.

Two volcanogenic massive sulfide (VMS) deposits have been identified to date including the Upper Zone (Mobrun Mine), which has been completely mined-out, and the 1100 lens, which is currently being mined. These two zones are approximately 250 metres across strike from each other. The upper part of the 1100 lens is 300 metres below the surface.

The 1100 lens is parallel to the regional foliation, and strikes at 110 degrees to 120 degrees, while the dip varies from vertical to steeply dipping to the south (Figure 3.1). The 1100 lens consists of one sub-vertical massive sulfide lens that plunges steeply to the southeast and consists of 85% pyrite, 5-15% sphalerite and 1-5% chalcopyrite. The western and upper portions of the deposit have relatively elevated concentrations of zinc, gold and silver and are often narrower, while the eastern portion of the lens is generally thicker but contains lower grades of zinc, possibly due to a wider dispersion of the mineralizing fluids.

In the upper portions of the lens (i.e. above Level 5), the "ore" outlines typically mimic the boundaries of the massive sulfide zone, while in the deeper portion of the deposit, wider sections of massive sulfide mineralization typically host several parallel zones of zinc mineralization. The ore outlines correlate with a natural geologic boundary where the zinc grades within the massive sulfide are typically in the range of 1-2% zinc, and increase to 4-5% zinc within the defined ore zone. Typical of most VMS deposits, a copper-rich zone of mineralization occurs along the footwall of the deposit. The continuity of grade and geometry of this copper-rich zone is less continuous compared to the more zinc-rich mineralization within the massive sulfide zone.

Figure 3.1: Schematic Cross Section Through the Bouchard-Hébert Mine (looking west) Showing the 1100 lens, the Mined-out Upper lens and Mine Infrastructure.



3.3 Nanisivik Mine

The Nanisivik sulfide deposits are hosted in carbonate rocks within a Proterozoic sedimentary sequence. This sequence developed as a Neohelikian intracratonic basin, the Borden Basin, on a peneplaned gneiss complex of Archean-Aphebian age. The Borden Basin is one of a number of similar, penecontemporaneous, temporarily connected basins that developed by rifting along the northwest edge of the Canadian-Greenlandic Shield.

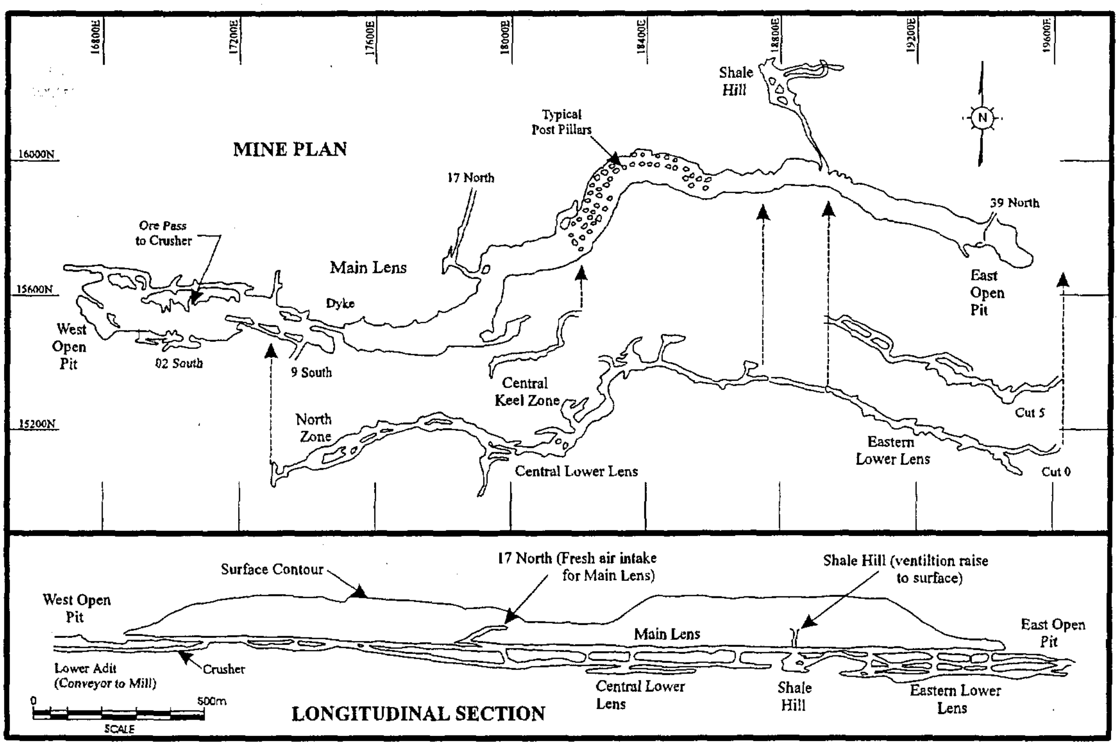
In the mine area, dips are usually quite shallow and the main structure is faulting, much of it of the horst and graben type. Major structures that are recognized in the mine include the South Boundary Fault, which marks the southern margin of sulfide mineralization, and the Keystone Graben Fault, which defines the southern margin of the Main Zone horst.

The various massive sulfide deposits aggregate more than 50 million tonnes of which barren massive pyrite bodies are, by area, the most extensive, and contain the largest sulfide tonnages. Zones carrying sphalerite are present within the massive pyrite bodies, but are areally more restricted in and are also confined to a restricted vertical interval. All of the known significant sphalerite deposits are in horsts adjacent to the Keystone Graben.

The Main Zone deposit (considered to be Mississippi Valley Type) is about 3 kilometres long. It is oriented east-west, although it is sinuous in plan. The deposit is broadly 'T' shaped, with a flat-topped upper section that is typically about 100 metres wide and 20 metres high (Figure 3.2). A remarkable feature of this deposit is the constant elevation of the top of the deposit over its entire length. The keel section of the deposit extends to about 80 metres below the upper section. In places, flat-lying "wings" of sulfides extend out laterally from the keel zone.

Internal structures in the mineralized zones tend to be complex, and range from massive and banded to chaotic or brecciated. Banding tends to be subhorizontal in both the upper section of the Main Zone and the keel section of the deposit, but it may be parallel to dipping dolostone contacts in some areas. As well, the mineralization is porous in places and large irregular zones of ice are present in some faces underground.

Figure 3.2: Nanisivik Mine, Plan and Longitudinal Section of the Mineralized Lenses



3.4 Bougrine Mine

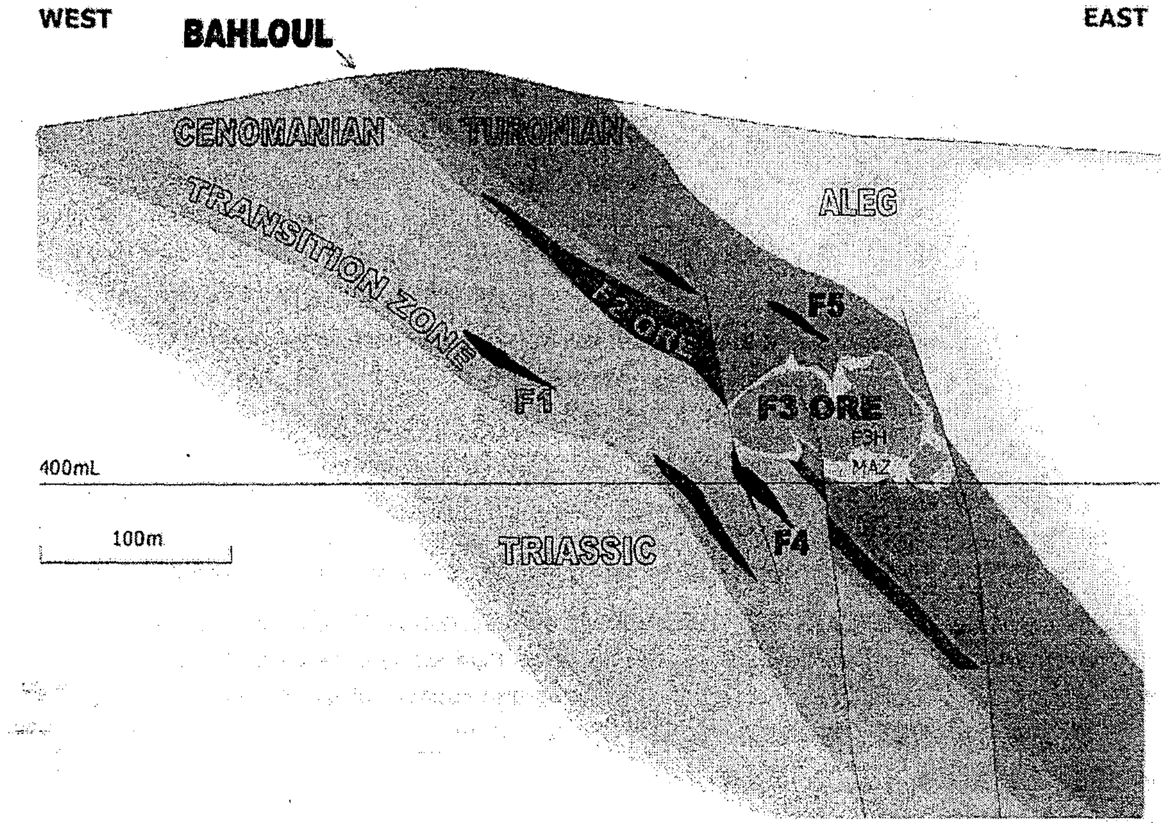
The Bougrine zinc/lead deposit lies in the "Zone de Dômes" of the Tunisian Atlas, a part of the alpine-age Atlas mountain range extending through part of Morocco, Algeria and Tunisia along the Mediterranean coast. The "Zone de Dômes" is characterized by Triassic salt diapirs (Figure 3.3). The Bougrine deposit is located at the north-eastern plunge of the Lorbeus Diapir. The strike direction of the diapir structure is deviated into a north-south direction, and the mineralized bodies lie on the eastern flank, dipping at about 40° to 60° towards the Sers Basin. The area is cut by several fault systems, with the highest intensity of faulting being in the centre of the deposit.

Currently five mineralized zones are distinguished within the Bougrine deposit area. They are designated in order of their discovery as F1 to F5. The majority of the proven and probable reserves are located in the F2 and F3 zones. The F2 zone is a stratabound dissemination in laminated limestone. The mineralization is sharply delineated. Using a 6% zinc-equivalent cutoff, the flat lens-shaped zone extends for about 500 metres along strike and 500 metres down-dip. Maximum true width is close to 40 metres and average mineable width is 10 metres.

The F3 zone is an irregular pipe-like deposit about 130 metres long, striking west to east and plunging slightly eastwards. It is up to 80 metres high and typically has a width of 40 metres to 50 metres. A semi-massive core zone returning grades in the range of 5-15% lead and 24-45% zinc is surrounded by a moderate zinc dissemination in marly limestone.

The F1 zone consists of various small lenticular, strata- or fault-controlled deposits. It is the only part of the Bougrine deposit with surface outcrops. Average grades are below 10% zinc and 2% lead. The F4 zone was discovered in mid-1995 and is thought to be the feeder zone of the F3 zone. Drilling to date has resulted in a number of very high grade but fairly narrow intersections. The F5 zone can be regarded as a lateral continuation of the low grade halo of the F3 zone with grades averaging below 10% zinc.

Figure 3.3: Bougrine Mine, Schematic Cross Section (looking North) Through the Bougrine Deposit

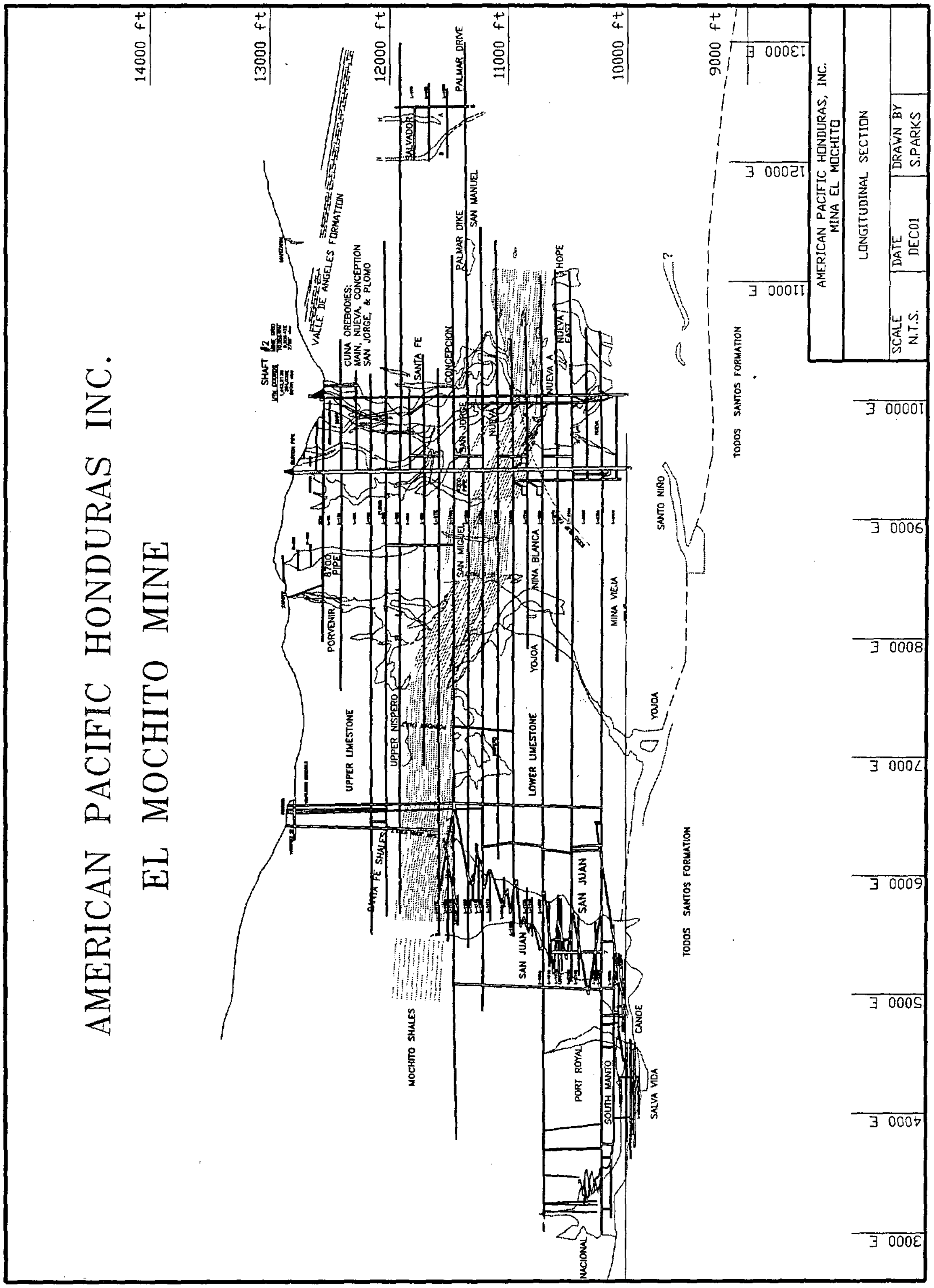


3.5 El Mochito Mine

Zinc/lead/silver mineralization at El Mochito occurs in sedimentary rocks of Cretaceous age, and belongs to the economically important class of high-temperature replacement zinc/lead deposits in carbonates. Carbonates are particularly susceptible to replacement by acid hydrothermal solutions which, in the case of El Mochito, have deposited skarn minerals such as garnet, epidote and pyroxene together with sulfides of iron, zinc and lead.

The replacement deposits can take two shapes, some follow the essentially flat bedding of their host rock ("mantos") while others cut across the rocks ("chimneys" or "pipes"). At El Mochito, both of the replacement deposits are prominently developed, with mantos forming at the lower contact of the Cretaceous Atima limestone, where upwelling solutions emerged from the underlying Todos Santos siltstone package (Figure 3.4). Mantos also formed at the lower contact of the Mochito shale, a 150 metre thick limy siltstone unit some 550 metres above the base of Atima limestone. In many cases, a chimney-type connection between the lower and upper mantos is present, the largest of which is the San Juan pipe, now largely mined out. Others are the Nacional, Salva Vida, Yojoa, Nina Blanca and Nueva pipes. Overall, some 75% of the total known tonnage at El Mochito occurs in the chimney/pipe setting.

Figure 3.4: El Mochito Mine, Longitudinal Section



There is also a tendency for the formation of manto-like bodies immediately above the Mochito shale from which a number of individual pipes rise into the 450 metres of the overlying upper Atima limestone where a number of high grade pipes or chimneys sustained the mine in its earlier history. The known mineralization at El Mochito occurs within a rock volume measuring some 2.5 kilometres east-west and 600 metres north-south, with a vertical (stratigraphic) extent of more than one kilometre. Within these dimensions, the known mineralized bodies occupy approximately 0.3% of the overall volume.

El Mochito has a pronounced overall vertical metal zoning. It is generally accepted that a set of pre-existing faults guided the ascent of the mineralizing fluids. Of particular importance are the sub-parallel, generally east-northeast trending Porvenir, Main and Nacional/Salva Vida faults. All of the major mineralized bodies discovered to date at El Mochito are localized by the intersection of these faults with north-northeast trending "N" style faults.

The discovery in 1999 of the high-grade Port Royal zone, which is similar to smaller high-grade zones mined decades ago, however, over 770 metres west of similar structures previously mined, confirmed the continuing exploration potential of these structures.

Post-mineralization faulting is ubiquitous. A prominent set strikes northeasterly and has steep dips, with the hanging-wall side moving down. While the offsets along these faults are generally small, they create poor ground conditions. Knowledge of their location in space is required for detailed mine layout.

3.6 El Toqui Mine

The El Toqui property is situated in an area of Jurassic-Cretaceous volcanic and volcano-sedimentary rocks which are intruded by intermediate to felsic porphyritic bodies of Upper Cretaceous to Tertiary age. The volcanic and volcano-sedimentary rocks have flat to shallow dips in the El Toqui area.

The areas covering the El Toqui claims are mainly characterized by zinc, gold and copper mineralization in vein, stockwork and disseminated form. The host rocks are volcanic and sedimentary types, from the Jurassic-Cretaceous period.

The first belt is of the "Toqui type" and is approximately eight kilometres wide by thirty kilometres long with a northwest orientation, coinciding with the major structural trend in the region. This belt extends from the source of the River Mañihuales, in the northwest, to Cerro Huemules, in the southeast. Stratiform, skarn, mesothermal veins and replacement type mineralization are found. The area displays favourable geological conditions, geochemical presence of zinc, lead, copper, gold and molybdenum and important geophysical anomalies. In general terms, the targets located in this belt are regarded as attractive for the exploration and development of polymetallic deposits. Structurally, the area of the property is characterized by block faulting. The dominant strike direction is west-northwest to north, but northeast faults of secondary importance are also present (Figure 3.5).

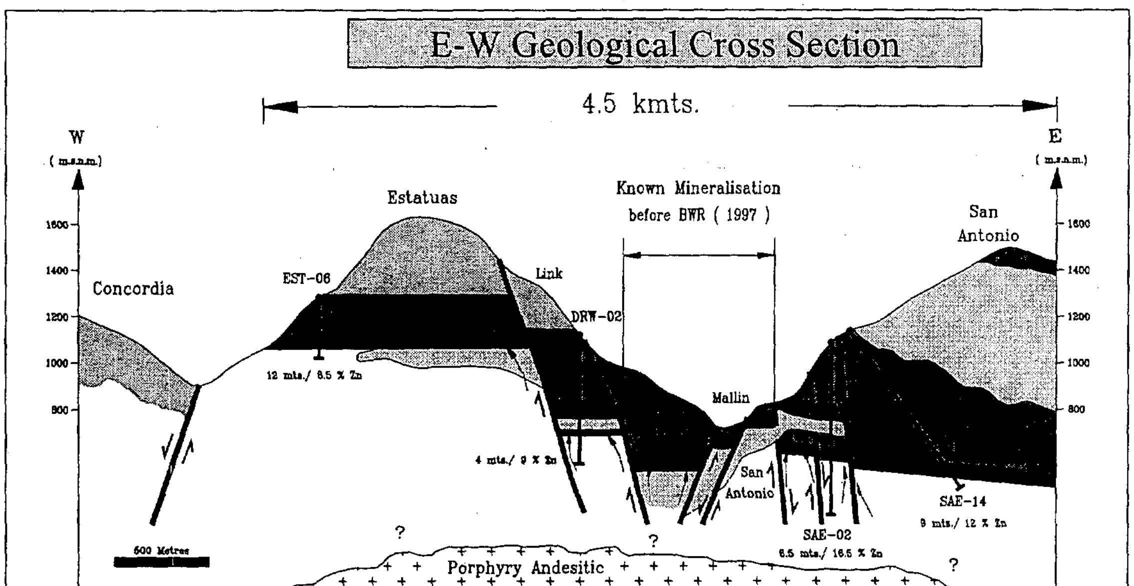
The El Toqui property can be divided into two sectors in terms of mineralization. Zinc and zinc/gold mineralization, as at the El Toqui Mine, is present in the western part of the property within a west-northwest trending area about eight kilometres by thirty kilometres. The El Toqui mining district occupies a small part of this sector, about five kilometres by six kilometres. It appears to represent a horst bounded by west-northwest faults.

Zinc/gold mineralization at El Toqui occurs as manto deposits hosted by an eight metre to twelve metre thick bed of fossiliferous limestone, known as the Main Manto unit. The Main Manto unit, or Coquina bed, occurs within a sequence of volcano-sedimentary rocks in the lower part of the Coyhaique Formation. Sulfides, mainly sphalerite and pyrrhotite, selectively replace the matrix of the fossil shells, and in places the fossil shells themselves, in the Main Manto unit. Other sulfides present in lesser quantities include pyrite, galena, chalcopyrite and arsenopyrite.

The Main Manto unit is not mineralized everywhere. Vertically, the Main Manto unit contains economic grades in the mine area with thicknesses from a few metres to twelve metres. In most places, a porphyry sill forms the footwall to the Main Manto. Locally, apophyses from this sill extend up through the Main Manto unit.

The three original deposits at El Toqui, Doña Rosa, Mallin-Monica and San Antonio, adjoin each other but are separated by faults. They were likely part of the same mineralized body, but are now at different elevations due to block faulting. The Doña Rosa deposit, the most westerly, is faulted down 40 metres to 50 metres from the Mallin-Monica deposit, which in turn is stepped down by faulting from the San Antonio deposit, the most easterly body. The faults are occupied by porphyry dikes. In 2000, sufficient exploration activity occurred to identify several new deposits (Estatuas, Aserradero and Mallin-sur), all believed to be part of the original mineralized body but now at different elevations due to faulting.

Figure 3.5: El Toqui Mine, Geological Cross Section



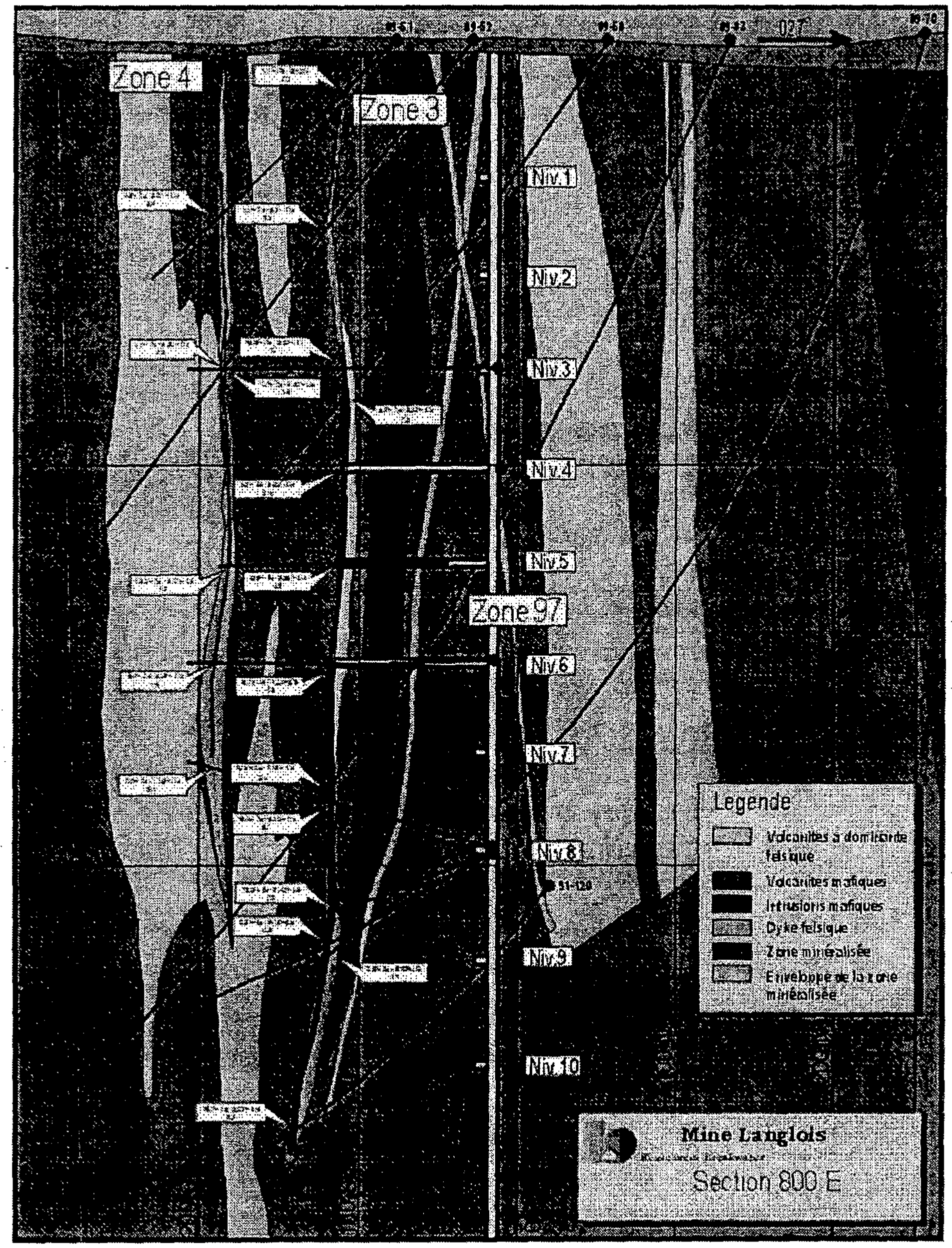
3.7 Langlois Mine

The Langlois Mine produces zinc (along with lesser values of copper, silver and gold) from narrow, tabular VMS bodies. They are hosted within mafic to intermediate volcanic and volcaniclastic units in the central-east portion of the northern Archean volcanic belt of the Abitibi Sub-province, or more precisely, within the Miquelon segment. The lithologies in the area predominately consist of a succession of mafic to intermediate lava flows and volcaniclastic with less abundant felsic volcanic and sedimentary units. The rock sequence has been affected by a regional deformation, which forms sub-vertical isoclinal folds. The regional metamorphism reached the green schist facies. The predominant structure in the area is the Cameron shear zone, which trends 120 degrees and extends for more than 80 kilometres along strike and is up to 5 kilometres thick. The massive sulfide horizons at the Langlois Mine are hosted by the strongly schistosed rocks of the Cameron shear zone.

The Langlois Mine contains four zinc-rich orebodies consisting of zones of massive sulfide, primarily pyrite and sphalerite, occurring within a thick, highly deformed felsic volcanic sequence injected by numerous barren mafic dikes. Each massive sulfide body is relatively thin (1 to 8 metres), but with considerable vertical and lateral extensions (> 500 metres in either direction). The massive sulfide zones trend easterly with a near vertical dip, sub-parallel to the regional structural fabric. The zones are stacked across the felsic sequence along a narrow corridor slightly oblique to the main structural trend. From southwest to northeast the zones are: Zone 5 (small uneconomic lens near surface), Zone 4, Zone 3 and Zone 97. In longitudinal section, each massive sulfide zone portrays an elongated lensoid shape, whose long axis plunges moderately towards the southeast, parallel to the plunge of the regional stretching lineation. In addition, the centre of gravity of each lens becomes progressively deeper moving along the stacking corridor toward the northeast. Consequently, the top of Zone 97 is located at approximately 300 metres below surface. Ore production at the Langlois Mine has come exclusively from two zones, namely Zones 3 and 4, while Zone 97 was discovered in 1994 but was not fully defined until recently (Figure 3.6).

Mafic dikes cut the mineralized zones in many areas, and have historically been a major contributor to dilution. In addition, the well-foliated, chloritic volcanic host rocks have contributed to ground-control problems and often excessive dilution.

Figure 3.6: Typical Cross Section (looking west) Through Zone 97 Showing Geometry of the Sulfide Mineralization.



3.8 Caribou Mine

The Caribou underground and open pit mines are located in Northern New Brunswick, where over 30 massive sulfide deposits occur in the province. The deposits are located, broadly, between footwall sediments and hangingwall volcanics (Figure 3.7). The hangingwall to the massive sulfides is a sequence of clastic felsic volcanic flows with local bedded tuffaceous horizons. Sulfide minerals are not present in the hangingwall. The footwall rocks are rhyolitic with tuffaceous horizons and, unlike the hangingwall sequence, disseminated sulfide minerals occur in a stringer zone immediately below the massive sulfide body. The deposit is cut by two diabase dykes of Devonian age, one in the centre of the deposit and one at its southern end. These dykes trend northwest/southeast and dip steeply. The central dyke is 2 to 5 metres thick. Three phases of folding have been identified, all of which predate the intrusion of the diabase dykes. The first phase imposed gentle folds along the long axis of the deposit. The second phase resulted in local, minor kink folds and the third phase resulted in steepening of the flanks of the deposit, approximately along strike.

The footwall in direct contact with the sulfide mineralization is, usually, a band of phyllite averaging about 15 metres in thickness (but ranging from 3 to 25 metres). This unit contains disseminated pyrite near the contact with the massive sulfide lenses. The sulfide mineralization is overlain by felsic volcanics. In the mine area, a band of phyllite one to three metres thick occurs between the massive sulfides and the felsic volcanics. Two major faults cut the Mine Sequence, one north/south and the other east/west.

The mineralization delineated to date consists of six discrete massive sulfide lenses, overlapping in part, and distributed in an echelon fashion in, and conformable within, the Mine Sequence. The lenses have been numbered from 1 to 6. The massive sulfide lenses consist of about 90% sulfide minerals, mostly pyrite but with significant amounts of magnetite. In order of abundance, the major minerals are sphalerite, galena and chalcopyrite. The sphalerite is reported as being marmatitic. Minor metallic constituents include tetrahedrite, arsenopyrite, marcasite and gold. In contrast to other massive sulfide deposits in the district, pyrrhotite is rare. The gangue minerals mostly consist of siderite, stilpnomelane, quartz and chlorite.

The sulfide minerals are very fine-grained with grains generally less than 50 microns across. Textural and mineralogical banding is common, with the latter type, usually made up of pyrite and sphalerite/galena layers, being characteristic of higher grade zones.

The individual massive sulfide lenses exhibit well-developed metal zoning. As well, there is an overall metal zoning from lens to lens; thus, the zinc/lead ratio is the highest in lens 1 and lowest in lenses 4 and 6, mainly due to an increase in galena in the latter lens. The deposit also exhibits distinct metal zones not coincident with the general metal zoning.

Silver varies proportionately to lead as most of the silver is in solid solution in galena in lenses 1, 2, and 3, and in solid solution with tetrahedrite in lenses 4 and 6. Gold occurs mainly in electrum, but is also noted in arsenian pyrite and arsenopyrite.

The Caribou open pit (Restigouche Deposit) is unlike others in the Bathurst area in that it lies entirely within volcanic rocks. Overall, the deposit is tabular in form, with the long axis striking north 15 degrees west, and with a dip of less than 35 degrees to the west (Figure 3.8). The deposit plunges approximately 20 degrees on a bearing of north 35 degrees west. The tabular, massive sulfide body extends 425 metres along strike and is up to 120 metres wide and 45 metres thick.

Figure 3.7: Caribou Mine, Plan of the Deposit

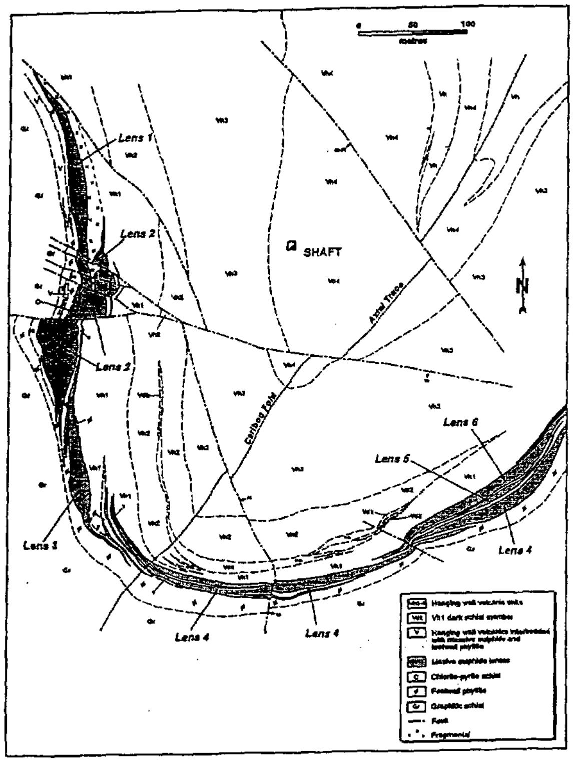
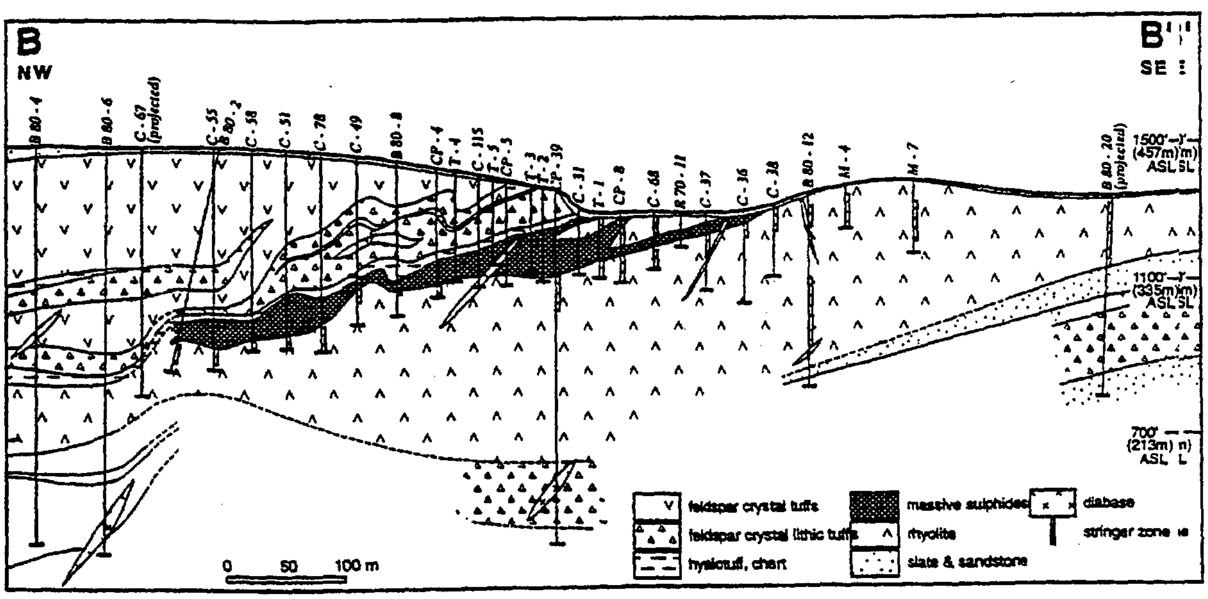


Figure 3.8: Longitudinal Section of the Restigouche Deposit



4. MINERAL RESOURCES AND MINERAL RESERVES

4.1 Introduction

The mineral resources and mineral reserves for each mine have been classified by BWR's Qualified Person according to the "*CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines*" (August, 2000). Accordingly, the Resources have been classified as Measured, Indicated or Inferred and the Reserves have been classified as Proven and Probable based on the Measured and Indicated Resources.

The mineral resources and mineral reserves have been based on BWR's medium term projected metal prices.

4.2 Bouchard-Hébert Mine

4.2.1 Introduction

Two VMS deposits have been identified to date including the Upper Lens (Mobrun Mine), which has been completely mined-out, and the 1100 lens, which is currently being mined. In the upper portions of the deposit (i.e. above Level 5), ore outlines typically mimic the boundaries of the massive sulfide zone, while in the deeper portion of the deposit, wider sections of massive sulfide mineralization typically host several parallel zones of zinc mineralization. Typical of most VMS deposits, a copper-rich zone of mineralization occurs along the footwall of the deposit.

BWR utilizes two different methodologies to calculate the resources and reserves, a 2-dimensional, polygonal methodology in areas of dense drill data and underground workings, and ordinary kriging in areas where the density of data is much less, below and above Level 4, respectively.

The mineral resources and mineral reserves were independently audited by SRK in May, 2001 in a report entitled "Audit of Resources and Reserves for the Bouchard-Hébert Mine".

4.2.2 Data

The geological database maintained at the mine is complete and contains current assay, density, survey and geological data from definition drill data primarily from ATW size core that is sampled in its entirety, exploration drill data from BQ size core, and underground chip sampling data, all of which is required to complete resource and reserve calculations and mine planning. Although the chip samples from mine workings and the definition drilling samples are assayed onsite, internal quality control ensures the accuracy of the zinc, copper, silver and gold assays and density. Due to the visible nature of the mineralization, estimates of the zinc grade are made during underground face mapping and core logging and these correlate well with the assay results, thereby providing an opportunity to identify any assay discrepancies.

BWR commissions a semi-annual audit of the database by an independent consultant to identify any irregularities. During the May, 2001 audit, SRK had the opportunity to reconcile the reserve estimates with the metal production. A review of the geological data indicates that previous exploration personnel and current mine operating personnel have exercised great care and attention to detail in the collection, verification and storage of the data. The data is currently stored in a Geostat computer software program.

4.2.3 Continuity of Mineralization

Mine geology personnel extrapolate the zones of "economic" mineralization from hole to hole only when sufficiently confident. Mineralization across strike drops very sharply below 6% zinc. Typically, drilling data spaced 40-50 metres apart is adequate for outlining zones of higher-grade zinc mineralization within the massive sulfide horizon. The drill spacing at Bouchard-Hébert is generally 15-20 metres for definition drilling and 40-50 metres for exploration drilling. Only the area above level 4 is drilled on the wider 40-50 metre spacing. However, the existing mine workings on level 3 and level 4 provide additional data for the delineation of the higher-grade zinc zone within the massive sulfide mineralization. In this area of the deposit, the limit of the higher-grade zinc zone approaches the boundaries of the massive sulfide mineralization.

In general, only regular ore shapes are delineated. Isolated zones of mineralization, areas of erratic grade, or areas of geologic uncertainty are not extrapolated, and are therefore not included in the mineral resources and mineral reserves until additional confirmation data is acquired. SRK considers this to be an appropriate approach.

Typical of most VMS deposits, a copper-rich zone of mineralization occurs along the footwall of the deposit. The continuity of grade and geometry of this copper-rich zone is less continuous compared to the more zinc-rich mineralization within the massive sulfide zone.

4.2.4 Compositing and Geostatistical Analysis

Once the outlines of the economic mineralization are defined, the assays are composited across the entire width of the mineralized envelope in order to accommodate the two-dimensional, cross-section model. The assays are composited based on length and density, since a density measurement is completed for each assay interval. The mineralized zones are delineated based on a $30 per tonne NSR cutoff grade, which has not been changed since the recent drop in the zinc price because the deposit is fully developed. A significantly higher cutoff grade would mean the deposit would start to "break-up", with much less continuity at higher cutoff grades.

Although the majority of the sampling of the drill core and underground chip sampling is completed over a 1.5 metre interval, compositing across the massive sulfide zone is completed initially over 1.5 metres in order to normalize the data for statistical and geostatistical analyses.

Composites of assays over 1.5 metres shows that copper and zinc are bimodal with low and high grade populations, and although skewed to the left (i.e. mean is to the left of the mode), approach a normal distribution when transformed to log values. Gold and silver are log-normally distributed. Although the variograms vary somewhat by metal, they generally show the direction of maximum continuity to be vertical, good continuity horizontally along strike and very poor continuity across-dip. The most characteristic feature of the mineralization is the extremely low local variability in grade, with the nugget value (measure of local variability) for zinc approximately 1/10 of the sill value (measure of variance). This suggests that local predictability of grades is good and a technique such as polygonal methods on cross-section that honours local data may be suitable, given the density of data.

4.2.5 Mineral Resource Estimation

The Bouchard-Hébert mineral resource was calculated using polygonal blocks on vertical cross-sections at 20 metre intervals. The polygonal blocks were drawn to define regular shapes of higher-grade zinc mineralization as defined by diamond drilling and face mapping of the adjacent underground workings. The polygons were drawn mid-point between two intersections. The grade of the polygon was calculated using the composited grade over the length that each drill hole occurred within the polygon. Although this portion of the resource is calculated manually, the resource estimates are detailed and managed very effectively. Two-dimensional methods often incorrectly estimate local areas because they rely on local data, however, variography has shown the local variability in grade is very low and local estimates are reliable. Considering this, and the fact that this method is used only in areas with dense drill data and underground workings, SRK believes the results are reliable.

Above level 4, the mineral resources were calculated within the boundaries of the mineralized zone as defined in the geological model using ordinary kriging into 5 metre x 5 metre x 3 metre blocks. Although the model blocks indicate a resolution not supported by the 40-50 metre drill spacing, the very low relative nugget value indicates that this does not have a material impact on the resources. Kriging was utilized in this area because the drill hole spacing is typically in the range of 40-50 metres. If polygon estimates were used, they could give an erroneous estimate of the grade because of the reliance of this method on local data. Kriging on the other hand, provides a better estimate of global grade in this area considering both the data density and its spatial distribution. Kriging often results in the predicted grade approaching the mean of the deposit, providing less importance to local intersections, based on the results of the variography. The kriging was examined visually to compare the extent of the "ore" zone with the ore outlines drawn manually by mine personnel. Some adjustments were made, thus calibrating the kriging to the desired results from areas where calibration was possible. Kriging was completed over several areas to compare the kriging results with actual mined stopes. The difference by stope was typically plus or minus 6-7%, which is adequate for resource estimation considering that kriging better estimates the global resource and not the stope by stope grades, depending on the drill spacing.

Very few "outliers" exist at the Bouchard-Hébert Mine, particularly when considering the high density of data, and therefore, capping of the zinc grades was not considered to be necessary. The zinc, copper, silver and gold grades are "well-behaved", approaching a log-normal distribution. As for the area above level 4, where the drill density is much less, kriging was utilized to estimate the grade of the deposit. Kriging accounts for any high-grade outliers and therefore no capping is required in this area.

After reviewing the grade interpolation, SRK believes that the combination of these two methodologies is a valid way to estimate the resources.

4.2.6 Conversion of Mineral Resources to Mineral Reserves

The indicated or measured mineral resources were converted to mineral reserves by the application of a minimum mining width of 3.0 metres and an NSR cutoff value of $30. The reserves so determined consist of contiguous zones of mineralization delineated in the geological model, while isolated areas are not included.

4.2.6.1 Choice of Mining Method

The mining method and stope designs were chosen based on the context of the resources. The favourable geometry of the deposit, and the very competent ore and surrounding rock have allowed the efficient use of the longhole open stoping method with large stable stopes and paste backfill. Levels are spaced 60 metres vertically and 165 mm diameter production blast holes are drilled. A stope and pillar sequence is used throughout the mine with 15 metre wide primary stopes and 20 metre wide secondary stopes. Four to five stopes are cycled at a time, ensuring that tonnage requirements to the mill are sustained.

The NSR cutoff value of $30 per tonne used to define reserves is based on the historical mining and milling cost. The January 2001 cost statement for the mine was reviewed, confirming these costs. The $30 mining and milling cost is low compared to many other similar mining operations producing at this rate, reflecting the efficiency of the operations at Bouchard-Hébert. This efficiency stems from the large stable stopes and 60 metre level spacing that serve to minimize development costs on a per tonne mined basis. The large stopes also allow the use of highly productive drilling, blasting and mucking equipment.

4.2.6.2 Dilution and Mining Recovery

Ground conditions are very good at the Bouchard-Hébert Mine, allowing large stopes to be developed with minimal over-break beyond the limits of the known reserves, thus minimizing dilution and maximizing recovery. Internal dilution, and to some degree, external dilution, are accounted for during the initial construction of the two-dimensional sectional polygon, when regular shapes are drawn and the dilution is included in the composited grades.

Primary stopes generally use an estimated mining dilution of 4% (by volume) with a recovery of 100%. For secondary stopes, a dilution of 12% was used with a recovery of 90%. The above parameters have been established from very detailed volumetric examinations of the mined-out stopes, reconciled to the reserve estimates.

It is SRK's opinion that the planned dilution and mining recovery factors are appropriate given the characteristics of the rock strength, continuity of grade and geometry, and the mining method. The geology department follows good grade control practices, ensuring maximum recovery of the reserves while minimizing dilution.

4.2.6.3 Reconciliation

Based on a reconciliation of grades with the milled tonnage, the copper grade of the reserve is increased by 5%, the zinc grade decreased by 5%, and both silver and gold grades are increased by 3%. This is based primarily on experience and it has not been possible to gain an understanding of the underlying causes of these grade adjustment factors. Possible sources of these factors may be found in the mill reconciliation, the resource estimate, or the dilution and ore recovery percentages applied.

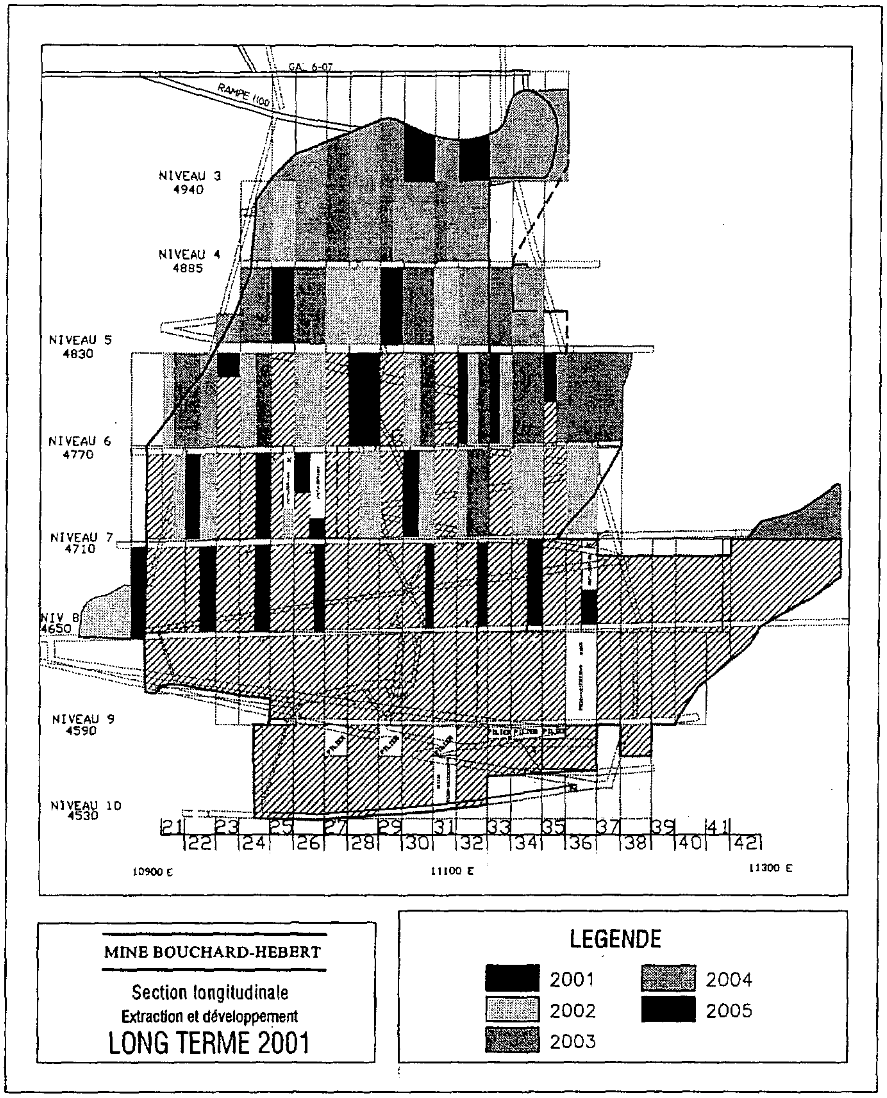
During 2000, considered to be a typical year, unrecoverable reserves left in pillars amounted to 177,000 tonnes. Also during the year, an additional 116,000 tonnes was gained while mining the existing reserves, as favourable changes to the interpretation were verified by mining progress. The net loss of reserves is therefore approximately 60,000 tonnes. Although mine personnel believe that they can better recover the ore with less lost to pillars, SRK considers it prudent to subtract the historical annual loss of 60,000 over the remaining mine life reserves of four years, for a total of approximately 240,000 tonnes.

4.2.7 Mine Plan

The mine is currently producing approximately 90,000 tonnes of ore per month and is able to meet the forecasted mill production. The mining department is very competent at developing the short and long term mine plans. The general philosophy for the mine plan is to continue to develop the upper levels while mining the secondary stopes from the lower levels of the mine. This will ensure that new working places in the upper mine are provided on a timely basis to support continuous production. As development advances in the upper portion of the mine, there will be sufficient time to refine the reserve estimate in these new mining areas.

In SRK's view, the current mine plan (Figure 4.1) for the remaining reserves is logical and straightforward and provides an opportunity to identify additional adjacent resources such as the copper footwall mineralization. Although there exists some opportunity to alter the current mine plan because of the current lower metal prices, this is not considered to materially alter the current reserves. Any selective mining of higher grade areas could jeopardize the mine's ability to recover some ore and therefore such a strategy is not pursued in the plan.

Figure 4.1: Longitudinal section of the 1100 lens showing the reserves to be mined in the current mine plan.



4.2.8 Mineral Resource and Mineral Reserve Classification

A large portion (approximately 75%) of the current reserves for the Bouchard-Hébert Mine are classified as proven. These reserves are located below level 5. Of these proven reserves, approximately 2/3 are located in secondary stopes where mining has already exposed the mineralization on four sides. The remaining proven reserves are located between level 5 and level 6, where definition drilling on a 15-20 metre spacing covers the ground between the underground workings on these two levels. The remaining reserves are classified as probable, above level 4, where the drill spacing is typically 40-50 metres.

SRK considers the classification of the resources and reserves to be appropriate and in compliance with the CIM standards (August, 2000).

4.2.9 Mineral Resource and Mineral Reserve Statement

SRK's audited mineral resource and mineral reserve statement presented in Table 4.1 is based upon an audited statement effective 31 December, 2000, depletion to 30 November, 2001, planned pillar losses, and estimated irrecoverable pillar losses.

Table 4.1: Bouchard-Hébert Mine, Summary of Mineral Resources and Mineral Reserves

Reserves	Tonnage	Zinc (%)	Copper %	Gold (g/t)	Silver (g/t)
Proven	3,222,000	4.77	0.69	1.2	37.6
Probable	1,192,000	5.14	0.57	0.9	30.3
Sub-Total	**4,414,000**	**4.87**	**0.66**	**1.1**	**35.6**
Mined to 30 Nov.01	954,768	4.59	0.81	1.51	46.9
Expected Losses to pillars (60,000 tonnes/annum)	240,000	4.94	0.62	0.98	32.9
Total	**3,219,232**	**4.94**	**0.62**	**0.98**	**32.9**

Meas. and Ind. Resources	Tonnage	Zinc (%)	Copper %	Gold (g/t)	Silver (g/t)
Total	**3,402,029**	**4.94**	**0.62**	**0.98**	**32.9**

Production based on milled tonnes and grade to 30 November, 2001.
Measured and Indicated Resources include Proven and Probable Reserves.

A portion of the total measured and indicated mineral resources of 3.4 million tonnes grading 4.9% Zn, 0.6% Cu, 1.0 g/t Au and 33 g/t Ag has not yet been converted to mineral reserves. These mineral resources are located in two areas:

- 86,000 tonnes grading 7.3% Zn, 0.5% Cu, 1.1 g/t Au and 19 g/t Ag located along the east side of the deposit above level 5 that have a good potential to be converted to mineral reserves.
- 165,000 tonnes grading 1.6% Zn, 2.4% Cu, 0.9 g/t Au and 47 g/t Ag near level 12, having less potential to be converted to mineral reserves as additional development will be required.

No inferred mineral resources have been identified at the Bouchard-Hébert Mine due primarily to the well-defined deposit that is not currently known to extend beyond its limits.

SRK believes that the mineral reserves are very robust, because ore outlines generally appear to mimic geological boundaries where the zinc grade within the ore is distinctly higher than that of the surrounding massive sulfide/volcanic rocks. In addition, the average NSR value of the ore is almost twice the economic cutoff grade, making the mineral resource and mineral reserve estimate less sensitive to marginal increases in costs or metal price fluctuations.

4.2.10 Exploration

BWR's land holding in the area includes a number of mining exploration properties that cover the same stratigraphic rock sequence that hosts the Bouchard-Hébert massive sulfide deposit. Extensive historical surface exploration work and diamond drilling resulted in the outline of several targets, but failed to locate other massive sulfide horizons of economic merit.

The 1100 Lens, which is well defined by underground openings and drilling, accounts for all of the quantified reserves of the Bouchard-Hébert Mine. No additional inferred mineral resources were outlined within the mine working area. A portion of the perimeter of the 1100 Lens was delineated with a possible increase in the resources on the east side. At depth (below 1,400 metres from surface), the west side of the zone remains open, but it is complex, possibly due to the displacement of the mineralized zone.

Other targets within the mine working area include the east and west perimeters of the 1100 Lens from levels 5 and 6 and a large Pulse-EM off-hole anomaly to the west of the shaft. Potential also exists to expand the resource/reserve base along the secondary plunge on the east side of the deposit at level 4-5 and level 7, and on the west side at level 8.

4.3 Nanisivik Mine

4.3.1 Introduction

The current mineral resources and mineral reserves (30 November, 2001) for the Nanisivik Mine have changed significantly since the 31 December, 2000 reported mineral resources and mineral reserves. This is due primarily to a modification to the life-of-mine (LoM) Plan, which was designed to maximize the cash flow over the remaining 10-month mine life and subsequently cease mining operations in September, 2002. The change in the mine plan is a result of mining to a higher cutoff grade necessitated by the recent decrease in the price of zinc.

4.3.2 Data

The geological database maintained at the mine is very complete and contains current assay, survey and geological data for diamond drilling and underground mapping and sampling, all of which is required to complete resource/reserve calculations and mine planning. Although the majority of the production samples and definition drilling samples are assayed onsite, internal quality control ensures a high quality in the assay results. In addition, due to the visible nature of the mineralization, estimates of the zinc grade are made during underground face mapping and core logging and correlate well with the assay results, thereby providing an opportunity to identify any assay discrepancies.

The data is currently stored in a Gemcom computer software program. A review of the geological data indicates that previous exploration personnel and current mine operating personnel have exercised great care and attention to detail in the collection, verification and storage of the data. Although SRK did not complete verification sampling while on site, a detailed examination of the reconciliation was preformed, as well as the comparison of forecast and actual production, in order to assess the reliability of the data.

4.3.3 Continuity of Mineralization

Generally, the continuity of zinc mineralization is very good within the Main Zone and the Keel Zones (directly beneath the main zone), where mineralization consists of massive sulphides with regular waste/ore contacts. The grade and geometry of the Satellite Zones is often less predictable, where high-grade zinc mineralization occurs in discontinuous sulphide bodies, brecciated areas and in veins. The majority of the deposit has been drilled on 12.5 and 25 metre sections with drill spacing of 5-10 metres on section in areas of complex geology. Based on the continuity of grade and geometry, a drill spacing of approximately 12.5 metres, or even less locally, provides adequate data density to obtain an accurate estimate of the resource/reserves in the majority of the Satellite Zones.

Considerable effort has gone into understanding the geology of the Nanisivik deposit. The work completed to date has concentrated primarily on the Main Zone mineralization, while only recently has attention been focused in the area of the satellite deposits. The geological staff regularly complete detailed geological mapping of the underground faces, recording the geology and more importantly the controls on mineralization. The delineation of the mineralized zones is used to define the geometry of the deposit and stope design that greatly improves the confidence and predictability of the resource/reserve models.

4.3.4 Mineral Resource Estimation

The current mineral resource was calculated by BWR's Qualified Person using polygonal blocks on 12.5 and 25 metre cross-sections. In order to accommodate the two-dimensional, cross-section model created by Nanisivik, zinc grades were length composited across the entire width of the zone. The composite lengths were chosen to attain a greater than 3.8% Zn cutoff grade. However, BWR has used an elevated cutoff grade of 6% zinc for the current mineral resources as a short-term strategy to counter the lower metal prices currently being realized. However, some lower grade material has been added to the mineral resources, and hence the mineral reserves, in order to ensure continuity for mining and to meet production schedules.

The resource blocks were drawn with straight horizontal and vertical lines, thus incorporating some unmineralized dolostone along the perimeter of the mineralized zone. The polygonal blocks were drawn so as to extend across the entire width of the mineralized zone on a 5 metre cut as defined by diamond drilling and face mapping of the adjacent underground workings. The 5 metre cut height was used to represent the expected mining method. The grades of the polygons were calculated using the arithmetic average of the composited grade over the length of each drill hole within the polygon (including dilution from dolostone country rock). A minimum of two intersections were required to construct a polygon and to estimate a grade, thus ensuring that no "one-hit wonders" are included in the resource base. Although the resources are calculated manually, the resource estimates are detailed and managed very effectively.

The relatively uniform zinc mineralization of the Main and Keel Zones suggests that grade capping is not required. In contrast, the variable and high grade mineralization within several of the Satellite Zones require grade capping. A recent analysis by BWR has defined a cutoff grade of 20% zinc for several of the satellite zones, particularly in areas of insufficient, large-spaced drill data.

4.3.5 Conversion of Mineral Resources to Mineral Reserves

4.3.5.1 Choice of Mining Method

The mining method and stope design were chosen based on a 6% zinc 'economic' cutoff grade and the width and orientation of the mineralized zone. The room and pillar mining in the Main Zone has performed very well. Bench and slashing is being utilized more extensively in the Satellite Zones, providing a very flexible mining method to handle variable geometry of the mineralized zone above cutoff grade. In addition, longhole stoping is used regularly for the recovery of sill pillars.

4.3.5.2 Dilution and Mining Recovery

Dilution and mining recovery estimates at the Nanisivik Mine are accounted for during the initial construction of the two-dimensional sectional polygon and are taken into account based on the mining method for each block, the geometry of the mineralized zone, and the historical information from that part of the mine. The mature part of the mine has a thick orebody with very good ground conditions including almost zero dilution.

4.3.5.3 Reconciliation

In addition to the falling metal price, BWR updated the mineral resources and mineral reserves in order to better reconcile budgeted reserves with actual production. Several changes in the resource/reserve methodology include:

- Elimination of the practice of visually estimating the grade of the pillars. Since mining began to diverge from the actual recovered grade by more than 1%, assaying control has been implemented to improve upon this situation.
- Grade capping has been utilized to prevent overestimation of the resource in several of the satellite deposits with high-grade outliers.
- A more realistic interpretation of the discontinuous zones of mineralization has been utilized, particularly in the mineralized zones peripheral to the Main zone.
- The current reserves account better for mineability (recovery and dilution) based on the new cutoff grade. Previous reserve calculations were based on historical mining methods that were not always applicable to the various geometries of the satellite zones.

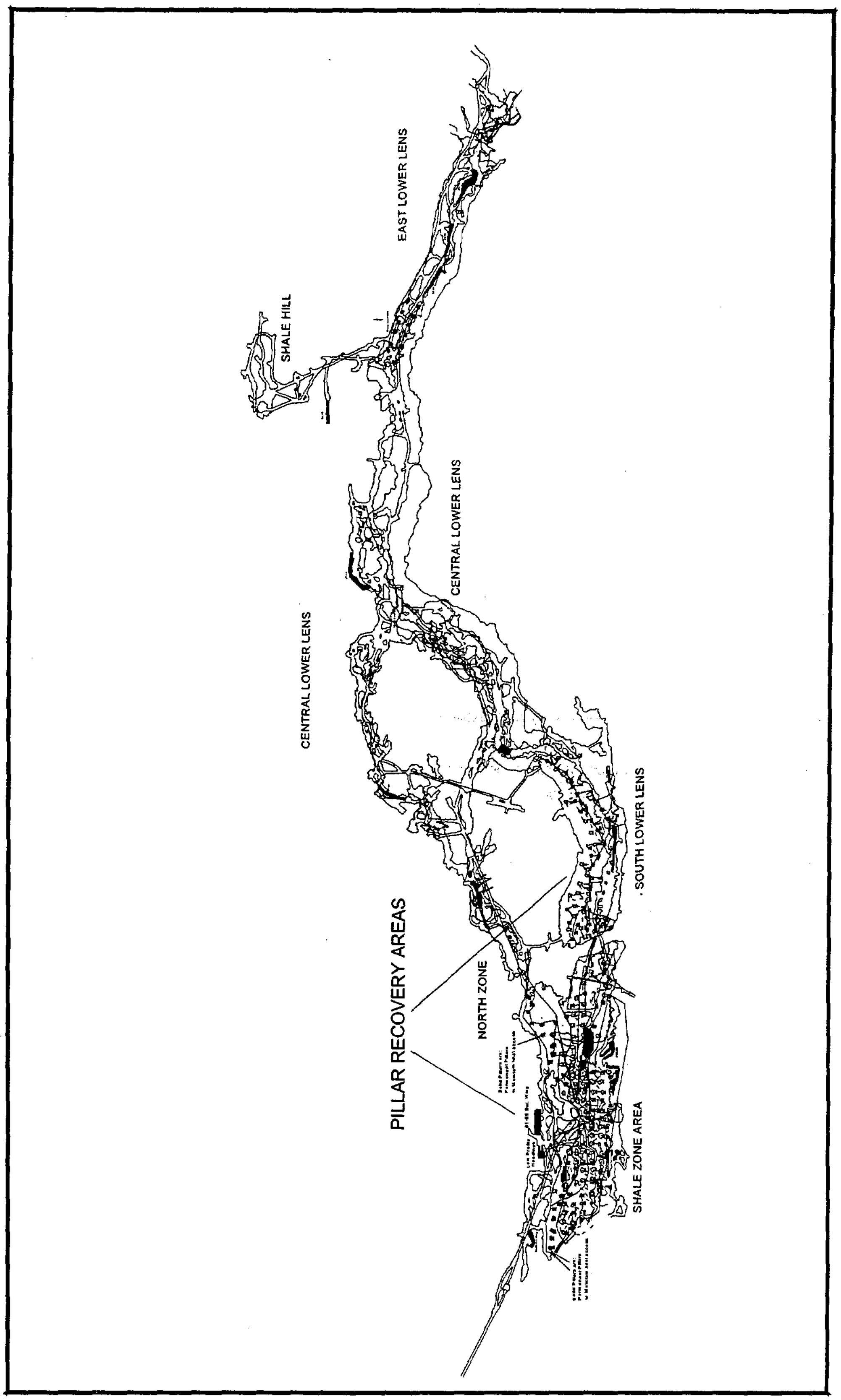
Since the adjustments to the mineral resources and mineral reserves of September, 2001, the mine has consistently improved its ability to achieve the forecast production. Based on the current mineral resources and mineral reserves and mine plan, SRK considers the current production plan to be achievable.

4.3.6 Mine Plan

The current mine plan for 2002 includes mining of 695,296 tonnes grading 7.43% Zn, 0.28% Pb. The objective of the plan is to maximize the cash flow at the Nanisivik Mine and to cease mining in 2002 as a result of falling zinc prices. The decision to close the Nanisivik Mine in September 2002 is largely based on the prevailing market price for zinc. The termination date of the operation was based on the shipping season constraint of the site and the substantial lead-time required to secure supplies. The necessity to change the mine plan and accelerate mining of the pillars resulted in the permanent elimination of access to various mining blocks, thereby revising some of the previously defined reserves to resources and shortening the overall mine life.

The focus of the residual mining activity will be to mine the remaining pillars as well as all higher-grade areas. A minimal amount of development is planned, as the focus will be on retreat mining and scavenging. No new zones will be established. The mining activities will focus on the mining of 122 pillars (Figure 4.2), which account for 47% of the tonnage and 65% of the contained metal. Proper scheduling that follows a standard procedure is required for the plan to be successful. SRK audited this approach in March 2001 and at that time BWR was found to be consistently following the standard procedures.

Figure 4.2: Nanisivik Mine, Plan Indicating Pillars



4.3.7 Mineral Resource and Mineral Reserve Classification

Approximately 50% of the mineral reserves and 40% of the current mineral resources are contained within mining pillars and have typically been mined and sampled on several sides. It is for this reason that this portion of the mineral resources and mineral reserves is known with a sufficiently high degree of confidence to be classified as measured resources and hence converted to proven reserves.

The remaining indicated resources are based primarily on drill hole information. In the opinion of SRK, there is sufficient knowledge of the grade and geometric continuity of the mineralization for the majority of the deposit, given the data density, to determine the grade and tonnage of the majority of the resources consistent with the classification of indicated resources. Inferred resources are located primarily within the satellite zones, along the perimeter of the deposit, where the grade and geometry of the zinc mineralization is often more erratic.

4.3.8 Mineral Resource and Mineral Reserve Statement

It is SRK's opinion that BWR's estimate of the mineral resources and mineral reserves as of September, 2001, and further, the updated reserves as of 30 November, 2001, have been prepared in a professional and diligent manner in accordance with CIM Standards (August, 2000), as summarized in Table 4.2.

Table 4.2: Nanisivik Mine, Summary of Mineral Resources and Mineral Reserves

	30 November, 2001				31 December, 2000			
	Tonnes (000)	Zn (%)	Pb (%)	Ag (g/t)	Tonnes (000)	Zn (%)	Pb (%)	Ag (g/t)
Proven and Probable Reserves	772	7.43	0.4	30	2,868	6.9	0.4	28
Measured and Indicated Resources	2,012	7.15	0.4	30	4,152	6.3	0.4	24
Inferred Resources	110	4.62	0.4	30	359	5.1	0.4	19

Measured and Indicated Resources include Proven and Probable Reserves
Pb and Ag grades for November 30, 2001 resources and reserves are based on previous mining experience, grades are expected to be 0.4% Pb and 30 g/t Ag. Pb is not a payable metal, while Ag adds approximately 1% to NSR value.
Proven and Probable Reserves include 77,255 tonnes for December, 2001 as prorated from 2002 mine plan.

The current mineral reserves are based on the current mine plan. The plan has eliminated access to other areas of the deposit, and the previously-defined reserves outside of the mine plan have since been down-graded to resources. However, if prices improve in the short to medium term, it may be possible to convert these resources back to reserves via a different mine access, although a detailed plan to assess the economic feasibility of such a scenario would be required. The mill is expected to be decommissioned in the summer of 2003, after which time the remaining resources will not have "reasonable prospects for economic extraction" (a requirement for a mineral resource based on CIM Standards) and thereby will be eliminated.

4.4 Bougrine Mine

4.4.1 Introduction

The resource estimation procedure involves the creation of separate block models for the main mineralized zones. The resource estimation methodology has been well developed by BWR based on the experience of mining the Bougrine deposit.

4.4.2 Data

The database for the Bougrine deposit consists of the original surface diamond drill holes, underground diamond drill holes, and underground grade control samples. The geological database maintained at the mine is very complete and contains current assay, survey and geological data, all of which is used to complete resource/reserve calculations and mine planning. The majority of the production samples and definition drilling samples are assayed onsite. Internal quality control ensures a high quality in the assay results and check assays have shown the on site laboratory to have good accuracy.

The data is currently stored in a Gemcom computer software program. A review of the geological data indicates that previous exploration personnel and current mine operating personnel have exercised great care and attention to detail in the collection, verification and storage of the data.

Although SRK did not undertake verification sampling while on site, a detailed examination of the reconciliation was preformed, as well as the comparison of forecast and actual production, in order to ensure the reliability of the data.

4.4.3 Continuity of Mineralization

Mineralization comprising the Bougrine deposit consists of two main styles: stratabound mineralization in the Bahloul ("F2") and crosscutting mineralization in the Turonian limestone ("F3"). The F2 is the larger of the mineralized zones (originally 3.3 million tonnes with an average grade of 12% Zn). The F3 mineralized zone is smaller but of higher grade (originally 0.75 million tonnes with 22% Zn). The F2 in long section looks like a donut, fairly circular in shape, with the F3 cutting through its centre. The continuity of the grade and geometry of the mineralization is often very irregular, ranging from a few percent to more than 30% Zn within a few metres. Two additional mineralized zones, the F1 and F4 are present. Both are small with respect to the main F2 mineralization.

Post-mineralization faulting of medium and small scale has affected the continuity of individual mineralized zones.

4.4.4 Mineral Resource Estimation

The current mineral resources for the Bougrine Mine have been estimated using block models created with a Gemcom software package. The block models are constrained by geologic models that were created by correlating the various mineralized zones and their interaction with faults on all pertinent sections and plans, updated annually with new information.

Block models were created separately for each of the F2, F3 and F4 mineralized zones. For F2 and F3, individual assays were used for grade interpolation without prior compositing. While for F1, full-width composites were created from the drill hole assays separately for material >6% Zn and for material satisfying a 5 metre minimum mining width. This is considered to be acceptable, as nearly all of the samples are 1 metre long. Grades were interpolated into the individual block models using inverse distance squared weighting of grades based on the anisotropic ratio (direction of grade continuity) defined by the semi-variogram curves. Both the drill hole data and the underground grade control samples were used in the grade interpolation.

The resource model for F3 was created in October, 1996. Virtually no new data has been added because the BWR activities have mainly consisted of mining the F3 (except for limited drifting which showed good correlation with the existing model). Although the mineralization is very irregular, as shown by the short ranges on the semi-variogram curves (20 metre along strike × 20 metre down-dip × 10 metre across strike), given the good data density within the F3, the overall results would not be expected to change noticeably by a change in variogram ranges.

For F4, no computer model has been created for this irregular and difficult-to-predict type of mineralization that contributes only a small tonnage to the year-end 2000 mineral reserves, and computations are handled essentially using a polygonal (sectional) approach.

For F5, no mineral resources are carried for this type of mineralization, and there is thus no model.

In the opinion of SRK consulting the 31 December, 2000 resources have been appropriately estimated.

In the opinion of the Bougrine Geology Department, the Bougrine deposit is now so well delineated that only limited scope remains to augment the current reserves, primarily in the area down-dip of the F2 zone that has shown thick but low-grade mineralization that is part of the halo around Bougrine. Similarly, there is considered to be only limited potential to identify any significant new deposits amongst the exploration properties being evaluated by BWR.

4.4.5 Conversion of Mineral Resources to Mineral Reserves

Measured and indicated mineral resources are converted into proven and probable mineral reserves by making appropriate provisions for dilution and mining recoveries.

4.4.5.1 Choice of Mining Method

A variety of mining methods are in use at Bougrine, and they generally fall into two methods, namely, open stoping and modified room-and-pillar or drift mining.

The drift mining or room-and-pillar mining (also locally termed "jumbo" mining) is generally undertaken for the initial development of the F2 orebody. This mining method, which has been chosen based on experience gained during the mining operation, has been considered in the reserve estimation. However, in the F2, sublevel (longhole) stoping with cemented or loose rock fill was the predominant method, which provided for post pillars in the wider parts of the ore. The previous operator generally used cemented fill but had not kept up with the fill requirements and had left a number of stopes open. Filling of mined-out voids since 1998 by

BWR was initially with unconsolidated rock fill but subsequently, and currently with cemented or uncemented fill depending on the requirement for mining adjacent to or below existing stopes. Some filled stopes have been successfully re-accessed. This will be an ongoing requirement to an extent.

The 31 December, 2000 mineral reserve estimate is based on the premise that the proportion of long-hole stoping will increase to 72% in the F2, mainly in the narrower, lower parts of the zone. The top cut and bottom cuts will be completely developed except for post pillars to be left in wider portions, and will be at vertical intervals of 30 metres (centre-to-centre). A drill drift half-way between the two sills will only be partly developed, sufficient to allow production drilling. Sill pillars will generally be recovered as part of the top cuts. Plans for mining of the western part of the F3 call for the mining of vertical slices between sub levels, with the use of cemented fill as required.

4.4.5.2 Dilution and Mining Recovery

For the mineral reserve estimate, dilution (both from waste rock and from rock fill) and mining recovery parameters were assessed for each mining area, separately for the individual mining activities such as drifting and stoping, and with due recognition of the planned mining method. The current reserve calculation for F2 assumes mainly unconsolidated fill. In contrast, most areas will now be mined using conventional fill. This is expected to lower dilution and improve recovery. The mining losses and dilution indicate that the overall tonnage in the mineral reserve estimate is essentially unchanged from the tonnage in the mineral resource estimate, while the zinc and lead grades in the mineral reserves are reduced to 80% of the resource grades.

4.4.5.3 Cutoff Grade

A minimum mining width of four metres was observed and a cutoff grade of 6% Zn-equivalent was used to estimate the resources, whereby 3.3% Pb is equivalent to 1% Zn based on NSR value. Low-grade mineralization (i.e. below the cutoff grade) is included in the mineral resource and reserve estimates, where required for mining continuity. Significantly increasing the cutoff grade (due to lower metal prices) would reduce continuity of the deposit, and as such, Bougrine plans to continue with the cutoff grade as currently applied, which results in the inclusion in the reserves (and in the production) of lower grade material (i.e. below cutoff grade) to assure mining continuity.

4.4.5.4 Reconciliation

Mine staff are confident that there is no over-reporting of the in-situ resources and reserves, and it appears on average that high grade stopes generally return a higher than expected grade and low grade stopes generally return lower than expected grades. The mine staff attribute this to the averaging of grades in the resource model during grade interpolation. Previously, metal reconciliation has been shown to vary substantially over a one-year period, where previous extraction and deterioration of ground conditions often led to high mining losses. However, the recent change in the mining method, in particular the use of consolidated fill, will help to eliminate unexpected, significant losses during mining. In 2001, mining in F3 against cemented backfill resulted in very low dilution rates.

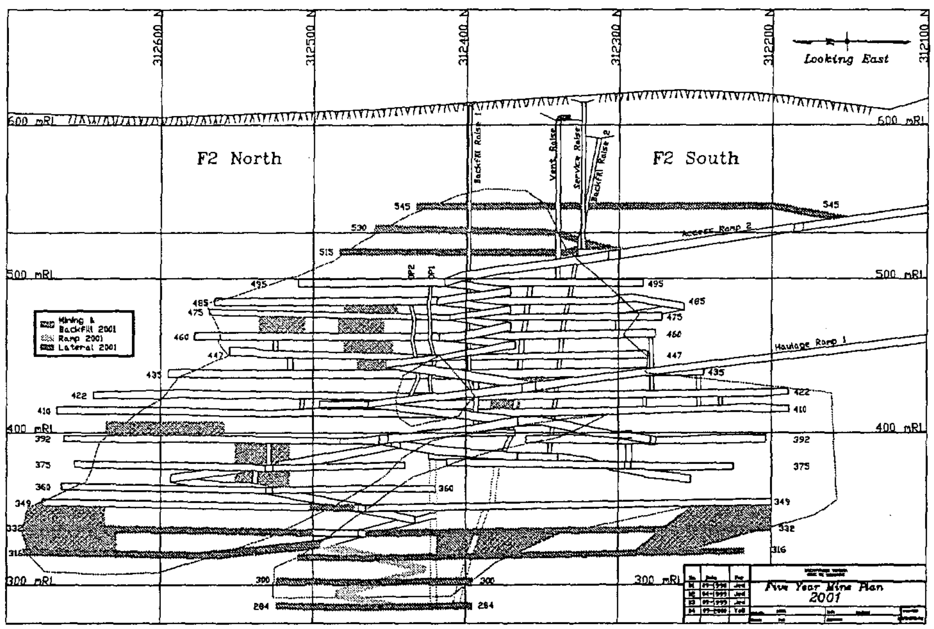
4.4.6 Mine Plan

The current mineral reserves at Bougrine allow for five years of production, to the end of 2005. The overall plan is to continue to mine the F2 and F3 zones, which can be blended to produce a reasonably consistent mill feed of approximately 12% Zn and 2% Pb.

In the short to medium term, it is expected that the mining department will continue to deliver the budgeted grade of feed that it does currently. Development to release more reserves has lagged budget in the past year, but only a limited amount of development is required over the next 1–1½ years to fully access the orebody.

The 640,000 tonnes of pillars included in the mineral reserves are scheduled for mining in the last two years, at which time the pillars will comprise 40 to 50% of the mine production. Although mining of the pillars may make it more difficult for the mine to achieve its production targets (since the recovery of these remnants are often affected by previous mining methods where unconsolidated fill may be adjacent to pillars), SRK remains confident of the mines' ability to meet the production plan (see Figure 4.3).

Figure 4.3: Bougrine Mine, Example Production Plan from 2001



4.4.7 Mineral Resource and Mineral Reserve Classification

The Bougrine mineral resources and mineral reserves are reported according to CIM Standards (August, 2000). The assignment to resource classes is adjusted to reflect the known local mining conditions.

4.4.8 Mineral Resource and Mineral Reserve Statement

Table 4.3 summarizes the mineral reserves and mineral resources of the Bougrine Mine as of 30 November, 2001, as estimated by a Qualified Person.

Table 4.3: Bougrine Mine, Summary of Mineral Resources and Mineral Reserves

	30 November, 2001			31 December, 2000		
	Tonnes (000s)	Zn (%)	Pb (%)	Tonnes (000s)	Zn (%)	Pb (%)
Proven and Probable Reserves (2)	1,684	10.94	2.1	1,975	11.20	2.1
Measured and Indicated Resources (1)	1,695	13.68	2.6	1,986	14.10	2.6
Inferred Resources	604	8.30	1.4	604	8.30	1.4

(1)Measured and Indicated Resources include Proven and Probable Reserves
(2)1 January, 2001 to 30 November, 2001 production includes 291,345 tonnes grading 12.7% Zn and 2.35% Pb from within reserve outlines, while an additional 53,078 tonnes grading 6.77% Zn and 1.15% Pb were mined outside of the reserve limits, and therefore, not subtracted from 31 December, 2000 reserves.

The reserves contain approximately 110,000 tonnes from the F1 zone; however, mining of approximately 10,000 tonnes during 2001 has indicated that the mineralization is considerably more structurally controlled than previously thought, and as such, some of these reserves may not be feasible to recover and a portion of the reserves may be converted into resources with further assessment.

The majority of the inferred resources occur within zone F1, totaling approximately 400,000 tonnes. Based on recent experience of mining in this zone, there exists only a low probability that these resources may be upgraded to reserves. However, the inferred resources in zone F2 have a much better chance of being converted into reserves with additional information.

4.5 El Mochito Mine

4.5.1 Introduction

One of the most characteristic features at El Mochito is the mine's ability to continually replace reserves, primarily from skarn deposits.

4.5.2 Data

The database for the mineral resource estimate consists of underground diamond drill holes, and to a lesser degree, chip samples and blast hole samples. In excess of 7,000 core holes and 10,000 percussion holes have been drilled, of which 42% were drilled in the last ten years alone, reflecting BWR's effort to replenish resources through mine exploration. The most recent drill holes (i.e. over the last few years) are stored in a Gemcom computer software program; however, much of the earlier data exists on sections and level plans.

As a result of the intensive diamond drilling in the past few years, most of the reserves have now been investigated on a nominal pattern of 15 x 15 metres. This includes the Nacional, SW Nacional, Canoe, Yojoa, Salva Vida, lower Port Royal and lower San Juan areas, all of which have also been extensively developed and are being exploited.

In contrast, the pillars of the Central San Juan, and the pillars and remnants of the Mina Vieja have received a great deal of chip sampling on the level openings developed for production drilling, but relatively little drill information is available between the levels.

Assaying of all grade control and drill hole samples is done at the mine assay office. A program of quality control was introduced in 1997 involving repeat assaying of pulps to check the precision of the lab, and the insertion of standards into the assay stream to check both precision and accuracy.

4.5.3 Mineral Resource Estimation

The mineral resources are calculated by BWR's Qualified Person using a traditional polygonal method on cross-section and then converted to cuts (for the current dominant mining method); basically averaging everything across the cut using all available geologic information, assay data and mine workings.

The mineral resources include continuous mineralization, typically above 3-5% zinc. Low-grade material is included as internal dilution where necessary to maintain mining continuity;

The available assay information is averaged to obtain a grade for the resource block. Declustering is often required to eliminate any bias of an above or below average grade that was over-represented from the data. Although BWR has not utilized grade capping, high-grade areas are frequently modelled separately in an attempt to restrict them to a realistic size and prevent over-estimation of the resource locally. In the absence of density data for individual assay intervals, grade averaging is by length-weighting.

SRK is in agreement with the methodology used to estimate the mineral resources.

4.5.4 Conversion of Mineral Resources to Mineral Reserves

It is important to recognize the relationship between the ore context and the mining method when classifying resources and converting those resources to reserves. What is critical is being certain that mining will take place in an area, and not precisely what an area contains. The current cut and fill mining method does its own definition and at the same time can be very selective, therefore if stoping is planned in an area it is prudent to convert these resources into reserves. The grade should be a function of the grade control, the layout of the method and un-mined waste. Therefore, the method provides a significant degree of flexibility on what grade is produced at the expense of recovery, suggesting that the mine should always be able to achieve planned tonnes and grade from the current reserves. This requires that there exists adequate advance waste development in order to access replacement ore.

4.5.4.1 Choice of Mining Method

The post-pillar, cut-and-fill mining method is ideally suited to the very variable geometry, grade distribution and rock conditions of the majority of the reserves and resources at the El Mochito Mine, and in the opinion of SRK, the method is well understood and well executed with good control over grade and dilution. The post-pillar cut-and-fill is designed on 15 metre centres with final 6 metre pillars leaving a nominal 16% in pillars.

4.5.4.2 Cutoff Grade

There is no official cutoff grade at present. The guide is 5.0% Zn with some recognition of the value from the other metals. BWR has previously used an incremental cutoff grade underground during day-to-day operations of 3.5% Zn as the mill is not operated at full capacity (it has potential for 2,100 to 2,200 tpd). It is suggested that it might be more appropriate to define cutoff grades on the cost to finished zinc, which would take into account the extra costs of mining in some more challenging areas and the cost of haulage and access.

4.5.4.3 Dilution and Mining Recovery

Internal dilution is included during the original construction of the polygons during resource estimation. External dilution is added by increasing the area of the cut polygon, as follows:

- 0.82 metres (2.5 ft) around the entire polygon for non-skarn mining (chimney deposits) with clear geological boundaries. A density of 2.8 tonnes/cubic metre is used for conversion to tonnes, and a grade of zero for all four elements.

- 0.33 metres (10 ft) for skarn mining with uncertain grade boundaries; grades for the skarn dilution are derived from appropriate intervals of nearby drill holes. Skarn dilution is assigned the same bulk density as the ore.

- Backfill dilution as 0.66 (2 ft) from the floor at a grade value estimated by the mill. The likely amount of backfill dilution to be expected depends on the mining method for that block. Grades used are the average mill tailings grades of 12 g/t Ag, 0.2% Pb, 0.5% Zn and 0.03% Cu. In a number of cases, "skin" pillars are excluded from the block as unrecoverable, and in these instances the fill dilution is set to nil. The resultant overall grade dilution is somewhat dissimilar for the different metals, reflecting different metal ratios in the ore and in the diluting skarn.

Finally, a mining recovery factor, which is essentially the ore left in pillars, is assigned to each block, depending on the mining method to be employed. This may include the provision for a "skin" of rock left behind in order to avoid excessive fill dilution.

4.5.4.4 Reconciliation

It has been difficult to reconcile from resource to reserve and from planned to actual over the years because of the large amount of production that has come from areas outside of the reserves. BWR uses this information to estimate extraction factors for resources/reserves that were previously outside of the reserves.

The reconciliation process includes:

- Comparison of volume broken to volume received based on bucket counts.
- Volume recovered versus ore outline (gains and losses).
- Reconciliation to the mill on grade with adjustments for whether sand-fill is being used or not.

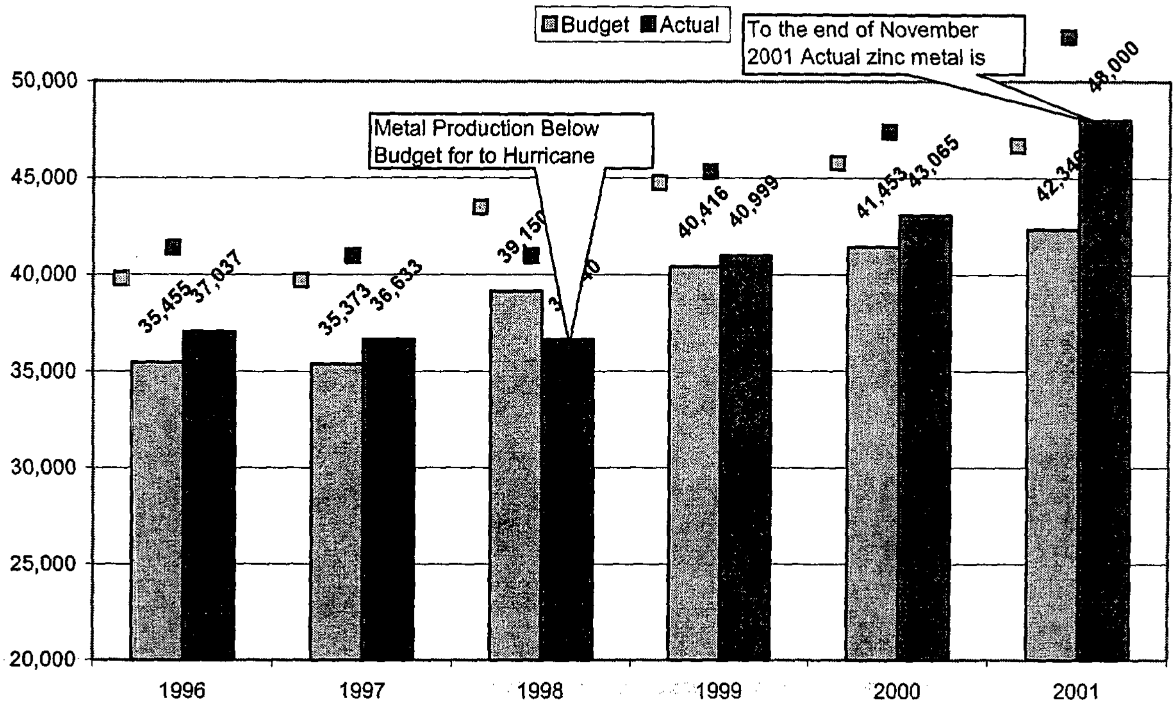
Generally the mill receives a higher grade than is being predicted and this may indicate that grades are being underestimated and possibly that some ore is being left in the stope prior to backfilling. BWR is currently modifying their reconciliation procedures in order to better define the extraction factors for the resources and reserves that were previously outside of the reserve boundaries.

There are concerns that there is an under-accounting of fill dilution. It is difficult to estimate due, again, to the large amount of mining outside of reserves. However, it may be a further indication that grade is underestimated and that dilution is being "hidden". There are plans to sample floors for ore and to better account for the amount of fill that is going to the process plant.

4.5.5 Mine Plan

BWR produces a 5 year mine plan on an annual basis. The 5 year mine plan was updated in the middle of 2001 based on the 31 December, 2000 reserve statement. However, BWR is currently following a higher grade strategy necessitated by the precipitate fall in zinc price consisting of selectively mining higher grade areas of the deposit (such as the very high grade "halo" areas around the sills in the San Juan deposit) and excluding typically low-grade areas referred to as "incremental" ore, which used to be processed if the mill had unused capacity. BWR has recently completed the 2002 mine plan and is currently finalizing the 5 year plan. SRK has confidence that this increase in grade can be sustained for at least 2002 but there are insufficient plans to assess how long a higher grade can be sustained or what would be the impact on mine life. In addition, SRK has a reasonable degree of comfort that the LoM Plan is much longer than is currently outlined, based on the update to the reserves that was done during the site visit. Figure 4.4 illustrates the mine's ability to achieve budgeted grades and tonnes.

Figure 4.4: El Mochito Mine, Zinc Metal Production



4.5.6 Mining Outside Reserves

Since 1995, 37% of the mill feed has come from outside the mineral reserves, due primarily to the extraction adjacent to reserve outlines, which were based on fairly conservative assumptions. In addition, mining in areas of inferred resources was also considered "mining from outside reserves" by BWR and is based on an overly conservative classification system. The cut-and-fill method allows BWR to follow shoots along the boundary, which on occasion has added very substantial resources. The 2002 plan is the first time that a forecast of mining outside of reserves has been used. In the past it was simply reported as something achieved. Based on SRK's recommendations, this consistent "over-extraction" has been included in the conversion of mineral resources to mineral reserves. There was considerable discussion between SRK and the El Mochito staff and BWR feels comfortable that they have sufficient certainty of the geology, continuity and grade to include this additional "extraction factor" into the reserves. SRK feels confident that these numbers are reasonable so long as the character of the deposits remains the same (which it would appear will be the case) and the current selective mining method is retained (which greatly reduces the amount of definition drilling and exposure that is needed prior to stoping).

Based on SRK's recommendations, BWR has added to the current mineral resource and mineral reserve base by including:

- Resources in the Palmar Dike and Santo Nino deposits.
- The halo ore left around previous high grade vertical crater retreat (VCR) stopes with an estimation of conversion factors.
- Upper portion of San Juan deposit.

- Area previously "condemned" adjacent to raise number 5.

4.5.7 Mineral Resource and Mineral Reserve Classification

The mineral resources and mineral reserves have been classified by BWR using CIM Standards (August, 2000) based on the following parameters:

Measured mineral resources are estimated using channel samples from exposures above and below with drill spacing on 15 metre centres. Indicated mineral resources do not have underground exposures; however, are based on drill data spaced at approximately 15 metres and where the geometry and the grade of the mineralization displays good continuity. There is no strict definition for inferred mineral resources other than a good understanding of the geological context and assumed continuity. In addition, scattered intersections or mineralized zones with less certain grade continuity are placed in the inferred category, with a nominal block of 15 by 15 metres usually assigned to a particular intersection. The inferred resource blocks at El Mochito constitute part of the exploration potential of the mine and serve as a reminder that more drilling and/or drifting needs to be done in those areas.

Generally, the definitions for measured and indicated resources are more suitable to the previously dominant method of open stoping, which required a much higher level of certainty on outline before stopes could be planned, Currently, the dominant cut-and-fill method uses its own extraction as a process of definition. As such, some of the inferred resources are transferred directly into reserves based on the mine plan.

4.5.8 Mineral Resource and Mineral Reserve Statement

Table 4.4 summarizes the mineral reserves and mineral resources of the El Mochito Mine as of 30 November, 2001 as estimated by a Qualified Person.

Table 4.4: El Mochito Mine, Summary of Mineral Resources and Mineral Reserves

	30 November, 2001				31 December, 2000			
	Tonnes (000s)	Zn (%)	Pb (%)	Ag (g/t)	Tonnes (000s)	Zn (%)	Pb (%)	Ag (g/t)
Proven and Probable Reserves	3,356	6.8	1.9	78	2,846	7.2	2.0	76
Measured and Indicated Resources	5,937	7.8	2.7	115	4,709	7.8	2.1	88
Inferred Resources	793	7.3	4.3	102	804	8.4	5.9	235

Measured and Indicated Resources include Proven and Probable Reserves

In summary, the large increase in the measured and indicated resources from 2000 has come from newly discovered resources through diamond drilling, resources from halos around old VCR stopes in the San Juan deposit, and resources and reserves from the "Extraction Factor".

4.5.9 Exploration

Over the years the El Mochito deposit has changed from being predominantly silver-rich to zinc-rich, generally having a lower NSR value. The considerable cost of exploring, or at least assessing, the potential endowment of the deposit has limited the work that could be done.

Anomalies that have been drilled by various companies in the past have been limited to short surface holes to look for vein systems. The original deposit was started on vein systems but currently is extracting the large bodies mostly below the Mochito shale. These previous surface holes were well short of where larger more interesting bodies might be located.

Also the El Mochito "system" was arbitrarily limited by faults that may or may not have been significant; this reduced the area of more intense exploration. There are many anomalies and indications of systems outside of what was called El Mochito.

SRK considers El Mochito to have excellent exploration potential. It is important to distinguish between pure exploration (which usually means finding other deposits) and near-field extensions (where the trends are very well understood and most of the work is to test extensions and define). It is very difficult to do pure exploration at El Mochito due to the large amount of development required (and the capital cost needed for this as well as expense of drilling from surface).

Near-field exploration is concentrated on following known structural controls on mineralization, which are:

- For decades, exploration has been driven by the Porvenir fault which confines the mineralization. This has now been closed off to the west but it is recognized that other trends do exist.
- The Nacional deposit follows a different trend more northerly and is open in both directions.
- The Port Royal is a very recent trend which trends even more to the north but dies out to the southwest. BWR has commenced exploration to the northeast (which had been avoided in the more distant past due to wet conditions).

There is also significant potential in other areas:

- Everything has so far been connected along the basement limestone so there is potential to connect deposits located above the basement and along the basement similar to existing manto deposits; the N/E Salva Vida at approximately 0.7 million tonnes is just one example (and this is currently in the indicated category).
- All of the major deposits have "roots" down to the basement; for the eastern deposits these roots have yet to be tested.
- Also, major deposits have appeared above the Mochito shale such as the San Juan so there may be extensions through the shale for some current mining areas.
- Finally there is often a "ballooning" out just below the Mochito shale that has been seen on some deposits, which again is an area with good potential for success.

BWR is currently investigating four or five chimney/manto systems. The average of the known systems that have already been mined, or are being mined, is 0.7 million tonnes, varying from 0.25 to 1.4 million tonnes. This gives a reasonable potential for adding to resources in the near term of 3 million tonnes. There are large areas with no drilling between these four or five systems with similar potential.

4.6 El Toqui Mine

4.6.1 Introduction

The estimation of mineral resources and mineral reserves at El Toqui is a two-stage process:

- First, all mineral resources are estimated for the main manto unit at a minimum width and cutoff grade based on metal prices, costs, recoveries and other considerations.
- Second, mineable mineral reserves are estimated by applying a series of factors and dilution to the mineral resources. The resource and reserve terminology at El Toqui reflects the room-and-pillar mining method.

4.6.2 Data

There is no concern identified with the quality of the database used for resource and reserve estimation. The amount of sampling information and the drilling density are appropriate for the purpose of resource and reserve estimation. Collar coordinates are well surveyed, supported in a recent geodesic survey carried out by an independent company.

During previous exploration and mining at El Toqui, BWR established and implemented a number of QA/QC programs to ensure the highest quality of data. Several independent audits, most recently by Alejandro Kakarieka in October, 2001, have been carried out to verify the quality of the data. In addition, SRK had an opportunity to review the results of ore reconciliations that have been completed at the mine since 1995. This reconciliation compared the resources/reserves with the mill production, providing an opportunity to verify the quality of the data.

All drill hole information is stored in a Gemcom software package.

4.6.3 Mineral Resource Estimation

The use of a polygonal-block-based method combined with regularly spaced drill holes, provides a reasonably accurate estimate of the mineral resources. Drill hole intercepts are plotted on plans and sections using Gemcom software. Circles of 25 metre radius and 50 metre radius are drawn around each drill hole intersection that exceeds the cutoff grade. Polygons are constructed where the circles from adjacent holes overlap. Polygons are generally constrained by faults, intrusives, dikes, and the boundary between barren and mineralized Main Manto unit. Geologic interpretation is based on underground mapping and drilling. SRK considers this to be an appropriate technique to estimate the resources.

Although the gold values for Mallin-Monica, San Antonio and Estatuas are typically too low of grade to be of economic importance, in the case of some areas within the Doña Rosa deposit the gold grades are assumed to be zero, which is a somewhat conservative estimate of the resources. A policy of systematic rock-chip sampling has to be introduced and implemented, especially because the future mine plan will bring into production areas with high gold content (Aserradero). It is suggested that gold assaying should be done at site, but with strong external checking to ensure accuracy and precision of mine assays.

4.6.4 Conversion of Mineral Resources to Mineral Reserves
4.6.4.1 Dilution and Mining Recovery

To convert mineral resources from mineral reserves, the following assumptions are made:

- Dilution is added to all resource tonnages at a rate of 10% at zero grade, which is based on a normal over-break of 0.2 metres on the roof and floor for a typical 4 metre high room.
- Room and pillar mining results in a general recovery factor of 80% before dilution. In the case of the Mallín-Mónica blocks and polygons and of the Doña Rosa blocks, the first pass recovery is set at 80% followed by a 50% recovery for the remaining pillars.

For pillars, mine geologists are applying a number of factors, depending on the mineral resource category (indicated, inferred) and the vertical section of the pillar (first or second bench), respectively. An indicated or inferred pillar is a theoretical pillar that really doesn't exist yet. A real existing pillar is a measured mineral resource. Currently, a 50% recovery factor is being applied to pillar recovery.

4.6.4.2 Cutoff Grade

The resources and reserves are based on a cutoff grade of 4.75% Zn equivalent, and although this does not appear to reflect current metal prices, there is only a marginal amount of the deposit between 4.75% and 6.0% Zn equivalent. During mining operations, BWR never exploits a face that is less than 6.0% Zn.

4.6.4.3 Reconciliation

Control of grade in room-and-pillar operations, especially where ore is very distinct from waste and breaks cleanly at a contact, is relatively easy. Visual estimation of grade is used and results correlate very well with mill receipts. Check assaying is done occasionally. Whether this visual estimation will work for the gold rich area of Aserradero remains to be seen. All faces are mapped for dilution estimation and control.

In the past, BWR has not done a formal reconciliation with the process plant and a redistribution of metal difference but they do have records of metal delivered and received, which are reasonably similar. This good agreement between reserve estimation, planning and actual delivery to the mill demonstrates the predictability of the resource and the mining achievements, which is not unusual in a room-and-pillar environment. SRK recommends that a more formal reconciliation be done in the future. BWR is finding that there is a good correlation between the planned metal and the actual metal.

4.6.5 Mine Plan

BWR prepares a 5 year mine plan on an annual basis. The latest 5 year mine plan indicates that at present production levels the current mineral reserves will support and maintain mine life beyond the 5 years of the plan. During the 5 year plan, reductions in production from Sectors Doña Rosa and Mallin Sur are offset by increases in San Antonio Este and Mallin-Monica. A modest amount of tonnage beginning in 2003 is scheduled from the only as yet undeveloped Sector Aserradero.

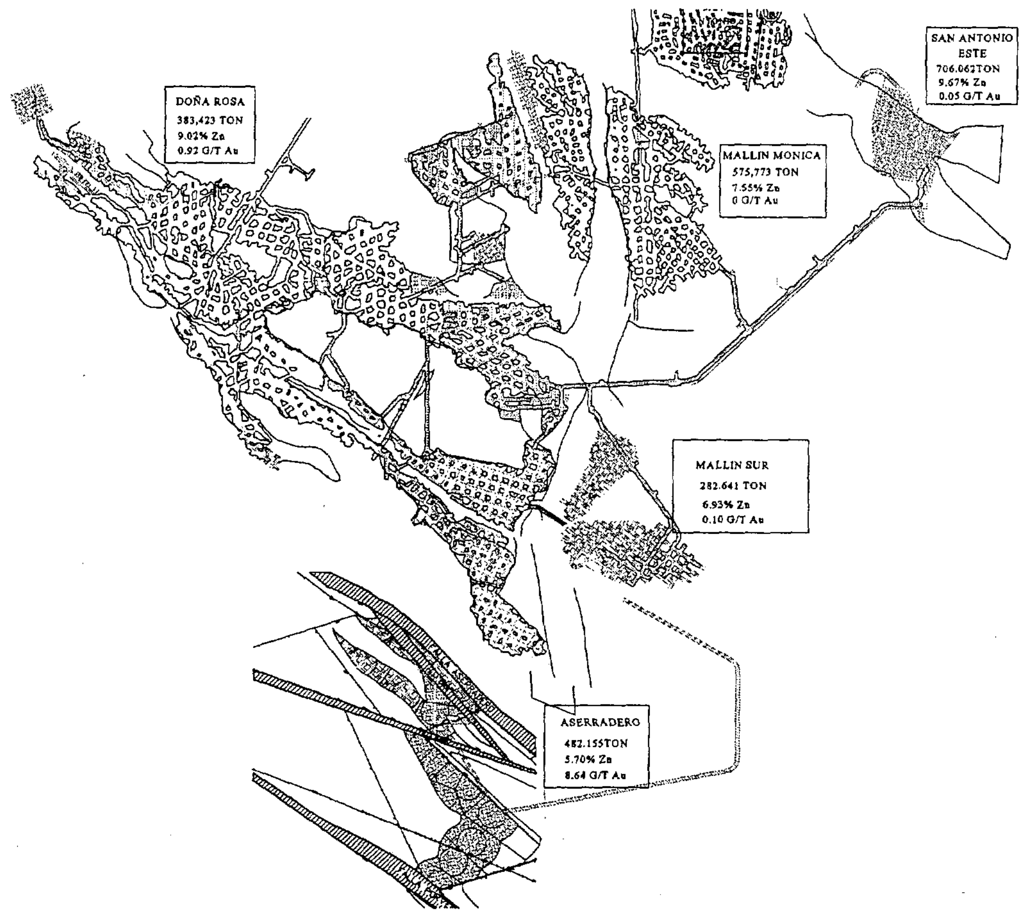
The current increase in grade is being supported by pillar slashing in the high grade Doña Rosa Mine, which is not possible in other deposits. The "pillar" reserves make up 25% of the metal over the next 5 years, which SRK considers acceptable in the short term but not over a period of 5 years, primarily because pillar recovery can only be adequately assessed once an area is very well known (i.e. most of the pillars have been formed and the back stability is well understood). BWR has had very good success with pillar recovery from well-known areas.

Key components to the success of the plan are:

- Timely production from Sector San Antonio Este contributing 38% of the planned zinc metal.
- Continued success in the slashing of high grade pillars in non-active areas of the Doña Rosa Mine accounting for 13% of the zinc.
- Increase in the amount of pillar production, from all areas, from a year-to-date average of 15% to 22% of the total tonnage.

SRK agrees that the 5 year plan demonstrates a logical sequence from the best-known material to the least known (presented in Figure 4.5). SRK has a high level of confidence that the 5 year plan can be achieved (and there has been a very close comparison of actual to budget over the last few years). The plan in most areas has extraction close to existing areas, which increases confidence.

Figure 4.5: El Toqui Mine, 5-Year Plan



Only the Aserradero sector might have marked differences from other areas as it is a very gold-rich zone, although in geological character it appears to be very similar to other areas. The Aserradero sector has not been exposed although it is in the 5 year plan. Currently, mining does not start until 2003. It has a gold grade of 8.7g/t but only 5.7% Zn. BWR is currently doing work with Lakefield Research Laboratories and Leslie Engineering to expand the mill capacity.

4.6.6 Mineral Resource and Mineral Reserve Classification

In summary, the classification system used by BWR is as follows:

- Measured mineral resource (subsequently converted to proven reserve) if within 10 metres of a face.
- Indicated mineral resource (subsequently converted to probable reserve) within 25 metres of a face and 25 metres from a drill hole.
- Inferred mineral resource (a portion of which was converted by BWR to "possible reserve" based on inclusion into the mine plan) between 25 metres and 59 metres of a drill hole.

Based on these definitions, BWR has not been able to avoid putting some of the inferred resources (termed "possible reserves" by BWR) into the mine plan; however, the correlation of the plans with delivery to the mill is very good even though large amounts of these "possible reserves" have typically been included. Based on the continuity of the grade and geometry of this mineralization SRK recommended that the "possible reserves" be re-classified as probable reserves based on CIM standards. Hence, BWR has re-classified these "possible reserves" as probable reserves.

In addition, SRK recommended and BWR completed:

- Unclassified resources for Concordia were reclassified as an inferred resource.
- A total of 536,243 tonnes grading 8.0% Zn and 1.7 g/t Au currently classified as "possible reserves" was reclassified as probable reserves.
- BWR reclassified the reserves for Estatuas as resources since it does not have a feasibility or a mining plan.

4.6.7 Mineral Resource and Mineral Reserve Statement

The 30 November, 2001 mineral resources and mineral reserves as reclassified by SRK are summarized in Table 4.5.

Table 4.5: El Toqui Mine, Summary of Mineral Resources and Mineral Reserves

	30 November, 2001			31 December, 2000		
	Tonnes (000s)	Zn (%)	Au (g/t)	Tonnes (000s)	Zn (%)	Au (g/t)
Proven and Probable Reserves	2,506	8.0	1.8	2,185	8.4	0.3
Measured and Indicated Resources	2,910	8.5	1.6	2,978	9.1	0.4
Inferred Resources	6,362	8.0	0.4	4,334	7.7	1.1

Measured and Indicated Resources include Proven and Probable Reserves

4.6.8 Exploration

During the past three years, BWR has carried out an aggressive exploration program in the El Toqui region focusing on the discovery of high-grade zinc mineralization.

The exploration program was designed to drill test a number of coincident base-metal soil geochemical anomalies that exhibited characteristics associated with known zinc deposits. Work conducted on the project included several phases of geochemical soil sampling, surveying, analysis and a detailed stratigraphic study. The work has outlined several prominent surface anomalies towards the west (Estatuas), the east (San Antonio-SE and Mallin-S) and the south (Aserradero- Doña Rosa-South) of the producing Doña Rosa Mine. The results to date confirm the presence of a very large and continuous zinc mineralized manto.

Recent exploration has intersected typical lithologies and grades, and as such, SRK believes that there is excellent potential to identify new zones of mineralization that could be added to the current resource base.

4.7 Langlois Mine

4.7.1 Introduction

The mineral resources and mineral reserves for the Langlois Mine occur within three separate zones, namely Zones 97 and 3, which host the majority of the resources and reserves, and Zone 4. The current mineral resources and mineral reserves for Zones 3 and 4 are essentially the same as the 31 January, 2001 mineral resources calculated by BWR, while the mineral resources for Zone 97 have been re-estimated by SRK incorporating the latest drill results (SRK – Langlois Feasibility Study, 2001). The current mineral reserves are based on a mining plan that was designed to optimize the economics of the mineral resources after examining various mining and processing scenarios.

4.7.2 Data

The database for the Langlois Mine consists of 1,954 surface and underground core holes, in excess of 2,000 underground chip samples and several thousand muck samples. The database includes the survey, assay, density and geological data for each drill hole and channel sample. The database is maintained at the mine site utilizing the Prolog computer software program, and has since been imported into the Gemcom software package by SRK. The current resources and reserves have been estimated based on 221 core hole intersections and 62 channel sample strings for Zone 97, 389 core holes and 113 chip samples for Zone 3 (main), and 45 and 49 core holes and numerous chip samples for Zones 3A and 4, respectively.

The majority of the drilling was completed prior to BWR acquiring the mine in 2000. Since that time, only a limited number of holes have been drilled in Zone 97. The majority of drilling is either BQ or AQ diameter core and is typically drilled along sections aligned north-south, intersecting the mineralized zones at as close to right angle as possible. The section lines are spaced at 20 metre intervals in Zone 97 and 10 metres in Zones 3 and 4. Both the drilling samples and chip samples were designed to cross the entire width of the massive sulphide mineralisation. Drill core recovery is typically greater than 95%. The drill core is stored on site in racks, with sample numbers marked for easy review.

Sampling and assaying methodologies and check assay procedures for drill core have been well documented by BWR. The procedure for the geologist is to describe the lithology, alteration, structure and other details and mark out and label the sample intervals and numbers on the core boxes. The sample intervals are designed not to cross lithologic boundaries or massive sulphide facies.

After reviewing the sampling and assaying procedures and the extensive QA/QC program implemented by BWR, SRK is confident of the quality of the data. Although verification sampling was not completed by SRK, numerous drill cores were examined to compare the recorded geology and mineralization with the assay grades. In addition, SRK had an opportunity to review the results of ore reconciliation that has been completed at the mine since 1995. This reconciliation compared the resources/reserves with the mill production, providing an opportunity to verify the quality of the data. SRK believes the quality of the data is good and that the sample preparation, analysis and security measures were carried out in accordance with best practice industry standards.

4.7.3 Mineral Resource Estimation - Zones 3 and 4

In general, BWR has calculated the mineral resources and mineral reserves utilizing two-dimensional polygons on cross-section, using the information from core drilling and underground chip sampling across development faces. Two-dimensional kriging and polygons on longitudinal section have also been used to a lesser degree, typically in areas where there is less available data. The resources and reserves are based on a minimum mining width of 3.0 metres.

Mine geology personnel extrapolate the zones of "potentially economic" sulfide mineralization from hole to hole only when sufficiently confident. Generally, the limit of the "ore grade", higher-grade zinc mineralization is congruent with the boundaries of the massive sulfide mineralization. In general, only regular ore shapes are delineated. Isolated zones of mineralization, areas of erratic grade, or areas of geologic uncertainty are not extrapolated, and are therefore not included in the resources/reserves until additional confirmation data is acquired. SRK considers this to be an appropriate approach.

Typically, drilling data is spaced at approximately 40-50 metres in Zone 97, except in the area adjacent to the underground workings on level 9 where drilling is spaced at 15-20 metres, and 15-20 metres in Zones 3 and 4. Although this drill density is considered adequate for resource estimation, it is not adequate for predicting the occurrence of numerous mafic dikes and late stage faults that transect the mineralization. It is BWR's usual practice to obtain a drill density of 15-20 metres prior to final stope design.

The numerous mafic dikes that crosscut the sulfide mineralization are too thin to be selectively mined, and therefore, are included in the mineralized zone for resource estimation.

The polygonal blocks were drawn to define regular shapes of sulfide mineralization as defined by diamond drilling and chip sampling of the underground workings. The polygons were drawn mid-point between two intersections. The grade of the polygon was calculated using the composited grade over the length that each drill hole occurred within the polygon. Although this portion of the resource is calculated manually, the resource estimates are detailed and managed very effectively.

There are two exceptions, Zone 3 between level 6 and 7 where two-dimensional ordinary kriging was used to estimate the resource based solely on the results of the surface exploration core drilling, and a portion of Zone 4 where two-dimensional polygons on longitudinal section were used, particularly along the perimeter of the mineralized zone. These methods were used due to the somewhat larger drill spacing. The resources estimated using ordinary kriging for a portion of Zone 3 have been compared with the results of the adjacent chip samples taken along levels 6 and 7 and correlate well. Any areas estimated using polygons on longitudinal section are classified as inferred resources (CIM Standards, August 2000).

4.7.4 Mineral Resource Estimation - Zone 97

Although BWR had estimated the mineral resources for Zone 97 as of January 31, 2001 using a polygonal methodology, SRK re-estimated the mineral resources primarily because:

- The resource for Zone 97 was initially estimated over a 3.0 metre minimum mining width, which masked the opportunity to determine the in-situ resource, and therefore, evaluate different mining methods such as reduced mining widths, different production rates, etc.

- Based on the relatively high variability of grade in Zone 97, SRK utilized ordinary kriging to estimate the resource, which provided a more appropriate amount of averaging of grades during interpolation, particularly with the larger drill spacing of approximately 40 metres.

- Kriging was also considered more suitable to Zone 97 than a polygonal method because of the pronounced trend of the zinc mineralization moderately down plunge to the east, which was evident on the grade contour plots on longitudinal section and confirmed by the very well defined curves on the semi-variogram (tool to measure the spatial continuity of grade).

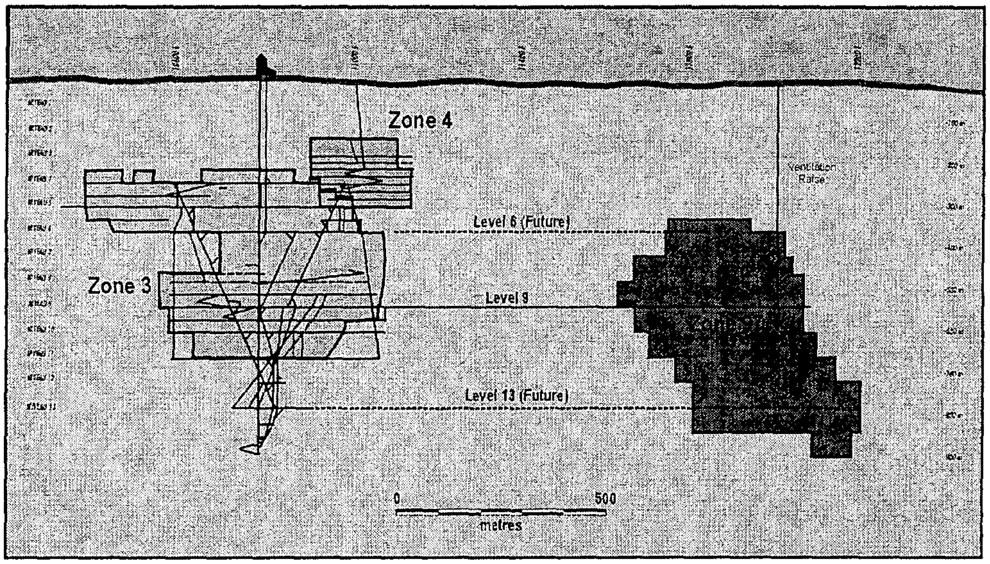
SRK then constructed a new resource model for Zone 97 using ordinary kriging to interpolate grades. In order to construct this model, SRK completed a preliminary review of the zinc grade distribution by completing a variographic analysis. Although the variograms differ somewhat by metal, they generally show the direction of maximum continuity to be moderately plunging to the east. The most characteristic feature of the mineralization is the moderately high local variability in grade, with the nugget value (measure of local variability) for zinc approximately 3/10 of the sill value (measure of variance), confirming that local predictions of grade would not be reliable and grade interpolation using kriging was an appropriate technique.

In order to complete the resource estimate, the assay values for each metal (Zn, Cu, Ag, Au) and density were composited using density and length weighting over the true thickness of the zone. Variography and kriging were completed on the product of the grade (including Zn, Cu, Ag, Au and the density) by the true thickness. This value per model block was subsequently divided by the true thickness to obtain the grade. In this way the true thickness and density are considered during grade interpolation, something not possible with cross-sectional interpolation. Capping of the zinc grades was not deemed to be necessary and is appropriately accounted for during kriging.

SRK had an opportunity to compare the results of the exploration and production core drilling and the underground chip samples for the same area/volume of the deposit in order to ensure that the chip samples and drill hole data have similar sample support, and could therefore, be combined and used for grade interpolation. The mean grades and the coefficient of variation, a measure of grade variability, compare very well indicating congruent sample support. In addition, any chip samples that were isolated and did not form a continuous string of samples across the mineralized zone were not used in the resource estimate.

There were also a number of historic holes, which have a number of missing gold assays. In these areas, BWR has typically used the average gold grades based on mill head grades, since gold does not have a definable correlation with any of the other metals. Fortunately, the majority of these holes lacking adequate gold assays occur in Zones 3 and 4 where there is production data to estimate the gold grade. Zone 97 has an adequate number of gold assays from drilling and underground sampling to estimate a reliable resource. Figure 4.6 presents Zone 97 in relation to the other zones.

Figure 4.6: Langlois Mine, Longitudinal Section (looking north) Showing Location of Zones 3, 4 and 97 and the Current Mine Development.



4.7.5 Conversion of Mineral Resources to Mineral Reserves

The mineral reserves were calculated by converting indicated or measured resources based on a minimum mining width of 3.0 metres for Zones 3 and 4 and 2.2 metres for Zone 97. The mineral reserves consist of contiguous zones of mineralisation delineated in the geological model, while isolated areas are not included.

4.7.5.1 Choice of Mining Method

Although the majority of resources and reserves are calculated over a 3.0 metre minimum mining width, several NSR cutoff grades have been utilized to determine the reserves, including NSR$60 for mineralization less than 2 metres in width (true thickness), NSR$55 for mineralization from 2.0 to 4.5 metres in width and NSR$45 for mineralization greater than 4.5 metres in width.

The reserves for Zones 3 and 4 are based on the mining method selected for developed and undeveloped areas. Where stope development already exists, stope dimensions will remain 30 metres in height. For the portions of Zones 3 and 4 that are not yet developed, sublevel spacing will be reduced to 20 metres. All stopes will be mined in a retreating sequence and will be accessible by ramp.

The current reserves for Zone 4 remain essentially the same as estimated January 31, 2001, with the exception of a pillar now being left behind. The reserves for Zone 3 are substantially different for the feasibility study, which consists of fewer tonnes but at a higher grade. This is primarily due to the fact that mining of the shaft pillar (higher grade portion of the Zone 3 resource) has been included in the mine plan. In addition, several of the relatively lower grade areas comprising Zone 3 have been excluded from the mine plan.

Zone 97 will be mined using an overhand benching method of stoping. Stope dimensions are planned at 9 metres high and, on average, 94 metres in strike length. The average ore width is approximately 3 metres.

Stope sequencing will be in a retreat fashion to the central access cross cut provided at each sublevel. The reduced stope height is designed to control dilution while allowing long, yet stable stope dimensions. The longer strike length provides for more continuous mining operations, such as drilling with less moving between work places. Ramp access is planned, as is backfilling with cemented paste fill.

Typically, drilling data is spaced at approximately 40 metres in Zone 97 except between level 8 and 9 (i.e. 60 metre level spacing), where the spacing is 20 metres. This is considered to be an adequate drilling density for delineating the zones of sulphide mineralization, however, this drilling is not sufficient to predict the occurrence of numerous mafic dikes that occur within the ore zones and late stage faults that transect the mineralization. Accordingly, BWR is currently planning a definition-drilling program for Zone 97, and for a portion of Zones 3 and 4 to obtain sufficient data density before final stope design.

4.7.5.2 Dilution and Mining Recovery

BWR has calculated internal dilution, and to some degree external dilution, during the initial construction of the two-dimensional sectional polygons, when regular shapes are drawn and the dilution is included in the composited grades. External dilution, which has been assigned a density of 2.75 t/m^3 and a grade of zero, is 25% by volume (approximately 18-20% by tonnage) for Zones 97 and 3. Due to the poor rock quality of the sheared and chloritic volcanic host rocks hosting the ore in Zone 4, a dilution of 35% by volume has been used to convert resources to reserves. Dilution in the past has typically come from "overbreak" in the volcanic host rocks. Other sources of dilution, although to a lesser degree, include late stage fractures that often displace the ore body several metres, causing sharp turns in the development drifts to follow the sulfide mineralization, and disintegration of ore passes that have constantly been a problem for dilution.

The ore recovery for Zone 97, excluding pillars is 95%, which is comparable to the historical recovery rates realized in Zones 3 and 4. Because of the highly variable width of mineralisation in Zone 97, SRK constructed a "skin or layer" adjacent to the mineralized zone to calculate dilution, rather than a global dilution factor that ignores geometry and assumes a constant dilution percentage over mineralized zones of variable thickness. A thickness of over-break equivalent to 35 centimetres off of each wall has been assumed with a minimum mining thickness of 2.2 metres. This constitutes an average dilution factor of approximately 30% by volume, which correlates well with the historical dilution for Zones 3 and 4.

4.7.5.3 Reconciliation

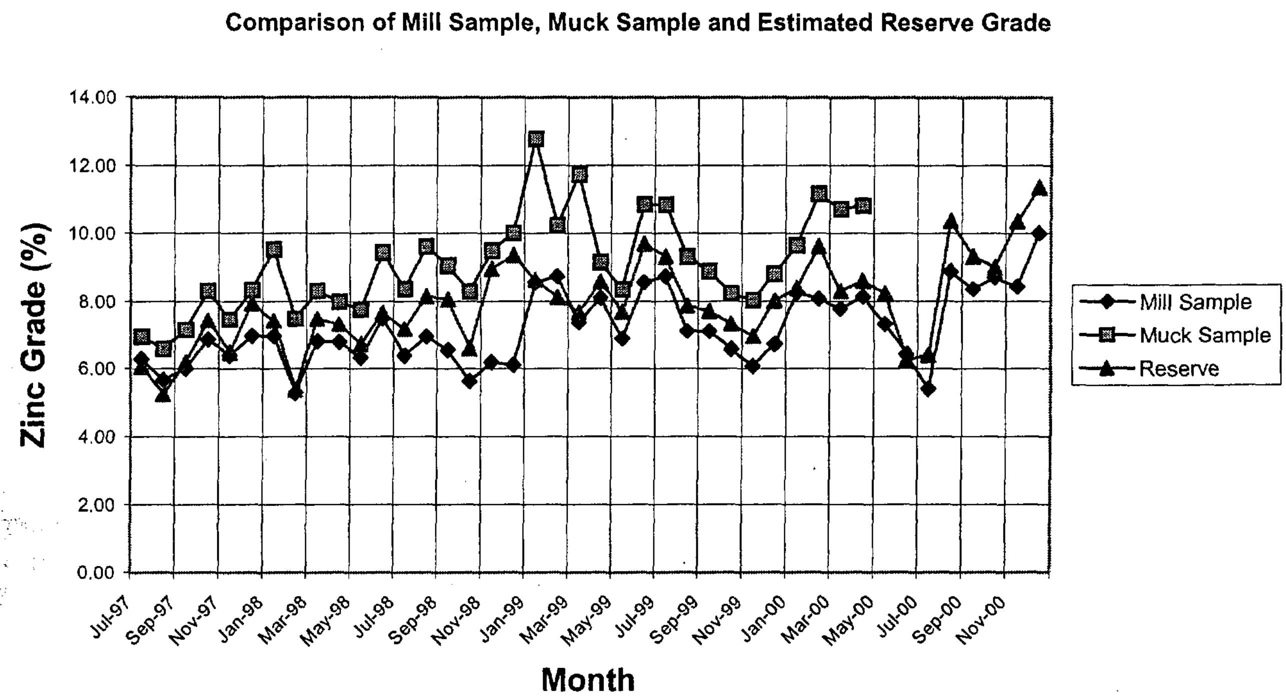
Since 1995, a detailed reconciliation of the ore reserves and mill production has been completed on the mined-out stopes. The reconciliation of the mining reserves with the mill shows an ore dilution of 30% by volume and a mining ore recovery of 98.7% for 2000 (based on a combination of various stope heights in various zones), which compare well with the dilution and ore recovery rates used to calculate the current reserves.

Based on the reconciliation of reserves versus mill production for 2000, BWR has calculated ore dilution to be 30% by volume with a mining ore recovery of 98.7% (based on a combination of various stope heights in various zones). The calculation of dilution based on volume is a more accurate determination of dilution than using zinc grade since the waste rock often contains lower grade zinc mineralization (i.e. 1-3% zinc).

Based on detailed volumetric examinations of mined-out stopes compared to the stated reserves from 1995 to

December, 2000, ore dilution has averaged 26.1% with an ore recovery of 88%. In addition, a detailed record of reserve grades, mill samples and muck samples have been kept to aid in the ore reconciliation (Figure 4.7).

Figure 4.7: Comparison of Langlois Mill Samples, Muck Samples, and Estimated Reserve Grade



4.7.6 Mine Plan

The mine plan currently includes the mining of ore simultaneously from Zones 97, 3 and 4. This will ensure that there will be a sufficient number of working places in order for the mine to support continuous production. In SRK's view, the current mine plan for the remaining reserves is logical.

4.7.7 Mineral Resource and Mineral Reserve Classification

The resource classification is based essentially on the density of drill hole and chip sample data and the continuity of zinc grade (since zinc accounts for the majority of the value of the deposit). Isolated areas of mineralization or areas that have currently no indication that they can be mined at a profit have not been included in the reserves.

Measured mineral resources, and hence proven mineral reserves, occur where the grade and geometry of the deposit is known with a high degree of confidence, allowing detailed mine design and mine planning to proceed. Measured resources are typically located in Zones 3 and 4, adjacent to underground workings, where the data spacing is typically less than 15 metres. The resources adjacent to the development on level 9 in Zone 97 were classified as proven reserves as of 31 January, 2001, however, they have now been classified as probable due to the results of the variographic analysis and the kriging variance.

In the opinion of SRK, the density of assay data and the knowledge of the geometry of the majority of Zones 97, 3 and 4, provide sufficient confidence to determine the grade and tonnage of the deposit. In addition, the continuity of grade and the geometry of the deposits are known with sufficient confidence to complete a mine design and mine plan. Although the actual location of planned stopes may vary somewhat during mining, this is not expected to significantly change the economics of the project. The indicated resources, and hence probable reserves, have been defined primarily within that portion of the deposit having a drill spacing of approximately 40 metres or less.

Inferred resources are located primarily along the perimeter of the deposit along the strike and down-dip extensions of the deposit.

4.7.8 Mineral Resource and Mineral Reserve Statement

The mineral resources and mineral reserves for the Langlois Mine occur within three separate zones, namely Zones 97 and 3, which host the majority of the resources and reserves, and Zone 4 (Table 4:6). The current reserves total 2.9 million tonnes grading 11.2% Zn, 0.7% Cu, 52.88 g/t Ag and 0.08 g/t Au, which consists of a lower tonnage with a higher average grade than the January 31, 2001 reserves totalling 3.9 million tonnes grading 10.2% Zn, 0.64 Cu, 49.04 g/t Ag and 0.08 g/t Au. The current reserves are based on a mining plan that was designed to optimize the economics of the resources after examining various mining and processing scenarios.

Table 4.6: Langlois Mine, Summary of Mineral Resources and Mineral Reserves

	30 November, 2001					31 December, 2000				
	Tonnes (000s)	Zn (%)	Cu (%)	Au g/t	Ag g/t	Tonnes (000s)	Zn (%)	Cu (%)	Au g/t	Ag g/t
Proven and Probable Reserves	2,903	11.2	0.7	0.1	53	3,892	10.2	0.6	0.1	49
Measured and Indicated Resources	4,862	11.0	0.7	0.1	53	3,942	11.8	0.7	0.1	47
Inferred Resources	1,547	8.1	0.5	0.1	37	1,111	9.6	0.6	0.1	35

Measured and Indicated Resources include Proven and Probable Reserves

4.7.9 Exploration

BWR's land holdings around the Langlois Mine include several mining properties that cover a total area of 5,836 hectares in 375 claims. These properties extend over more than 12 kilometres laterally and are underlain by the same stratigraphic rock sequence that hosts the massive sulfide zones at the Langlois Mine. The massive sulfide horizons at the Langlois Mine are elongated and narrow.

Extensive exploration works conducted by Cambior and its predecessor companies in the area located several scattered mineralized horizons that carry anomalous zinc values. These horizons are predominately located along the general elongated trends or parallel to the Zones 3, 4 and 97.

The "Grevet B" Zone is a narrow massive sulfide horizon that occurs near the surface some 1,000 metres south and 2,000 metres east of the mine working area. Ressources Metco Inc. completed a preliminary drilling program on the Grevet "B" property. Under an option agreement, Ressources Metco completed nine drill holes (1,558 metres) to assess the Grevet "B" deposit (220,000 tonnes at 8.82% Zn) as part of a feasibility study that was completed in June 2001.

There is a strong potential for discovery of other satellite lenses. In 1998, exploration drilling intersected sub-economic massive sulphides 300 metres west of Zone 97, between Levels 8 and 10. This sector is open down-plunge and should be drill tested when underground development on Level 13 is completed. Lens 4 has also the potential to host satellite lenses below Level 9. BWR is currently using lithogeochemistry to define a marker horizon and other favourable horizons for exploration.

4.8 Caribou Mine

4.8.1 Introduction

The mineral resources and mineral reserves for the Caribou Mine include the Caribou deposit and the Restigouche deposit. Both deposits have experienced mining on several occasions, open pit and underground mining at Caribou and open pit mining at Restigouche. The current resources and reserves at both deposits have been estimated by BWR's Qualified Person and independently audited on several occasions.

4.8.2 Data

The mineral resource estimate was created from a database of approximately 550 diamond drill holes and 6,000 chip samples and maintained at site using a combination of numerous section and level plans and in a Gemcom computer software program. The geology and mining departments have done an outstanding job in data collection and storage.

During previous exploration and mining at the Caribou and Restigouche deposits, BWR established and implemented a number of QA/QC programs to ensure the highest quality of data. Several independent audits have been carried out, most recently by Roscoe Postle Associates Inc. (RPA), 1999, to verify the quality of the data. In addition, SRK had an opportunity to review the results of ore reconciliation that has been completed at the mine since 1995. This reconciliation compared the resources/reserves with the mill production, providing an opportunity to verify the quality of the data.

4.8.3 Continuity of Mineralization

The massive sulfide horizons for each of the 6 lenses were extrapolated from drill holes to underground workings and include contiguous zones of mineralization. Within these outlines, BWR rigidly applied a 10.0% Pb+Zn block grade cutoff over a minimum true width of 3.0 metres (Luff, W.; 31 December, 1998). Subsequently, a revised mineral resources evaluation was undertaken using a 9.0% Pb+Zn block cutoff grade over 3.0 metres minimum width. This revised calculation was performed by taking the main blocks of tonnage at 10% Pb+Zn block cutoff grade, and checking for supplemental tonnage beside or along strike of the section used in the preliminary calculation. This additional resource, typically grading over 7.0% Pb+Zn, was included.

4.8.4 Mineral Resource Estimation

For the Caribou deposit, resource grades were calculated using traditional polygonal methods on level plans utilizing data points and measuring areas of influence intimately connected to those data points. In this manner, a chip sample face will have an area of influence half way to the previous face and forward half way to the next face. An ore outline is created using the grades deemed most economic across the sample faces and are weight-averaged, including any low grade material between higher grade intervals. The calculated grade, with area of influence, is used to calculate a weighted average grade over strike length of block or sublevel. The graded values along strike have an area of influence half way to the next sublevel, both above and below. Volumes and then tonnage can be generated, with weighted average of the mining block.

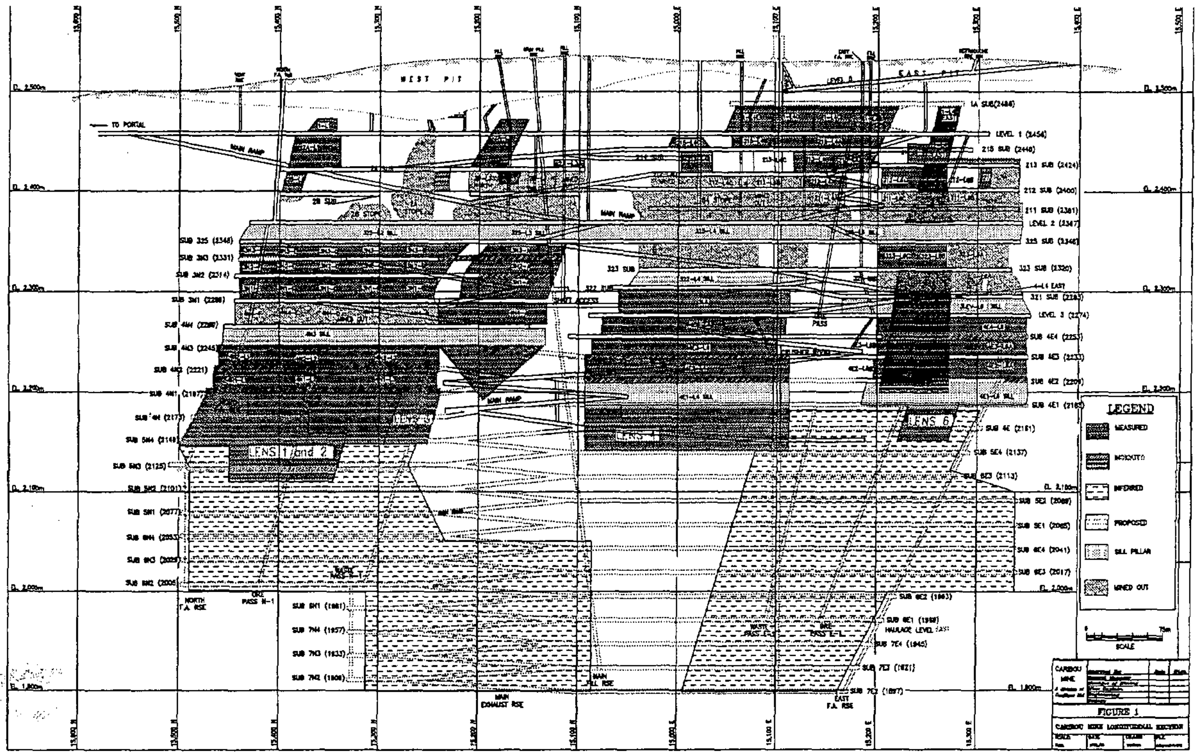
Diamond drill holes for blocks are incorporated similar to chip assays. A combination of both chip and diamond drill assays were used to calculate blocks. Chip assays were used over drill assays in most cases, but results were compared in several blocks. If chip assays were missing silver, these were assigned from diamond drill results. Inferred resources below the 2,200 metre elevation were calculated by the polygonal method using 9.0% Pb+Zn block cutoff grade. Each mineralized diamond drill hole intercept was calculated using the individual lead, zinc and silver assay interval grade. Each lead, zinc and silver assay grade was weighted over its sampled interval. A density of 4.27 t/m^3 for ore and 2.80 t/m^3 for waste was used.

Table 4.7 shows that the resources are very sensitive to cutoff grade. It also shows that the 10% Zn+Pb cutoff grade was not a true cutoff grade as the grade of the incremental tonnage added to the resources is of much lower grade than the 9% Zn+Pb cutoff grade. It is recommended that the re-opening plan critically consider that these lower grade resources should not be mined if below cutoff grade, unless there is excess capacity in the mill or required for mining continuity. A longitudinal section of the Caribou Mine is presented in Figure 4.8.

Table 4.7: Caribou Deposit, Mineral Resources at Different Cutoff Grades

	Tonnage	Zn	Pb	Ag
1998, 10% Zn+Pb	2,800,000	8.03	3.66	100
1999, 9% Zn+Pb	3,800,000	7.49	3.26	92
Difference	**1,000,000**	**5.96**	**2.12**	**70**

Figure 4.8: Caribou Mine, Longitudinal Section



For the Restigouche deposit, the open pit portion has been drilled on 15 metre centres. Neil D.S. Westoll & Associates Ltd. (Westoll, 1990) prepared the first resource estimate, based on a sectional polygon method. This was followed in 1995, by a resource estimate completed by RPA using a three-dimensional block model of the deposit. Both estimates were checked in the Kilborn Inc. (Kilborn) study of 1995 and found acceptable to evaluate the project. Another review was performed by Bharti Engineering in 1996, followed by an assessment of the pit wall design by Blackwell in 1998.

The mineral resource of the open pit was re-evaluated as of 31 December, 1998 by W. Luff of BWR. The resources were calculated by kriging on a block model using computer software (E. Puritch, July, 1997, final pit design). The underground mineral resources were taken from the 1995 Kilborn study while the inferred resources were taken from the 1990 Micon International study.

The parameters in the estimate of mineral resources from the open pit were:

- Specific gravity – 4.40 t/m^3 for ore; 2.8 t/m^3 for waste.
- Cutoff grade - 8% Pb+Zn.
- Database – 236 diamond drill holes.
- Block size in model – 5.0 metres by 5.0 metres by 2.5 metres.

BWR had J. W. Hendry Engineering Inc. review and check these estimates, and recalculate the tonnage of all +8% Zn+Pb grade ore using the set of geology cross-sections that were attached to Westoll's report. This estimation produced a total in-situ tonnage of approximately 1,169,000 tonnes. In order to check the average grade, the engineer calculated the statistical mean of the +8% Zn+Pb assay composites using RPA's computer database. The resulting average grades were 7.68% Zn, 6.15% Pb, and 135 g/t Ag. All of these values are very similar to the results obtained in the Westoll and RPA calculations and it was concluded that the RPA model was an acceptable basis for evaluation of the project.

4.8.5 Conversion of Mineral Resources to Mineral Reserves – Caribou Mine

The mineral resources for the Caribou and Restigouche mines were calculated as at 31 December, 1998 by BWR staff. The resources were calculated according to the CIM published guidelines using existing diamond drill hole data and chip sample assays..

4.8.5.1 Dilution and Mining Recovery

Measured and indicated blocks are converted to proven and probable reserves. Tonnage is added by multiplying by 113.2% and then multiplying by 90% for mine recovery in stopes and 75% for mine recovery in sill pillars. The 325 sill pillar below Lens 2 and 3 is deemed 0% recovery, due to ground conditions. Grades are diluted 13.2%, by multiplying grades by 86.8%.

4.8.5.2 Reconciliation

Based on reconciliation from surveyed stopes, tonnages and grades from the mine were compared with the mill production, as presented in Table 4.8. The dilution estimate of 13.2% used in the ore reserves reflects the actual dilution to date.

Table 4.8: Caribou Mine, 1998 Reconciliation

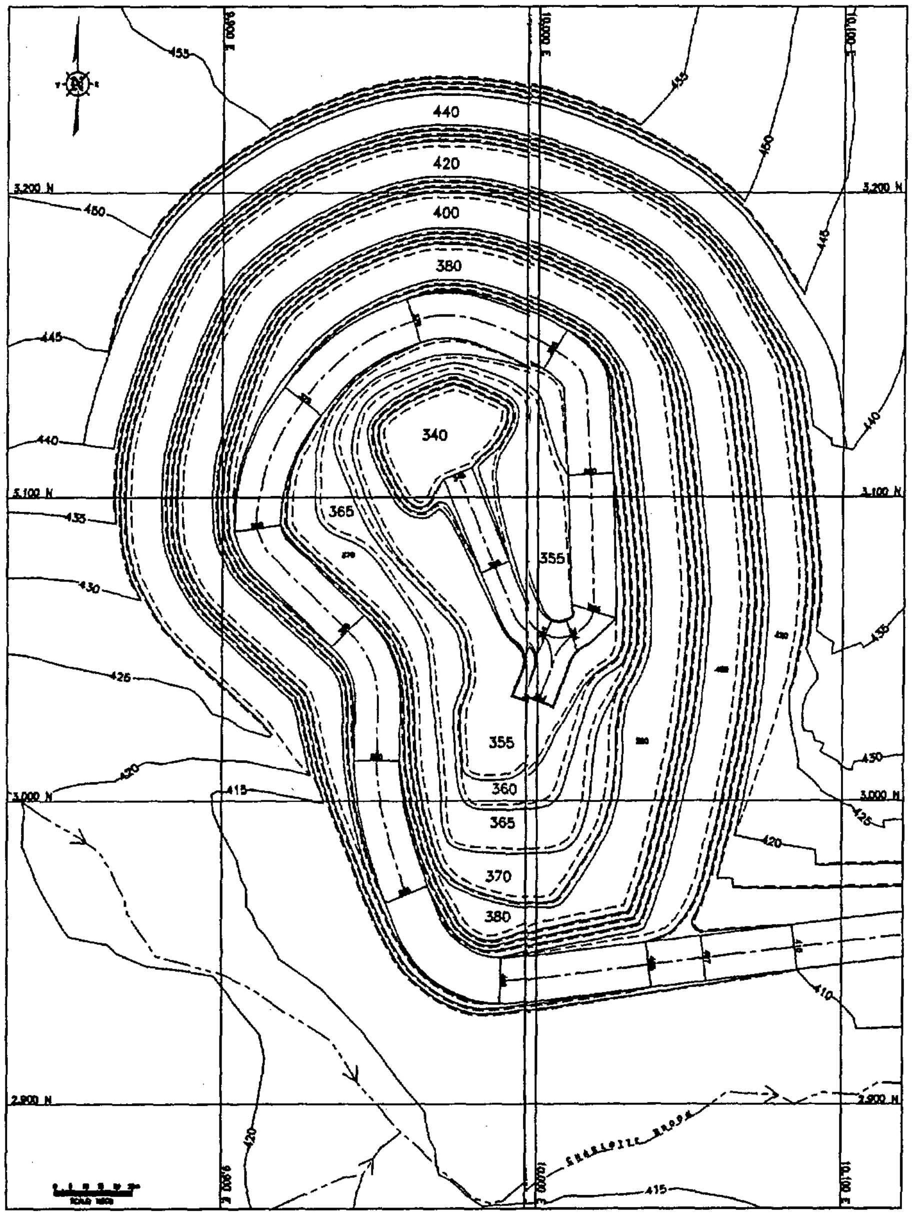
Type	Tonnes	%Pb	%Zn	Ag g/t	% Dilution
Mined	325,063	3.02	6.89	84	13.2
Milled	320,170	3.13	6.48	87	

Reconciliation used to predict dilution and recovery is somewhat suspect as the zinc grade is lower in the mill than the mine forecast; however, the lead and silver grades are higher in the mill than mine figures.

4.8.6 Conversion of Mineral Resources to Mineral Reserves - Restigouche Deposit
4.8.6.1 Dilution and Mining Recovery

The mineral resource can be converted to mineral reserve directly. Dilution was estimated to be 10%; resource grades were multiplied by 0.90 to convert to mineral reserves. Resource tonnage was increased by 10%. Mining recovery is expected to be 100%. Some of the material surrounding the pit resource is low grade (4 to 6% Pb+Zn), so some of this may be taken with ore. This accounts for the high recovery factor and the low dilution estimate. An open pit design for the Restigouche deposit is presented in Figure 4.9.

Figure 4.9: Restigouche Pit, Mine Design



The underground reserves are taken from the 1995 Kilborn study. The parameters are the same as for the open pit, except for dilution, which was at 10%, but dilution material was expected to include half the material at 50% of ore grade and the remaining 50% would be at zero grade. The last assessment of the underground resources at Restigouche was by Westoll in 1990. The total resource was estimated at 691,000 tonnes, of which 495,000 tonnes (65%) was thought to be mineable. The reserve at the time was called probable. As the deposit is well drilled at 30 metre centres and the recovery factor is low, the underground reserve has been classified as proven. The figure of 495,000 tonnes is mentioned in the Kilborn study of 1995.

4.8.6.2 Reconciliation

The dilution estimate of 10.0% is close to actual of 10.6% for 1998, as presented in Table 4.9. Higher dilution was observed when the top part of the 390 bench was mined and the ore zone narrowed.

Table 4.9: Restigouche Deposit, 1998 Reconciliation

Type	Tonnes	% Pb	% Zn	Ag g/t	% Dilution
Mined	176,943	5.18	6.66	124	10.6
Milled	174,279	5.36	6.26	129	

4.8.7 Mineral Resource and Mineral Reserve Classification

BWR classified the mineral resources and mineral reserves based on CIM, 1996 definitions and separated them into measured, indicated, and inferred mineral resources and proven and probable mineral reserves. Using CIM Standards (August, 2000) SRK agrees with the current resource and reserve classification. Measured resources are primarily contiguous zones of mineralization intersected by at least one sill, supported by close-spaced drill holes, usually 30 metres or less. Indicated resources are based solely on drilling (spaced 30 metres to 60 metres). The indicated resources are blocks that are extensions of measured blocks or separate blocks.

Inferred resources occur below the 2,200 metre elevation.

4.8.8 Mineral Resource and Mineral Reserve Statement

The mineral resources and mineral reserves for the Caribou and Restigouche deposits as of 30 November, 2001 have been estimated by BWR's Qualified Person and classified using CIM Standards (August, 2000) and are presented in Table 4.10.

Table 4.10 : Caribou Mine, Summary of Mineral Resources and Mineral Reserves

	30 November, 2001				31 December, 2000			
	Tonnes (000s)	Zn (%)	Pb (%)	Ag g/t	Tonnes (000s)	Zn (%)	Pb (%)	Ag g/t
Proven and Probable Reserves	5,057	6.5	3.4	90	5,057	6.5	3.4	90
Measured and Indicated Resources	5,152	7.4	3.9	95	5,152	7.4	3.9	95
Inferred Resources	4,163	6.7	3.2	98	4,163	6.7	3.2	98

Measured and Indicated Resources include Proven and Probable Reserves

30 November, 2001 Measured and Indicated Resources represent correction to a previous tabulation error. 31 December, 2000 Measured and Indicated Resources were previously stated by BWR as 5,264,000 tonnes.

4.8.9 Re-Opening Plan (1999)

A BWR re-opening plan was developed in 1999 and was updated in 2000. The plan calls for the production of ore from the Caribou underground mine at the rate of 1,650 tonnes per day and from the Restigouche open pit mine, at the rate of 1,350 tonnes per day. The reserves from the open pit will be depleted in 21 months and the underground reserves accessed from the open pit will be developed in time to produce 500 tonnes per day. At this time, production from the Caribou underground will be increased to 2,500 tonnes per day.

The Restigouche deposit will be mined in two stages. The first stage will be the resumption of the open pit operation with the second stage being the development of the underground deposit commencing immediately upon depletion of the open pit reserves.

For the purposes of mine planning and cost estimation, inferred mineral resources were estimated to the 1,900 metre elevation, then converted into possible mineral reserves using the same dilution and mining recovery. Table 4.11 presents the mineral reserves used in the re-opening plan at the Caribou Mine.

Table 4.11: Caribou Mine, Resources and Reserves in the 1999 Re-Opening Plan

	Proven & Probable Reserves				Inferred Resources			
	Tonnes	**Zn%**	**Pb%**	**g/t Ag**	**Tonnes**	**Zn%**	**Pb%**	**g/t Ag**
Caribou	3,725,996	6.48	2.83	80	2,204,389	5.99	2.90	95
Restigouche	1,333,108	6.53	5.05	100	-	-	-	-
Total	**5,059,104**	**6.49**	**3.41**	**85**	**2,204,389**	**5.99**	**2.90**	**95**

Although SRK does not believe that the following recommendations would materially change the current resources and reserves, they should be considered in any re-opening plan:

- Confirm gold grades by zone (i.e. typically higher in zones 1, 2 and 3).
- Dilution and mining recovery should not be assigned globally but by mining method, geometry and zone.
- Use NSR value for cutoff grade as opposed to Zn + Pb cutoff grade as the economics are different for these two metals and the ratio varies across the deposit.
- Better define costs and therefore cuoff grade (i.e. 8% cutoff grade for the open pit and underground mines at Restigouche).
- Clear determination of cutoff grade to ensure that lower grade material (below the cutoff grade) adjacent to higher grade mineralization is not mined and processed other than for mining continuity or because of excess capacity in the mill.

4.9 Consolidated Mineral Resources and Mineral Reserves

Table 4.12 presents a consolidated statement of mineral resources and mineral reserves for BWR's mining assets, and provides a comparison with end-of-year 2000 mineral resources and mineral reserves.

Table 4.12: Consolidated Mineral Resources and Mineral Reserves

for BWR's Mining Assets

	30 November, 2001						31 December, 2000					
	Tonnes (000's)	Zn (%)	Pb (%)	Cu (%)	Ag (g/t)	Au (g/t)	Tonnes (000's)	Zn (%)	Pb (%)	Cu (%)	Ag (g/t)	Au (g/t)
PROVEN AND PROBABLE RESERVES												
Bouchard-Hébert	3,219	4.9		0.6	33	1.0	4,414	4.9		0.7	36	1.1
Nanisivik	772	7.4	0.4		30		2,868	6.9	0.4		28	
Bougrine	1,684	10.9	2.1				1,975	11.2	2.1			
El Mochito	3,356	6.8	1.9		78		2,846	7.2	2.0		76	
El Toqui	2,506	8.0				1.8	2,185	8.4				0.3
Langlois	2,903	11.2		0.7	53	0.1	3,892	10.2		0.6	49	0.1
Caribou	5,057	6.5	3.4		90		5,057	6.5	3.4		90	
Total	**19,497**	**7.6**					**23,237**	**7.5**				
MEASURED AND INDICATED RESOURCES												
Bouchard-Hébert	3,402	4.9		0.6	33	1.0	4,665	4.8		0.7	36	1.1
Nanisivik	2,012	7.2	0.4		30		4,152	6.3	0.4		24	
Bougrine	1,695	13.7	2.6				1,986	14.1	2.6			
El Mochito	5,937	7.8	2.7		115		4,709	7.8	2.1		88	
El Toqui	2,910	8.5				1.6	2,978	9.1				0.4
Langlois	4,862	11.0		0.7	53	0.1	3,942	11.8		0.7	47	0.1
Caribou	5,152	7.4	3.9		95		5,152*	7.4	3.9		95	
Total	**25,970**	**8.4**					**27,584**	**8.2**				
INFERRED RESOURCES												
Bouchard-Hébert	-						-					
Nanisivik	110	4.6	0.4		30		359	5.1	0.4		19	
Bougrine	604	8.3	1.4				604	8.3	1.4			
El Mochito	793	7.3	4.3		102		804	8.4	5.9		235	
El Toqui	6,362	8.0				0.4	4,334	7.7				1.1
Langlois	1,547	8.1		0.5	37	0.1	1,111	9.6		0.6	35	0.1
Caribou	4,163	6.7	3.2		98		4,163	6.7	3.2		98	
Total	**13,579**	**7.6**					**11,375**	**7.5**				

Measured and Indicated Resources include Proven and Probable Reserves
*Previously stated by BWR as 5,264,000 tonnes

5. MINING

5.1 Introduction

This section includes discussion and comment on the underground mining methods applied at the Mining Assets.

5.2 Bouchard-Hébert Mine

At Bouchard-Hébert, a mechanized longhole mining method is used with the sequenced mining of primary and secondary stopes. Paste backfill is used to fill the voids. Production drilling uses 165 millimetre diameter blast holes and the stope heights are 60 metres. Drilling is done on a 3.2 metre by 3.45 metre pattern. Blasted material is removed using Toro 450 scooptrams with remote control.

The ore-pass system is used to direct the hauled mineralization from the stopes to the underground jaw crusher. The blasted material below Level 9 is hauled by trucks to the grizzly that feeds the ore-pass system feeding the crushing system on Level 9. A 519 metre inclined conveyor transports the crushed material to a 3,000 tonne capacity bin ahead of the shaft loading pockets.

Access to the Bouchard-Hébert Mine is by a 747 metre deep, three-compartment production/service shaft. A single hoist is used for both production and service hoisting. The skips used to hoist the crushed material to surface are 11-tonne capacity, bottom-dump units. A 15-man capacity cage is located below the skip to transport personnel and materials underground.

Stope preparation involves silling out the drilling and mucking levels over the whole width of the mineralized body, and rockbolting the back and hangingwall side. A 1.07 metre bored raise serves as a slot raise. The primary stopes are 15 metres wide by the thickness of the mineralized body and are mined vertically, two stopes at a time, for two levels, and backfilled before the secondary stopes between the primary stopes are mined. The secondary stopes are 20 metres wide. All the levels are accessible from Level 8 with a ramp access into the 1100 Lens. Remote scoops are used to remove the broken mineralization from drawpoints at the bottom of the stopes. Multiple drawpoints are used in the primary stopes and single drawpoints are used in the secondary stopes. An assay cutoff rather than a change in lithology determines stope limits.

All blasting is carried out using an ANFO blasting agent. Central blasting is used. Blasting is done in 15 metre slices, with the stopes kept as full as possible in order to reduce the wall dilution. When a sufficient void exists, the next cut is removed. The reamed slot hole is in the centre of the stope in order to avoid blast damage to any of the walls. The core of the stope is blasted and then the walls are blasted in a subsequent blast. The slot is not taken through to the level above. The sill and the slot are blasted in a 15 metre cut. Blasts up to 50,000 tonnes in size have been made successfully.

Development at the Bouchard-Hébert Mine is carried out by electric-hydraulic drill jumbos and scooptrams. The average drift size is 3.7 metres by 4.3 metres.

The mine infrastructure consists of seven mining levels spaced at 60 metres. The maintenance garage, pumping station, fuel and oil storages and main explosives magazine are located on the main mine Level 8.

The ventilation network consists of four 400 hp fans on surface, a 3.0 metre diameter fresh air raise at the eastern end of the mine, and a 2.4 metre exhaust raise at the west end. Part of the fresh air raise serves as an emergency manway from Level 10 to the upper part of the 1100 lens. From there, the main zone ventilation raise is used to access surface. Two fans are used in winter to supply 285,000 cfm, and all four fans are used during the summer to supply 330,000 cfm.

The paste backfill plant was designed to meet the mine's backfill requirements. It uses a backfill preparation process developed by Inco's research centre. The process involves thickening the flotation circuit tailings and storing them in a tank that feeds disk filters. The filtered tailings are stored in a hopper, then sent to a high-intensity mixer where cement, fly ash and water is added to produce a homogenous paste containing over 80% solids. The paste backfill is fed underground by gravity.

SRK considers the mining methods appropriate, and believes that the LoM Plan and associated production schedule are achievable.

5.3 Nanisivik Mine

The Main Lens being mined at the Nanisivik Mine is flat lying and outcrops on surface at both ends of the zone. The geometry of the mineralized body and the presence of permafrost permit large underground excavations and the use of large scale mining equipment.

In the Main Lens, where the dimensions of the mineralized zone are up to 150 metres in width and 20 metres in height, the primary mining method is room and pillar. This zone has been largely mined out, but post pillars remain that will be recovered. First pass mining recovery has been approximately 80%. For the other zones adjacent to the Main Lens, the mining methods are drift and slash stoping with some cut and fill stoping. The same mining equipment is used in these zones as in the Main Lens.

Stope backfill consists of low grade sulfide material sourced from surface stockpiles, development waste or shale from surface. Water is applied to the surface of the backfill which freezes generally within two days to form a strong working surface and also reduces dilution.

At Nanisivik, the permafrost contributes to mine stability. Care has to be exercised to remove sulfides from the roof, which can fail over time. The usual practice is to mine to the dolostone hanging wall contact. Rockbolting is systematic using 1.8 metre split set rock bolts on a one metre by one metre pattern. Screen is used in areas with high backs or poor ground.

In the Main Lens, recovery of pillars is being carried out. Operational experience has shown that the pillars can be recovered without major ground failure. Beginning in September 1997, scheduled pillar recovery began following strict guidelines. As of 30 November, 2001, 211 post pillars have been removed for a total tonnage of 462,226 tonnes at a grade of 9.0% zinc. To ensure the safety of the mining personnel, all pillars are removed in a sequential retreat manner utilizing remote control equipment. As a further precaution, a ground movement monitoring system utilizing remote monitored extensometers has been installed throughout the active mining areas.

Nanisivik is a unique mine in that dry drilling can be used due to the permafrost. The rock temperature is minus 12°C and the silica content of the rock is low. Respirable dust limits are maintained below 5 mg/cm3 using Atlas Copco DCT 160 dust collectors on all drilling equipment except the diamond drills. The drilling fleet consists of three Jarvis Clark MJM 21 two boom jumbos equipped with Tamrock C550 drills.

A Jarvis Clark ANM-12 ANFO loader is the primary equipment for loading explosives. Most blasting at Nanisivik is done using ANFO, non-electric delay detonators, and detonating cord, fired with an electric cap. The ammonium nitrate prills are mixed with fuel oil on site. Each blast is initiated from the mine office by a central blasting system after the mine has been evacuated at the end of each shift.

The primary mucking fleet at Nanisivik consists of two Cat 980 Loaders, a Wagner ST5 Scooptram and an Elphinstone 8 yard scooptram. Haulage trucks include three DJB D330 30 tonne trucks, one Wagner MT439 35 tonne truck, a DUX DT40 40 tonne truck and two Toro 40D trucks.

Primary ventilation of the Main Lens is up to 175 m3/s (375,000 cfm) with four 56 kW Joy 2000 series Axivane fans located at 17 North (see Figure 3.2). There is no direct mine air heating. With the ambient rock temperature of minus 12°C, the incoming ventilation air is warmed by the rock in the winter months and cooled in the summer months. Waste heat from the compressors is radiated into the fresh air supply. The ventilation network is simple with fresh air entering the central area of the Main Lens and flowing east and west. Secondary fans ventilate the satellite zones with air drawn through raises from the Main Lens. There are approximately 21 secondary fans in use, ranging from 22 to 93 kW.

Since the mine is located in permafrost, there is no water pumping requirement.

SRK considers the mining methods appropriate, and believes that the LoM Plan and associated production schedule are achievable.

5.4 Bougrine Mine

The main mining method is modified room & pillar with delayed unconsolidated backfill between sublevels. In the case of the F2 zone, this could also be described as a modified AVOCA method where intact slices are kept in the mineralized body to increase overall stability.

Pillar spacing is approximately 20 metres along strike. Discontinuities, bedding and fracture orientations are taken into account in pillar design. The system allows for an extraction of approximately 80% of the in-situ resources. The geometry of the F2 zone with its 45° dip requires a balance between additional dilution and recovery, as the minimum inclination of the slot raises and the footwall contact of the stopes is 60°.

The material is extracted by conventional drifting and slashing on sublevels with 15 metre spacing between sublevels. Longhole blasting is undertaken to extract the material between the sublevels. Backfill is then placed via the upper sublevel, which requires that some short access drifts be developed in the hanging wall sequence.

Mining in the upper levels and namely in the F3 zone has had to take into account the excavations established by the previous owners. In continuation of the previous mining plan several slices were recovered in the F2 area between pillars of cemented backfill. A redesign of the mining method of the F3 zone was necessary due to ground conditions in the east end of the deposit. Extensive rockfill with a high cement content was required to fill the entire block following which this end of the zone is being mined as a new block by jumbo method only.

Underground equipment includes three Tamrock 2 boom drill jumbos for development. Stope drilling equipment consists of two longhole drills, one Tamrock Solo drill, and one Tamrock Mono drill. Production is carried out with five Toro 5 m^3 scooptrams, one Toro 3.5 m^3 scooptram, and six Toro 35 tonne capacity trucks. A Toro 400 scooptram is presently being rented to assist with production.

Four ventilation fans supply 250,000 cfm. Mine workings include a 210 metre ventilation raise.

SRK considers the mining methods appropriate, and believes that the LoM Plan and associated production schedule are achievable.

5.5 El Mochito Mine

El Mochito utilizes a combination of mining methods dependent upon the size, geometry and geotechnical considerations of the various zones. Within the principal Nacional zone, BWR is presently mining an area of highly-variable grade and geometry by "post-pillar" methods utilizing hydraulic and pneumatic jumbos with hydraulic backfill, which yield 80% recovery.

Within the remnant, higher-grade fringe areas and pillars of the San Juan zone, conventional shrinkage and cut-and-fill stoping is practiced. Improvements in the transportation system, including extending the underground conveyor system, increasing broken material storage capacity and installing a crusher to feed the conveyor belt, were completed and commissioned in February 1996. Mineralized material in the Nacional and Lower San Juan is loaded by 2.7 cubic metre capacity load-haul-dump vehicles into 12 and 15 tonne trucks for transport to an ore-pass which feeds an underground crusher and then a 0.9 metre wide conveyor. The material is then conveyed upgrade to chutes on the shaft-access rail system where it is transported to the shaft ore-pass system. Material is hoisted via 10-tonne skips to the surface, where it is conveyed to the surface crushing circuit and subsequently to storage in a surface ore-bin.

Although the mine is old and needs investment, it functions well within the constraints of small shafts and consistently has improved over the years; it has continuing improvement plans and SRK has confidence that it will meet planned objectives.

The post-pillar, cut-and-fill mining method is ideally suited to the very variable geometry, grade distribution and rock conditions of the majority of the reserves and resources. The method is well understood and well executed with good control over grade and dilution. It is essentially room-and-pillar but with backfill to provide a working floor where mining heights are greater than a few cuts. In low height areas backfill is not used and the excavations are left empty with a maximum unfilled height of 16 metres over small areas.

The waste rock is mostly competent. The rock mass is generally reasonable and the stability of the back is therefore controlled by large structures and the span between pillars. The depths are moderate and the post-pillar, cut-and-fill method is only marginally impacted by stress. The post-pillar, cut-and-fill method is designed on 16 metre centres with final 6 metre pillars leaving a nominal 16% of the ore in pillars. In sill recovery areas, pillars are started on backfill. Old backfill areas used high cement contents and the fill is relatively stiff. BWR starts with large pillars on the first cut and reduces size as the pillars are surrounded with fresh backfill. From a rock mechanics point of view, the pillars generally behave very well from the outset.

SRK considers the mining methods appropriate, and believes that the LoM Plan and associated production schedule are achievable.

5.6 El Toqui Mine

The El Toqui deposits all dip gently to the south, and are amenable to room and pillar mining. The Doña Rosa Mine is accessed via an adit. Mining is carried out in a single horizon from flat lying mineralization varying in thickness from 8 to 12 metres.

A room and pillar extraction method is used with 11 metre rooms around 8 metre pillars. Generally, ground conditions are good and roof support consists of friction split set bolts installed on an as-required basis.

The operations are mechanized utilizing electric-hydraulic drill jumbos, a roofbolting jumbo, eight-cubic yard scooptrams and thirty tonne capacity trucks.

The mine, which is situated well above the valley bottom, is relatively dry. Rainfall in the area is high and some localized water inflow occurs that is collected and pumped out.

The mine operations are well staffed and have a generous amount of equipment. Daily production can be achieved with only two to three development blasts. There are three jumbos and four LHDs (three 8 yard and one 6 yard) and two bolters (with reasonable utilization this fleet should be able to achieve double the current production rate). All haulage is done by a contractor, with loading done by El Toqui personnel.

SRK has little concern with the mine achieving the planned production rate. BWR is currently addressing problems with low mechanical availability.

The shift schedule is attractive to workers:

- Two 12 hour shifts per day, seven days per week.
- Seven days on, seven days off.
- Workers are bussed to site where they are given breakfast; lunch is provided in the mine; an evening meal is also provided.

There are good options to increase the production rate in the future. BWR is considering upgrading the mill to a rate of between 1,500 to 2,000 tpd. The mineral reserves and mineral resources are very extensive and a higher production rate could easily be sustained.

A key element of the 5 year plan is recovery from pillars. It should be noted that whole pillars are not being extracted. "Pillar recovery" means "pillar reduction" usually to 50% of its current design size.

It should be noted that the pillar slashing to a final dimension has been accelerated to achieve a higher grade during a period of reduced metal prices. It is clearly a higher risk source of ore and in some areas might carry extra costs for rehabilitation of the back. There are extensive mineral reserves and mineral resources that can quickly be brought in the LoM plan. Approximately three areas have been fully "recovered" with no major problems.

SRK considers the mining methods appropriate, and believes that the LoM Plan and associated production schedule are achievable.

5.7 Langlois Mine

The mining method initially implemented in 1996 at the Langlois Mine was transverse longhole stoping using 114 mm ITH drills. The level spacing was 60 metres and blocks were 20 metres along strike and four to five metres thick. Remote scoops were used in mucking. Once terminated, the stopes were filled with high-density fill with 78% solids.

The mining method was plagued from the beginning with excessive dilution. The excessive dilution stemmed from the height of the stope and the sericitization and chloritization of the joint sets and the wall erosion from the ore passes. In 1997, a corporate decision was made by Cambior Inc. to stop operations and convert the 60 metre high stopes into smaller 15 metre or 30 metre stopes depending on the width of the mineralization.

The revised mining method saw the block sizes reduced to 20 metre heights and 20 metre lengths (15 metres between sublevels). Where widths were over 3 metres, sublevels were spaced at 30 metres. Smaller production drill holes were employed in order to reduce the blast damage (54 mm diameter). Remote scoops that range from 2 cubic yards to 3½ cubic yards were used in mucking.

Production drilling in the 30 metre high stopes was carried out using a CMS 360 ITH drill, drilling 114 mm holes. Patterns were three holes wide with a burden of 2.5 to 3.0 metres depending on the width. Two other production drills were used for production drilling and cable bolting. In the 20 metre high stopes, a standard carrier-mounted top-hammer longhole rig was used drilling 54 mm diameter holes. Hole lengths were 15 metres on a square drill patterns of 1.2 metres by 1.2 metres.

The stope and drift rounds were blasted using ANFO. Most of the holes were drilled down except in the extremities of the zone where no access is present above. Powder factors are 0.6 kg/t to 0.9 kg/t. 76 mm plastic casings were used in the 114 mm holes to reduce problems of lost holes and reduce the re-drilling problems. A gradual conversion of the production drilling to 54 mm and 64 mm diameter holes took place in order to reduce the blast damage caused by the larger diameter holes. All the blasting is done centrally and the fans are turned off before every blast in order to avoid damage from the potential sulfur blasts.

The mine is largely trackless, apart from some sublevels that do not have ramp access. In general, where the mineralized widths were over three metres, a ramp access to the sublevel was made. All broken material from the stopes was hauled to a V-shaped ore-pass system that converges to a crushing station on Level 14. Only one truck existed within the mine and it was used only intermittently.

All backs were screened with wire mesh. 1.5 metre rebars were placed on a 1.1 metre by 1.1 metre pattern. Resin bolts were used in the back to avoid corrosion of the ground support due to acid mine water. The walls were bolted with 1.5 metre split set bolts and strapped at two heights on the walls.

Cable bolting was used only on the top stopes. These were 6 metre cables spaced every 2.5 metres used to maintain the drift after the stope had been blasted. Three cables were installed on each side of the drift. Holes were drilled, one down, one flat and one up. The cabling helped in reducing the dilution at the drift elevation, but did little for the central part of the stope.

Two main ventilation raises supply the fresh air to the mine. During the summer they use a total of 248,000 cubic feet per minute. Exhaust is through the main production shaft and through a raise bore-hole in Zone 97.

Primary dewatering pumps are multistage Mather and Platts. Average pumping rate is 100 U.S. gallons per minute. The main pump stations are located on Levels 13 and 16.

Serious problems developed with the existing ore-pass system being used at the mine. The ore-passes located close to the shaft in Zone 3 failed due to the excessive scouring of the walls of the ore-pass causing excessive dilution. Production was also interrupted due to hang-ups in the ore-pass caused by large slabs of waste.

As mentioned in Section 2.5.6, SRK prepared a feasibility study with the primary objective of bringing the Langlois Mine back to profitability. The planned average zinc head grade of 11.17% in the feasibility study is significantly higher than the head grades achieved during the last four years of production, which ranged from 6.4 to 7.9%. The planned zinc grade is higher because:

- Zone 97 high-grade tonnes will become part of the production stream for the first time.
- The feasibility plan incorporates a higher cutoff grade than past mining.
- The feasibility plan includes mining of the higher grade shaft pillar which was previously considered sterilized.

The feasibility operating plan incorporates the following improvements to ensure reliability of production and to control costs:

- There are no ore passes planned for Zone 97 due to their unreliable nature. A fleet of new 20 tonne trucks will haul ore from stoping areas to the shaft area, and the past problem of collapsing ore passes will be avoided.
- A new steel lined storage bin is planned from Level 10 to Level 11 to provide ore storage while overcoming the past hang up problems.
- The feasibility study mining plan provides for pre-development of several sublevels in Zone 97 so that it can produce higher grade ore continuously when mine operations resume.
- Several improvements are planned for the underground mobile equipment fleet:
- Refurbishing of the existing mobile equipment prior to going back into service.
- Several new units will be purchased to meet mine plan requirements.
- Many improvements are planned to the mobile maintenance program.
- Two graders will be purchased, along with a small crusher for road material.

- A new underground garage is planned on Level 13.
- Zone 97 employs an overhand benching method with a reduced stoping height to control dilution and avoid the associated delays.

Before the Langlois Mine can resume full production, approximately 18 months of construction and development work is required. The major construction projects include:

- Construction of the Level 10 - 11 steel-lined ore bin.
- Steel lining critical ore pass sections.
- Construction work on ore and waste pass dumping points.
- Rehabilitation of the existing underground mobile equipment fleet.
- Construction of the upgraded ventilation system.
- Backfill systems including paste backfill pumping and delivery on Level 6 and cement slurry delivery to Zone 4.
- Development and construction of a new mobile maintenance shop on Level 13 and shop improvements on Level 9.

5.8 Caribou Mine

During the last half of 1998 and the first quarter of 1999, a re-opening plan was developed for the mine. This re-opening plan was further updated during 2000. The re-opening plan calls for production from the Caribou underground mine at the rate of 1,650 tonnes per day and from the Caribou open pit mine at the rate of 1,350 tonnes per day. The reserves from the open pit will be depleted in 21 months and the underground reserves accessed from the open pit will be developed in time to produce 500 tonnes per day. At this time, production from the Caribou underground will be increased to 2,500 tonnes per day.

As part of the re-opening plan at Caribou, the main ramp will be extended down to Level 7 at the 1900 metre elevation to recover the reserves in this area. The shaft will be deepened to Level 6 and a crusher and loading pocket installed. Material from below Level 6 will be hauled by trucks up the ramp to the ore-dump on Level 6. Ventilation raises, backfill raises and mineralization and waste passes will be extended down to Levels 6 and 7 as needed. An interim mining level will be created on Level 5N3 to increase the number of working areas and allow for the increased production rate of 2,500 tonnes per day.

The mining method proposed will be the same as used in the previous operating period. This is an AVOCA type method with the use of development waste and stockpiled surface waste as backfill.

BWR plans to mine the Caribou open pit in two stages. The first stage is expected to be the resumption of the open pit operation with the second stage being the development of the underground deposit commencing immediately upon depletion of the open pit reserves. The open pit is expected to be mined using a conventional bench method, and employ rubber tired loaders and trucks. Waste will be mined in ten metre benches, while the mineralization is expected to be mined in five metre benches to allow for better grade control. Mineralized material from the open pit deposit will be crushed at the open pit site and then hauled to a separate ore-pass at the Caribou mill site.

6. METALLURGICAL PROCESSING

6.1 Introduction

This section describes the mill processes of each mining asset.

6.2 Bouchard-Hébert Mine

Milling capacity has been stated as 3,000 tonnes per day using conventional grinding and flotation to produce separate copper and zinc concentrates. The Bouchard-Hébert process flowsheet is presented in Figure 6.1.

Primary crushing takes place underground utilizing a jaw crusher, which reduces the material to minus 100mm particle size. Primary grinding is done in a SAG mill operating in open circuit. Classification of the SAG mill discharge and flash cell tailings is by cyclones with the underflow feeding two ball mills in parallel. The ball mill discharges are combined and pumped to a flash flotation cell, while the cyclone overflow reports to the third stage of grinding to produce a final ground product of 80% passing 38 microns.

Copper flotation commences in a flash flotation cell that recovers 45% of the total copper at a grade of 14% to 18% copper with 40 grams of gold per tonne. The flash flotation cell maximizes the overall gold recovery and produced a final copper concentrate grade. The flash flotation cell tailings are returned to the primary cyclone feed.

The ground product of the third ball mill circuit is pumped to a conventional copper rougher and cleaner circuit (four stages) to recover and upgrade the remaining recoverable copper mineral in the mill feed. All copper flotation is done in an alkaline circuit.

The copper first stage cleaner tailings and copper rougher tailings are combined to form the zinc circuit feed. The zinc circuit uses conventional flotation cells to float a zinc rougher concentrate, which is upgraded in three stages of cleaning. Zinc flotation is done in a more alkaline circuit than the copper circuit. The zinc rougher tailings and zinc first cleaner tailings are pumped to the paste backfill plant.

Copper and zinc concentrates are pumped to their respective high capacity thickeners followed by filtering in pressure filters to obtain the final moisture content in the respective concentrates that are less than their transportable moisture limits.

Tailings not used for underground paste backfilling are disposed of in the tailings pond. Tailings pond effluent is treated with lime to maintain a pH greater than 9.5 in the settling and polishing ponds. Approximately 65% of the mill process water is recycled water from the backfill thickener overflow, and the remaining mill process water requirement comes from site surface pumping.

The paste backfill preparation process involves thickening and filtering the flotation circuit tailings. The filtered tailings are stored in a hopper and then sent to a high-intensity mixer where cement, fly-ash and water are added to produce a homogenous paste containing over 80% solids. The paste is fed underground by gravity.

An Outokumpu Courier 30 on-stream analyzer continually analyzes 17 streams for copper, zinc and iron mineral slurry. An Autometrics Multipoint PSM-400 (particle size monitor) has been installed to monitor the grinding circuit operation.

The zinc concentrate typically has a grade of 54.0% to 54.5% zinc at 85.5% recovery. The copper concentrate typically has a grade of 16.0% to 17.0% copper at 85.0% recovery. The copper concentrate contains 53% to 54% of the gold and 33% to 40% of the silver in the mill feed.

The zinc concentrate is sent by truck to a railway station in Rouyn-Noranda where it is transferred into railcars and sent to either CEZinc Refinery in Valleyfield, Québec owned by Noranda Inc. ("Noranda") or to the Port of Montreal.

The copper concentrate is sent by truck approximately 30 kilometres to Noranda's Horne Smelter in Rouyn-Noranda, Québec.

A review of BWR's historical production data shows that the Bouchard-Hébert Mill has processed up to one million tonnes per year during the period 1997-1999, decreasing in the year 2000 to 797,213 tonnes because of the SAG mill structural failure and rebuild which reduced the throughput. The year 2001 milling has been 867,715 tonnes to September 30 at ore grades of 4.56% Zn, 0.78% Cu, and 1.52g/t Au. If the current production is maintained, one million tonnes will be milled by year-end. Plant availability has been 83.6% in the year 2000, ~95% in 2001 year-to-date.

Concentrate grades for each of the separate concentrates averaged 54.42% Zn, and 16.56% Cu, for the period 1 January, 2001 to 30 September, 2001 at recoveries of 86.9% Zn and ~85% Cu.

The copper concentrate contained 58% of the gold in the mill feed as of 30 November, 2001, and 27,036 ounces were recovered. This makes Bouchard-Hébert one of BWR's major by-product gold producers.

Penalties charged for the copper concentrate are mainly for lead content when that content exceeds 1.5%. However, the normal lead content is less than 1%.

A fine grind is required at Bouchard-Hébert to optimize mineral liberation, producing a zinc concentrate with a particle sizing of 80% passing 36 microns, which is indicative of a fine grained mineralization. The penalty element in the zinc concentrate is iron, attracting a penalty if the iron content exceeds 8% Fe.

6.3 Nanisivik Mine

The mill has a proven capability of processing 780,000 tonnes per year using conventional crushing, rod and ball mill grinding, differential lead and zinc flotation, and concentrate drying. The mill process flowsheet is presented in Figure 6.2. The mill is 24 years old. Waste heat from the diesel power generators heats the buildings and dries the concentrates. Crushing is done underground using a jaw crusher and a cone crusher.

Lead flotation is carried out in one rougher and three cleaning circuits, using conventional flotation reagents. Zinc flotation is carried out in a rougher/scavenger circuit using conventional zinc flotation reagents, with the final concentrate being produced from a three stage cleaning circuit. Since mid-1997, all rougher concentrates are reground in a zinc regrind mill

Lead and zinc flotation concentrates are thickened, filtered and dried in rotary dryers to about 5% moisture, using waste heat from the power plant. The concentrates are trucked approximately three kilometres to the 125,000 tonne capacity storage shed at the shiploading dock.

Flotation tailings are pumped through a four kilometre pipeline to the West Twin Lake tailings disposal area.

BWR recently installed a dense media separation (DMS) plant at Nanisivik that came on-stream in July, 2001. The plant was designed to allow for the blending of mine plan grade ore with run-of-mine resources, and rejecting gangue material from the DMS feed, resulting in an increase in mill head grade. The DMS plant was a way of combating low ore grades which are now declining after 24 years of operation. A total capital cost of $7.3 million has been incurred for the plant and additional mining equipment.

The DMS plant experienced higher abrasive wear than expected on some of the operating equipment (DMS cyclones, screen cloths, and pump components). The recent replacement of these items with alloyed steel and polyurethane materials has corrected the problem.

Process results indicate that during September, 2001, DMS plant feed contained 4.48% Zn which was upgraded by the DMS plant to 6.2% Zn as feed to the flotation mill, indicating that the DMS plant was effective in upgrading the lower ore grade.

From 1996 to 2000, at an average mill feed grade of 7.35% Zn, a higher zinc concentrate grade (57.3% Zn) and recovery (96.3%) have been maintained. During the first three quarters of 2001, the mill feed grade averaged 6.5% Zn, yielding a zinc concentrate grade of 56.6% Zn and a recovery of 95.4%.

The zinc mineral (sphalerite) is easily liberated (57% passing 200 mesh), and is recovered into a good grade of concentrate.

The mill feed contains a very small quantity of lead mineral (galena) and a limited quantity of lead concentrate is produced. The galena is recovered by flotation, so it is not floated with the sphalerite, and does not report to the zinc concentrate.

The Nanisivik zinc concentrate contains minor amounts of the penalty elements such as iron (Fe), lead (Pb), bismuth (Bi), cadmium (Cd), arsenic (As), silica (SiO_2) and lime (CaO), making it readily marketable at any zinc smelter/refinery. The Nanisivik zinc concentrates contain 56% to 58% Zn and 160 to 300 g/t Ag.

Plant availability has historically been good, averaging 95% from 1998 to 2000, but year-to-date 2001 has been 93.1%, reflecting the problems previously referred to regarding the DMS plant.

6.4 Bougrine Mine

The Bougrine mine/mill operation was acquired by BWR in the later part of 1997 and recommenced production May 2, 1998, with commercial production being achieved by June 1, 1998. The mill process is presented in the flowsheet in Figure 6.3 in its latest state of revision, with additional flotation capacity.

The processing plant has the capacity to treat 1,200 tonnes per day. Stockpiles are maintained ahead of the crushing plant and ore is blended to provide a constant head grade (12% to 14% Zn) in feed to the mill. Ore from the mine stockpile passes through three stages of crushing and screening before it is conveyed to the mill fine ore bin.

Grinding is accomplished in a rod mill in open circuit and a ball mill in closed circuit with two stages of cycloning to produce the ground feed to flotation.

Conventional, differential flotation is employed to produce saleable lead and zinc concentrates. This is done in separate flotation circuits that have roughing and cleaning stages (lead mineral first, then zinc mineral) to maximize concentrate grades and mineral recoveries. These concentrates are separately thickened and pressure filtered before being stored at the mill site prior to shipping to custom smelters.

The flotation tailings are thickened with the overflow (tailings effluent) recycled to the mill process and the thickener underflow pumped to an impoundment area adjacent to the mill. The tailings effluent discharged to the tailings pond mostly evaporates and only a limited quantity is available for recycling to the mill. There is zero discharge of tailings effluent to the environment.

The proximity of the mineralized body to a salt dome results in the process water from the mine being highly saline. This high salinity causes poor selectivity in the flotation circuit when this water is used, and consequently impacts the concentrate grade. Revised operating procedures have lowered the salinity of the process water. A desalination plant, designed to provide better process water quality and thus allow better product quality, was constructed during 2000 and put into operation in late 2001.

The concentrates are hauled by train to the Port of La Goulette to BWR's storage shed where they are held for export to custom smelters for metal refining.

Since restarting the operations in 1998, the Bougrine mill has been processing ore at a rate of about 425,000 tonnes per year with metal content averaging 11.6% Zn and 1.82% Pb. It was noted that this grade of ore was about 2% lower than the grades processed in 1994 to 1996. However, the zinc concentrate grade has improved from an average of about 50% Zn prior to 1997 to the current average of 54.2% Zn for the year-to-date at 30 November, 2001. Zinc recovery has remained relative in the 81% to 83% range. Lead concentrate grades have improved from 64% Pb prior to 1997 to the current grade of 70% Pb. Lead recovery has improved from 74% to the current 76.4%.

The Bougrine zinc concentrate is very low in iron (less than 2% Fe) and the precious metals content is very low, with negligible gold and silver present. There is a notable low level of the normal 'penalty' elements such as arsenic, antimony, bismuth and cadmium. However, the zinc concentrate is in penalty levels for chlorine, silica and lime, as a result of the genetic mode of the ore (sedimentary in a marine environment) affecting the mineralogy.

Plant availability, which is a measure of the maintenance effectiveness, has been good, averaging 93.8% in the start-up year 1998, 96.2% in 1999, 95.9% in 2000 and ~96% as of 30 November, 2001.

6.5 El Mochito Mine

The El Mochito milling process flowsheet is presented in Figure 6.4. The mill is a conventional, differential sulfide flotation mill capable of processing 2,000 tonnes per day of ore, producing separate zinc and lead concentrates. The process consists of crushing, grinding, flotation, concentrate dewatering and tailings disposal.

Ore from underground is crushed in a three stage surface facility (jaw and cone crushers with vibrating screens) prior to trucking to the ore bin at the mill site. The grinding circuit consists of parallel open circuit rod mills, whose discharges are combined and past through two stages of ball milling. Each stage of ball milling is in closed circuit with cyclones. The ground ore is conditioned with reagents and then subjected to differential flotation in roughing and cleaning flotation circuits to produce saleable lead concentrate and zinc concentrate. These concentrates are separately thickened and then vacuum filtered before they are conveyed to storage sheds at the mill site.

The flotation tailings are pumped to the mine backfill facility where the tailings, upon demand, are subjected to cycloning to produce hydraulic backfill for the mine or by-passed to the tailings impoundment area. No tailings effluent is recycled to the mill, but it is treated and discharged.

The concentrates are hauled to a storage shed at Puerto Cortés via truck for subsequent export to custom smelters.

Since opening in 1948, the El Mochito Mine has processed over 14.4 million tonnes of ore with an average grade of 7.4% Zn, 3.9% Pb and 272 g/t Ag. Concentrate grades are in the 52% range for zinc, and in the 70% range for lead (containing ~3,000 g/t Ag). The high silver content in the lead concentrate combined with the good grade of lead makes this concentrate readily marketable. A penalty element in this concentrate is bismuth (Bi) over 0.35%. Typically, the bismuth content in the lead concentrate is 0.3 to 0.7% Bi.

The zinc concentrate contains 1.5% lead, 0.7% to 0.9% copper, 0.5% cadmium and 10% to 11% iron, of which penalty charges are paid for iron (Fe) over 8% and cadmium (Cd) greater than 0.30%.

Mill operating availability is currently averaging 90%.

6.6 El Toqui Mine

The El Toqui Mine and plant was bought by BWR in 1997. At that time, it was recognized that El Toqui was a high cost producer but certain operating improvements were planned to reduce the high costs. In September, 1998, milling was suspended while a 'capital improvement plan' was implemented, which plan was completed in January, 1999. The mill processes 1,000 tonnes per day of ore grading 8% Zn, 0.4% Pb, 0.91 g/t Au and 15 g/t Ag.

The El Toqui mill process flowsheet is presented in Figure 6.5. Ore from underground is trucked to surface stockpiles and/or the crushing plant. Crushing is performed in three stages (jaw and cone crushers with vibrating screens) to reduce the ore to minus seven millimetres. Grinding utilizes three parallel ball mill grinding circuits, each ball mill circuit is in closed circuit with cyclones. Conventional differential flotation including rougher and cleaner flotation circuits produce saleable gold/lead concentrate and zinc concentrate. The concentrates are separately thickened and filtered on vacuum and pressure filters before being conveyed to a storage area in the mill. The flotation tailings are pumped to a tailings impoundment area from which the tailings effluent is recycled to the mill. The concentrates are trucked 120 kilometres to Puerto Chacabuco for subsequent export to custom smelters for refining.

The gold/lead concentrate contains 15% to 30% lead, 100 to 120 g/t gold, and 1,500 to 2,000 g/t silver. Year-to-date to 30 September, 2001 reported 9,091 oz Au and 190,309 oz Ag contained in 2,924 dry tonnes of gold/lead concentrate. Penalty elements contained in the gold/lead concentrate are bismuth and arsenic which are penalized when they exceed 0.3% and 0.5%, respectively.

Zinc concentrate is 48% to 52% zinc content with 12% to 14% iron. The zinc concentrate iron content is penalized when it exceeds 8%.

6.7 Langlois Mine

The Langlois mill processed approximately 1,800 tonnes per day, five days per week, however, it has a capacity of 2,500 tonnes per day. The process flowsheet is presented in Figure 6.6. Copper and zinc concentrates are produced by differential flotation, with payable gold and silver recovered in the copper concentrate.

Ore is crushed in an underground jaw crusher and then sent to the grinding circuit, which consists of an open circuit SAG mill, and a ball mill in closed circuit with cyclones. The grinding circuit cyclone overflow feeds the copper flotation circuit, which contains conventional rougher, scavenger and three cleaner flotation stages with a regrind mill in the circuit to maximize the copper concentrate grade. The final copper concentrate is pumped to the copper thickener.

The copper first cleaner tailings and copper scavenger tailings are pumped to the zinc flotation circuit where they are conditioned in two tanks with lime to increase pH and depress pyrite. The zinc flotation circuit includes roughers, scavengers and three cleaner stages. The tailings from the zinc scavenger and zinc first cleaner scavenger constitute the final mill tailings and are pumped to the paste backfill plant. The zinc third cleaner concentrate is pumped to the zinc thickener.

Underflows from the zinc and copper thickeners are separately pumped to storage tanks and then to designate press filters. The filtered concentrates are conveyed to their respective loading stations. A storage building is annexed to the filter area/loading station for storage of final concentrates if direct shipping is delayed.

Approximately 60% of the tailings are used in paste backfill, and the remainder are sent to the tailings pond which covers an area of 1.88 square kilometres, located three kilometres from the mine site. The tailings deposition is subaqueous to prevent acid generation. Most of the tailings pond effluent overflow is recycled as mill process water, with a portion of the tailings pond effluent overflow released to the Wedding River after treatment with caustic lime to maintain pH levels in accordance with the regulations.

The paste backfill plant thickens and filters mill tailings, and then mixes the tailings with cement and water to make a paste that flows underground by gravity. The actual capacity of the paste backfill plant is 75 tonnes per hour using only one disc filter of the two available.

The zinc concentrate has an average grade of 53.4% Zn, and the copper concentrate has an average grade of 23.9% Cu. Penalty charges are paid for the iron content in the zinc concentrate and the lead content in the copper concentrate.

Zinc concentrate is loaded on CN railcars directly at the mine site and transported to Noranda's CEZ smelter in Valleyfield or to the port of Montreal. Copper concentrate is sent by rail to Noranda's Horne smelter in Rouyn-Noranda.

6.8 Caribou Mine

The Caribou deposit was first discovered in 1955 and has been subject to extensive metallurgical investigation since that time. In 1990, a bulk concentrate was produced for the Imperial Smelting process. The property next operated from July 1997 to August 1998 producing separate lead and zinc concentrates. This was possible due to the development of a new flowsheet and reagent scheme.

While the economics were enhanced with the newly developed flowsheet, plant performance did not reach the levels anticipated from the laboratory and pilot plant testing. The operation was closed in August 1998 due to low metal prices and lower than acceptable metallurgical performance. A pilot plant study, completed in November 1998, was followed up with a combined mineralogical characterization and metallurgical process development program in 2000. A less complex flowsheet was developed to selectively produce lead, zinc and copper concentrates.

From the compilation of pilot plant test results and data in November 1998, the operating period of 1997-98 and a 2000 Lakefield Research study, it was concluded that a number of flowsheet modifications are required to obtain the desired response to treatment of Caribou ore.

The future success of the concentrator is expected to be achieved by addressing the design and maintenance problems that plagued past operations, using high level process control equipment and practices, ensuring that all critical grinding and flotation parameters identified by Lakefield Research are achieved in the plant, utilizing a simplified flotation flowsheet and reagent scheme enhancing the operator's ability to manage the process, and ensuring that supervisors, technical staff and operators are well trained.

The "new" flowsheet developed at Lakefield Research, presented in Figure 6.7, is less complex than the existing flowsheet. Operators will be better able to understand the process and to respond to changes to ensure that process disruptions are minimized and that consistently good results are achieved. Problems experienced in the past balancing lead concentrate grade and lead recovery, which invariably resulted in high lead tailings impacting the zinc circuit, have been eliminated by open circuiting part of the lead cleaner tailings direct to final mill tailings. The new flowsheet also provides for improved revenues with the production of a saleable copper concentrate. While the plant has been redesigned to handle higher circulating loads, the actual circulating loads will be less onerous than in the past due to the simplified flowsheet and by ensuring that the fine grinds, required to process Caribou ores, are achieved.

The following is the process description of the modified circuit. The ore will be crushed to minus 150 mm (6") and conveyed from the mines into two, 2,000 tonne live capacity ore-bins. Caribou underground and Caribou open pit ores will be stored in separate bins to control the mix of the feed to the mill. Four feeders will discharge from the coarse ore-bins to a 1,000 mm conveyor. Two weightometers will be in use to control the ratio of Caribou underground and open pit ore being fed to the SAG mill at a combined rate of 140 tonnes per hour.

Ore will be ground in a 6,700 mm diameter x 2,133 mm long SAG mill with 1,500 kw connected power. SAG mill discharge will be sized on a Derrick vibrating screen with the plus 20 mesh portion being returned to the SAG mill. The minus 20 mesh portion will flow by gravity to the primary cyclone feed pumps. Further grinding will be completed in a 4,267 mm diameter x 1,875 mm long ball mill in a closed circuit with a cluster of 254 mm diameter cyclones. Final grind will be approximately 80% minus 27 to 30 microns.

Following aeration, the cyclone overflow will be directed to five, 16 m^3 Outokumpu lead rougher cells and a combination of two, 16m^3 Outokumpu and two, 8.5 m^3 Denver DR scavenger cells where the lead rougher/scavenger concentrate will be produced. This concentrate will be reground in the #1 lead regrind mill, a 3,250 mm diameter x 6,858 mm ball mill and will then be cleaned in a bank of eight, 8.5 m^3 Denver flotation cells.

The tailings from the first cleaner will be scavenged in a bank of four, 8.5 m^3 Denver DR cells. The first cleaner concentrate will be reground in #2 lead regrind, a 2,440 mm diameter x 3,048 mm ball mill. The ground concentrate will be cleaned in a bank of six, 8.5 m^3 Denver DR flotation cells. The tailings from the second cleaner and the first cleaner scavenger concentrate will be returned to the #1 lead regrind. The second cleaner concentrate will be cleaned in two successive stages using eight, 2.8 m^3 and six, 2.8 m^3 Denver DR cells to produce a final lead concentrate. The third and fourth cleaner tailings will be combined and directed to the copper separation circuit. A third 2,400 mm diameter x 2,440 mm ball mill will be available and configured into the lead circuit to supplement either the first or second regrind mill as required.

The copper separation feed, following SO_2 conditioning, will be directed to a four-cell, 2.8 m^3 Denver DR cell rougher. The rougher concentrate will be cleaned in two stages of countercurrent cleaning (three, 1.4 m^3 and two, 1.4 m^3 Denver DR cells) to produce a saleable copper concentrate and a tailings product that will be discharged to final tailings.

The final lead concentrate will be pumped to a high capacity thickener. The thickener underflow will be pumped to a stock tank for storage and subsequent pressure filtration. The copper concentrate will be thickened and stored in a stock tank for subsequent dewatering using the lead pressure filter.

The lead rougher tailings and the lead cleaner scavenger tailings will be combined as feed to the zinc circuit. After conditioning in two conditioners, the zinc rougher scavenger concentrates will be floated in six, 17.0 m^3 and six, 14.2 m^3 Denver DR flotation cells. These concentrates will be reground to 15 microns in a 3,200 mm diameter x 4,877 mm regrind mill, and then subjected to 10 minutes of high intensity conditioning. The zinc first cleaners will consist of seven, 14.2 m^3 Denver flotation cells followed by four, 14.2 m^3 cleaner scavenger cells. The second cleaner will be a bank of seven, 8.5 m^3 flotation cells, followed by five, 8.5 m^3 third cleaners and four, 8.5 m^3 fourth cleaners. The second, third and fourth cleaner tailings will be combined with the first cleaner scavenger concentrate and directed to the zinc regrind cyclone feed pump.

The final zinc concentrate will be thickened to 65% solids and pumped to a stock tank for storage and subsequent pressure filtration.

The flotation tailings consisting of the zinc rougher tailings, the zinc first cleaner scavenger tailings and the copper circuit tailings will be pumped to the tailings pond for subaqueous disposal. Process water requirement for the mill will be reclaimed from the tailings pond via a 400 mm diameter pipeline.

Figure 6.1: Bouchard-Hébert Mine, Schematic Flowsheet of the Metallurgical Plant

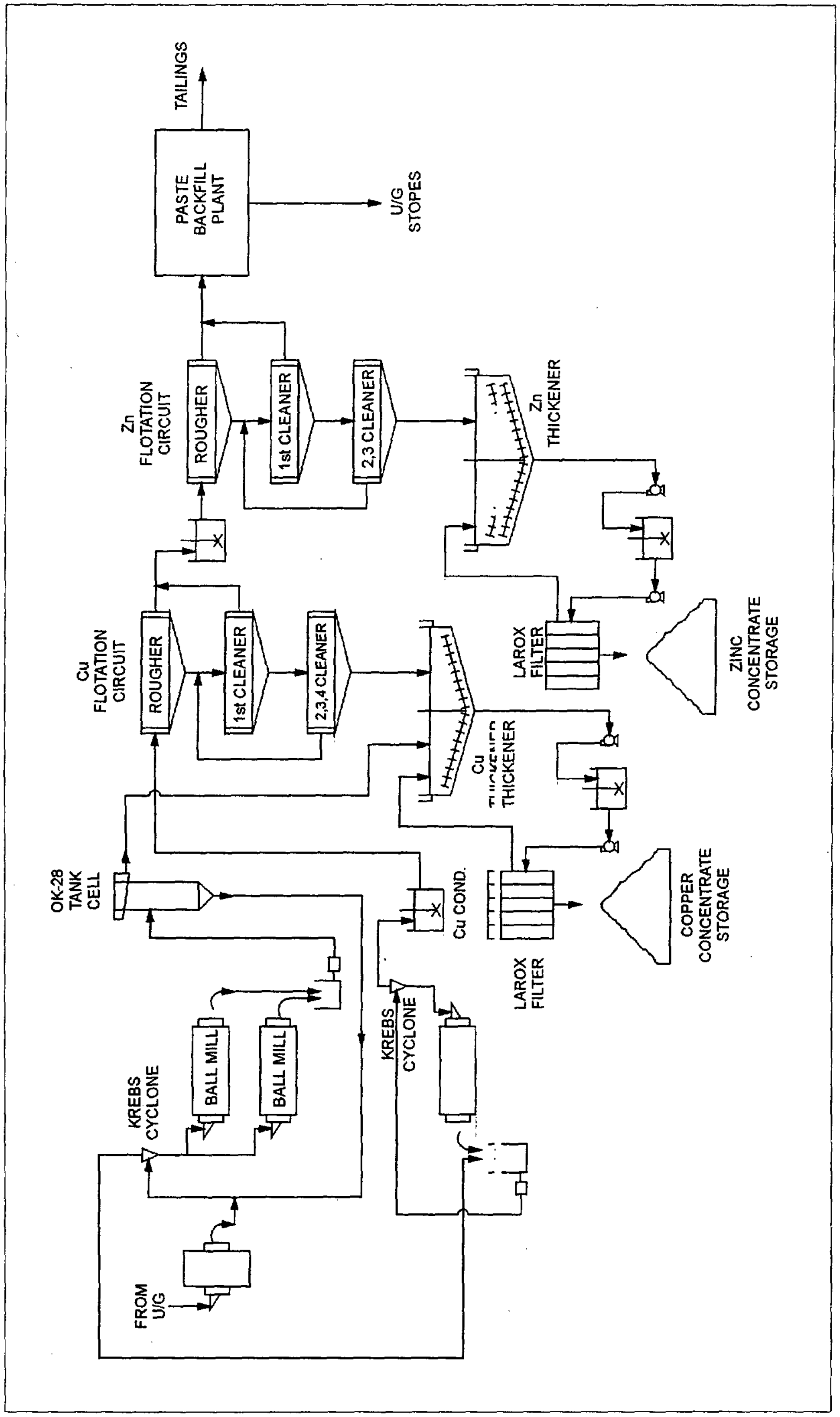


Figure 6.2: Nanisivik Mine, Schematic Flowsheet of the Metallurgical Plant

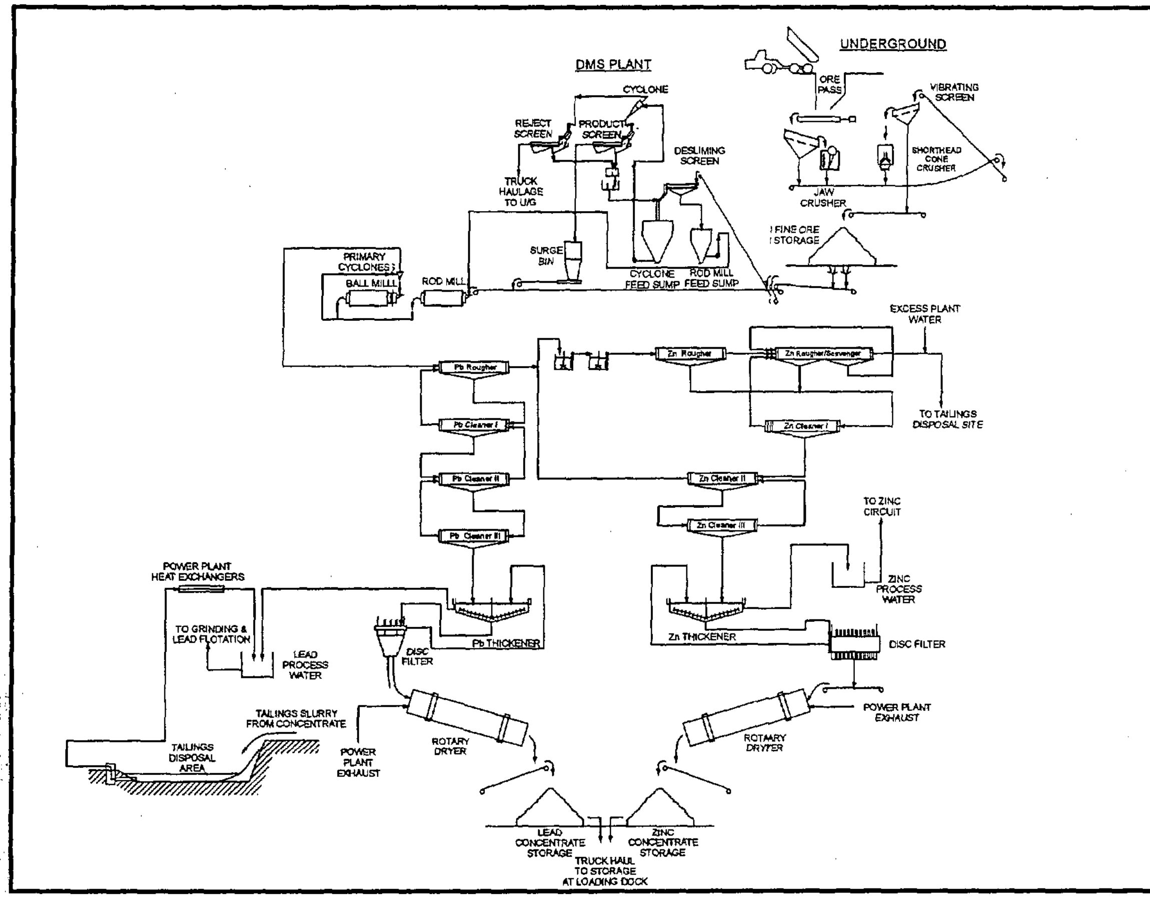


Figure 6.3: Bougrine Mine, Schematic Flowsheet of the Metallurgical Plant

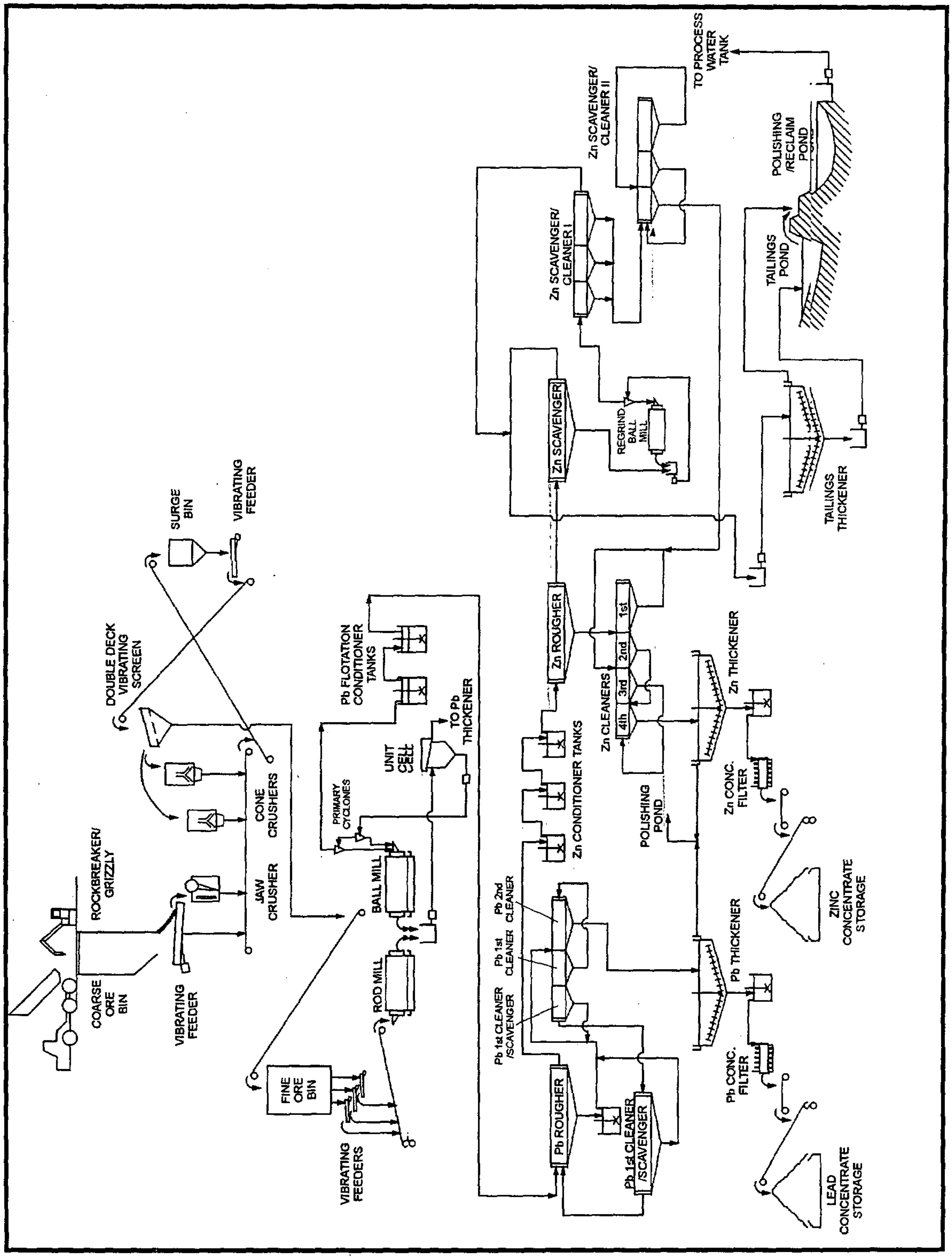


Figure 6.4: El Mochito Mine, Schematic Flowsheet of the Metallurgical Plant

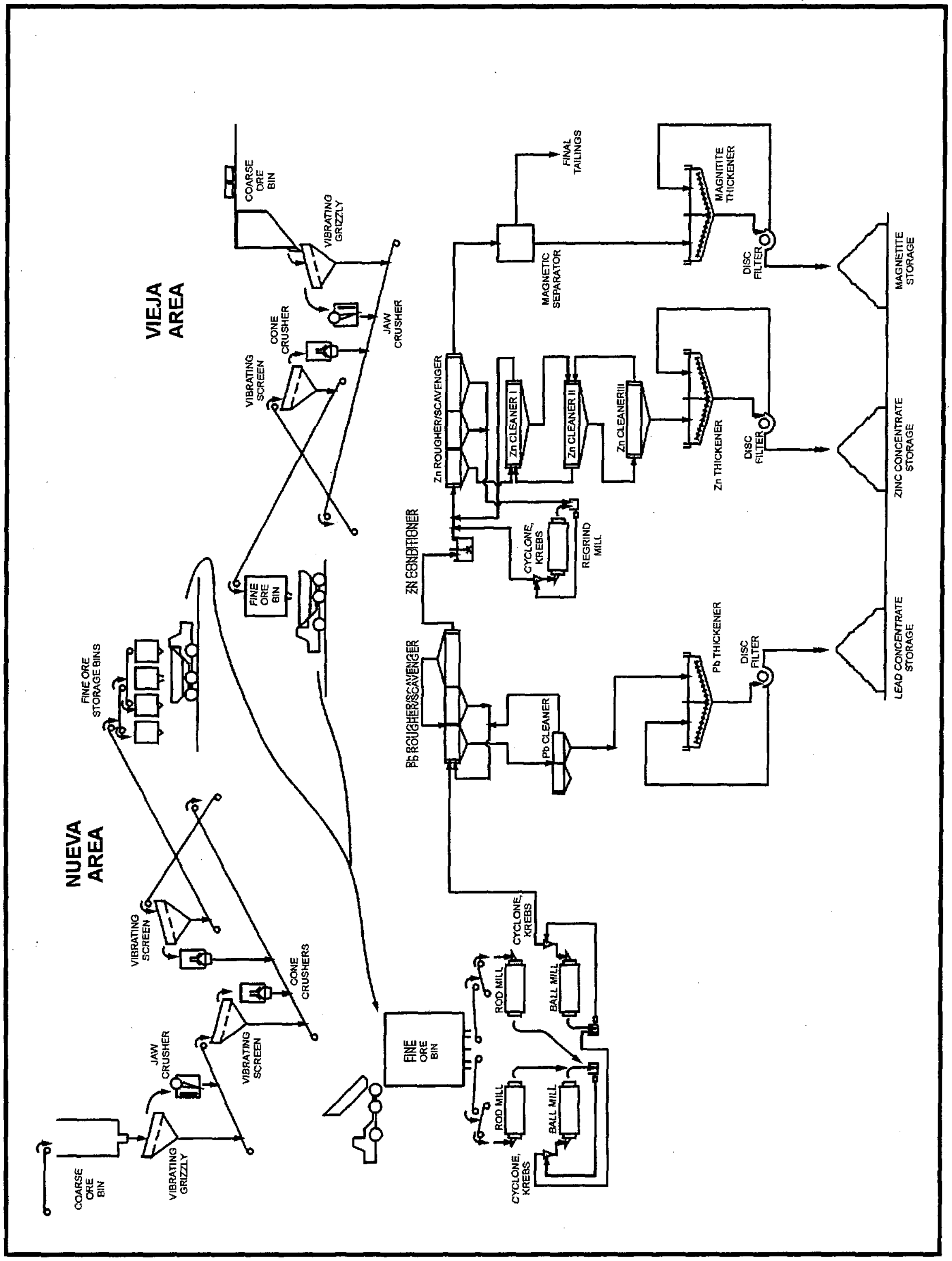


Figure 6.5: El Toqui Mine, Schematic Flowsheet of the Metallurgical Plant

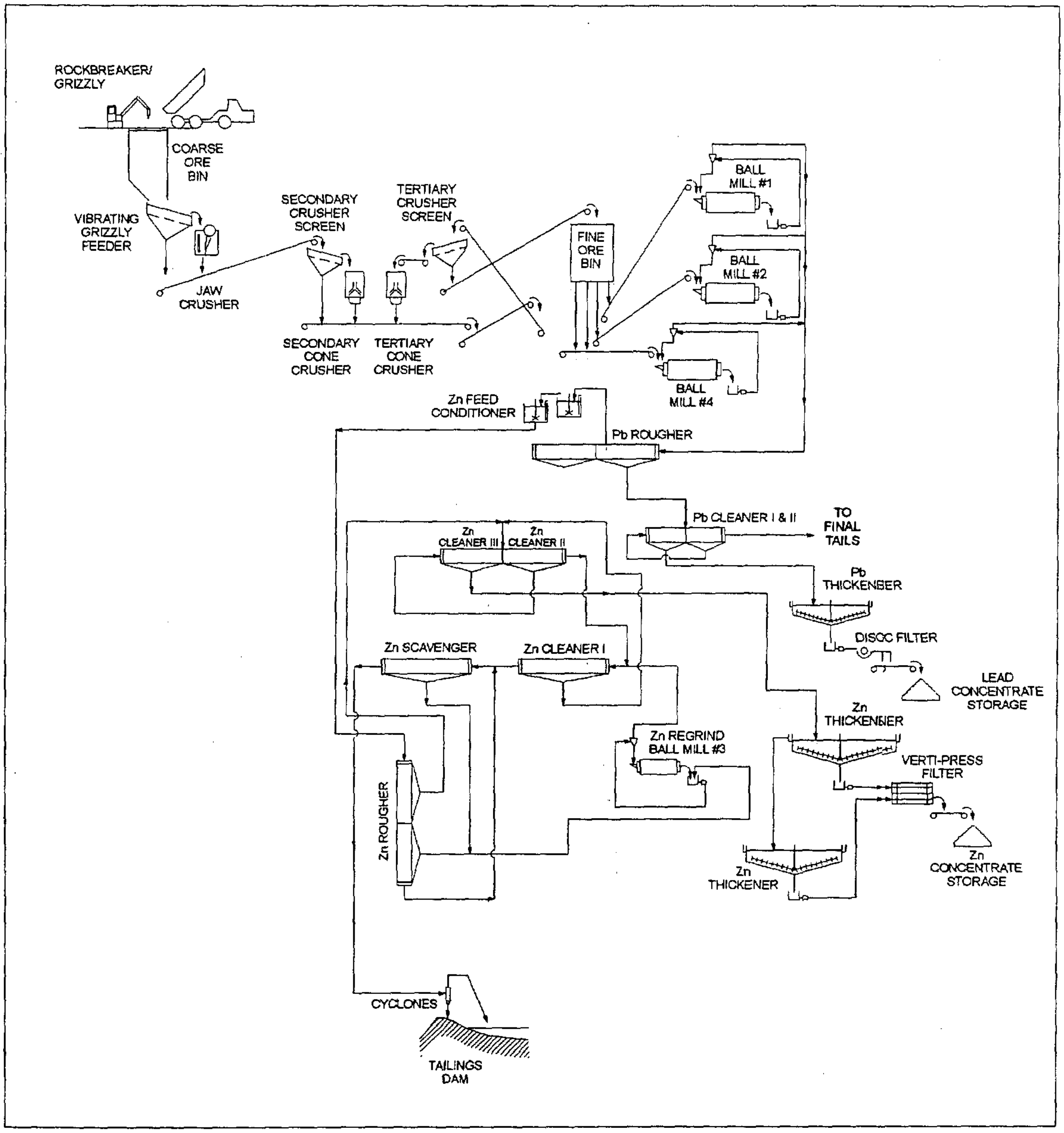


Figure 6.6: Langlois Mine, Schematic Flowsheet of the Metallurgical Plant

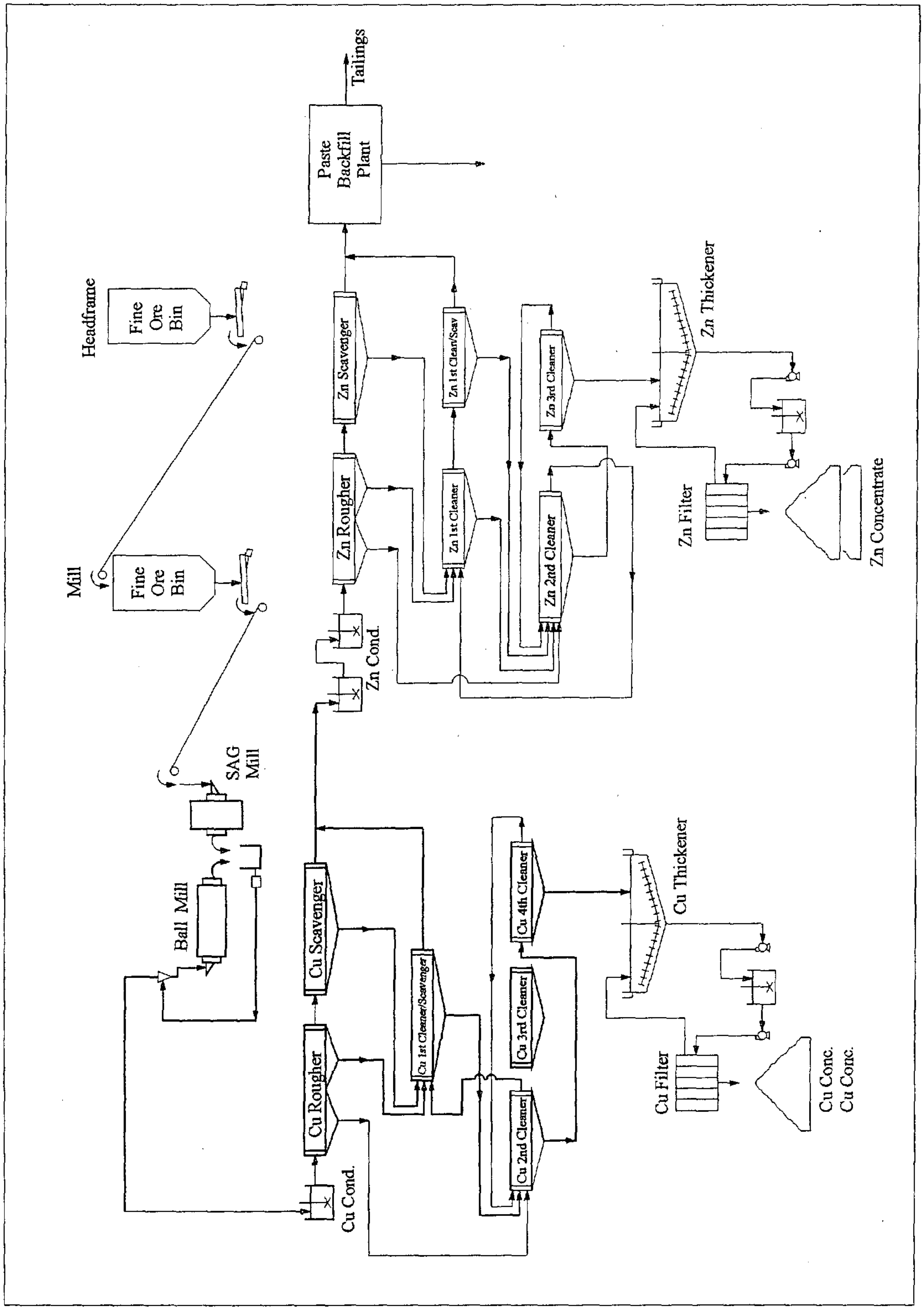
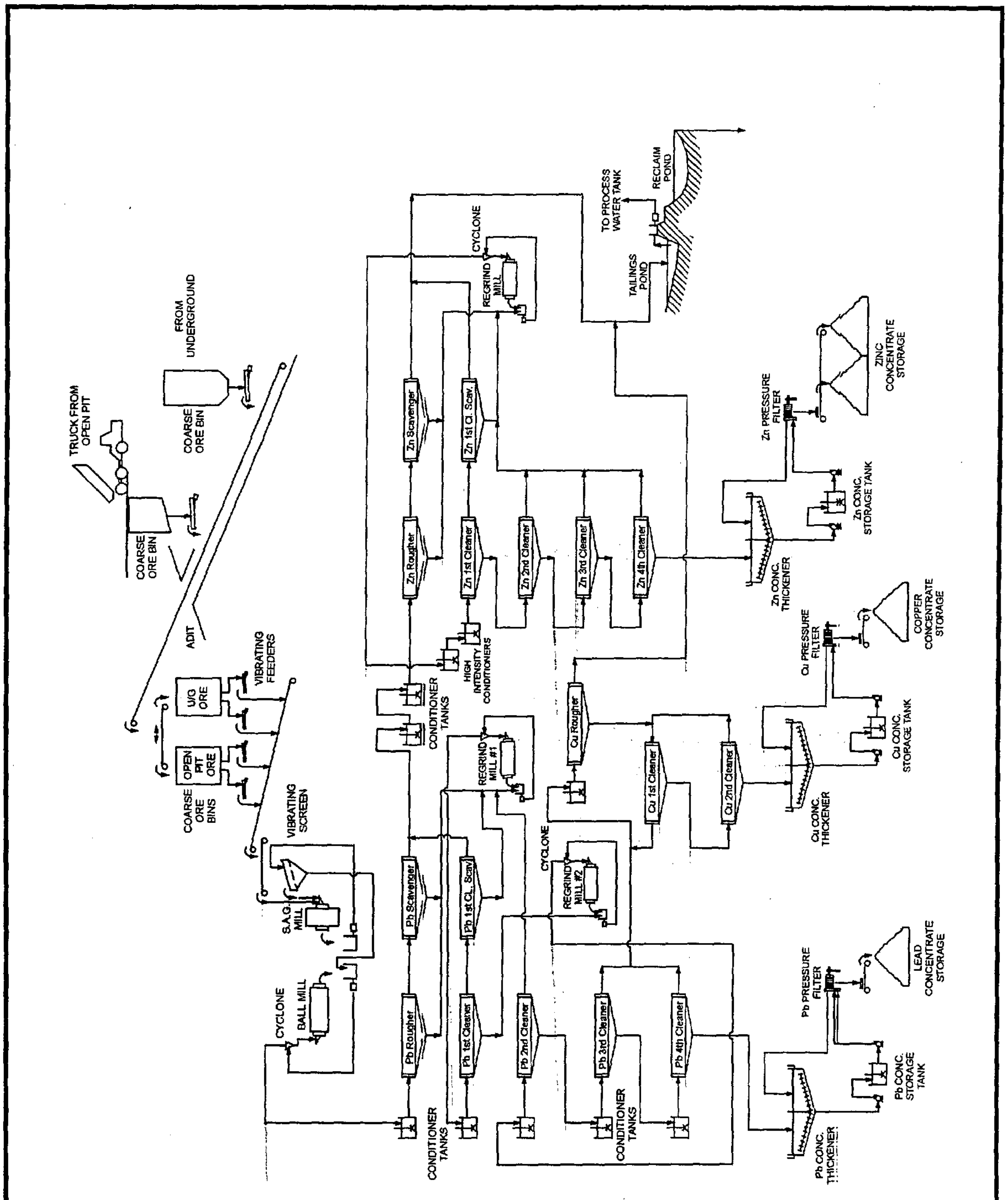


Figure 6.7: Carbiou Mine, Schematic Flowsheet of the Metallurgical Plant



7. MANPOWER

7.1 Introduction

This section lists the number of employees at each Material Property, and the unions that are present, if any.

7.2 Bouchard-Hébert Mine

Bouchard-Hebert currently employs 150 people, of which 100 are hourly employees. In addition to the mine's personnel, there are also 32 contractors. The hourly employees are represented by the United Steelworkers of America (USWA) pursuant to a certification order issued by the Bureau Du Commissaire General du Travail (the Labour Commission of the Province of Quebec) dated 2 March, 2001. BWR challenged the composition of the bargaining unit as proposed by the USWA and certain employees were required to vote indicating their preference to join the union or not. The employees in question opted not to join the union, but the process delayed the commencement of the collective bargaining process. Negotiation of the first collective agreement for Bouchard-Hebert is scheduled to commence in January 2002.

7.3 Nanisivik Mine

BWR employs 184 people at the Nanisivik Mine. Most employees work a rotation that allows for eight weeks of work followed by four weeks of rest. While on site, employees work a variety of five, six and seven day shift schedules, with most employees working either ten or twelve hours per day, depending upon operational requirements. Hourly-paid production, maintenance and service employees are represented by the United Steelworkers of America (USWA) and negotiations throughout 2000 culminated in a four-year collective agreement being signed effective 1 November, 2000.

7.4 Bougrine Mine

BWR Tunisia's workforce consists of 301 people, 10 being expatriate personnel and 291 being local personnel. Of the total 301 personnel, 289 are at the mine site and 12 are in Tunis. Some of the personnel in Tunis divide their time between the mine and the port on administration activities. The employees of Bougrine are represented by an employees' association with BWR and representatives of the employees review the remuneration package on an annual basis.

7.5 El Mochito Mine

The number of personnel currently employed at the El Mochito Mine is 718. This includes 25 expatriate staff, 140 clerical and local staff, 454 weekly employees and 99 temporary employees. There are also 86 security guards and 135 contract miners. The weekly employees are represented by the Union of American Pacific Workers. A three-year contract was negotiated with the union representing the weekly-paid employees for the period October 1999 through September 2002, which included wage and benefit improvements effective in October of 1999 and 2000, with a review of salary clauses in October 2001. BWR has informed the Union that under the current economic circumstances it is not in a position to revise the contract at the present, however, should the economic circumstances change BWR would inform the

Union and be willing to review the economic clauses with them at that time. BWR expects to commence negotiations on a new labour contract in September, 2002.

7.6 El Toqui Mine

El Toqui currently has a workforce of 239 people consisting of 197 hourly personnel and 42 staff personnel. Of the 239 personnel, 237 are at the minesite and 2 are in the Santiago office. The El Toqui employees are unionized with representation being provided by the "Sindicato de Trabajadores de Sociedad Contractual Minera El Toqui". A new three year collective agreement was signed in October, 2001.

7.7 Langlois Mine

When the mine was in full production there were a total of 162 workers, 19 contractors and 59 staff. Mine production was scheduled for five days per week, with weekends used for maintenance, hoisting and haulage when required. The mine crew normally worked two eight-hour shifts per day. The mill normally operated five days per week. The hourly paid employees of Langlois are unionized with representation being provided by the USWA. The negotiation of the first collective agreement was concluded in November 2001. The collective agreement is for a term of three years and provides for further discussion limited to wages and benefits when the mine announces its plans to re-open. The membership ratified the collective agreement at a vote held on December 21, 2001.

Most of the Langlois personnel live in the town of Lebel-sur-Quévillon, located 48 kilometres from the mine site.

7.8 Caribou Mine

The Caribou Mine employed 199 personnel at the time the operation was placed on care and maintenance. Various schedules were worked to maximize efficiencies and minimize employee travel time. Generally employees averaged 40 hours of work per week. The mill operated 24 hours per day, seven days per week. The mine operated two eleven-hour shifts per day seven days per week.

8. ENVIRONMENTAL MANAGEMENT AND PERMITTING

8.1 Introduction

The following section includes discussion and comment on the environmental and water management aspects of the mining assets. Specifically, detail and comment is included on the status of the environmental issues, environmental legislation and permitting, environmental management systems and environmental liabilities.

The technical and cost data concerning the environmental and closure aspects of the mining assets have been provided by BWR and their consultants. SRK has accepted this data, which forms the basis of the related discussion and conclusions in this report, as being factual and complete. SRK has reviewed the data for each mining asset and has concluded that, in general, the closure plans seem technically reasonable and the costs are close to what SRK would estimate. In some cases, SRK completed isolated checks to verify that the costs indicated are appropriate. The independent cost checks by SRK have correlated well with the BWR estimates in most cases. Final closure plans and associated cost estimates have yet to be prepared at any of the mine sites. A final closure plan is currently being prepared for the Nanisivik Mine.

Cost estimates have been reviewed on the basis of the expenditures deemed necessary to provide adequate environmental and water management, on-going remedial work, closure and post-closure. However, such costs cannot be regarded as definitive owing to the uncertainties inherent in potential changes to current legislation and that certain long-term issues are presently only addressed to a conceptual level by the mines. Issues, which represent major risks and which cannot be accurately quantified, are discussed below.

8.2 Bouchard-Hébert Mine

The mine operates under required Certificates of Authorization issued by the Québec Ministry of Environment under the Loi sur la qualité de l'environnement (L.R.Q., chapter Q-2; the Environment Quality Act). In addition, all approvals/leases required for land use (tailings pond, quarries and pits) have been issued by the Québec Ministry of Natural Resources in conformance with the Loi sur les mines (L.R.Q., c. M-13.1; the Mining Act).

Currently, mill tailings not used for backfill in the mine (approximately 35% of the total tailings) are deposited in the tailings pond as a conventional slurry. The tailings pond covers an area of approximately 70 hectares. Effluent from the tailings pond is sent to a retaining pond and then pumped to a treatment facility prior to seasonal discharge via a polishing pond, into the environment. The tailings pond was raised in 2001 to provide additional storage capacity. The current facility has sufficient capacity to meet life of mine storage requirements. Dam design and geotechnical evaluations are contracted to an independent firm. Based on the 2001 inspection report prepared by Journeaux, Bédard & Associates Inc., there does not appear to be any significant stability issues with the dams on site.

A closure plan is registered with the Quebec Ministry of Environment. A summary of the key elements, based on the plan provided by BWR:

- Excavate the sedimentation and polishing ponds and discharge the material to the tailings pond.

- Drain the tailings pond.

- Place acidic drainage producing material from the surface of the mine site into the tailings pond.

- Cover the tailings pond with an oxygen diffusion barrier.

- Allow the open pit to flood.

- Decommission openings to the underground with concrete caps, dismantle and salvage buildings, dispose of hazardous materials off-site and bury inert refuse on site.

- Active water treatment will be undertaken post-closure.

The Plan du Restauration was approved by the Quebec government in March, 1996. It was subsequently modified and approved in May, 1999. The closure plan reviewed by SRK provides a general description of the proposed closure activities. Although the document is not dated, the general nature of the plan is typical for a producing mine anticipating several years of continued operation.

Closure costs are estimated by others to be approximately $9.5 million.

8.3 Nanisivik Mine

The operating license for the Nanisivik Mine is an industrial water license issued by the Nunavut Water Board and overseen by DIAND. The license sets parameters for water use, water consumption, discharge water quality and waste disposal. The current license was granted in July, 1997 and BWR has been granted an extension to September, 2002. The renewal application process is underway for a seven-year licensing period that covers post-closure.

Tailings from the mill are pumped to the 60-hectare West Twin Disposal Area ("WTDA"). The WTDA is made up of a surface cell, reservoir and polishing pond. Tailings are deposited both sub-aerially and subaqueously, and following a final dyke lift in 2001 there is sufficient capacity in the WTDA to hold the tailings produced from the milling of the remaining mineral reserves. Ninety-five per cent of the process water required is recycled to the mill from the WDTA reservoir, while the excess water in the reservoir from the natural watershed precipitation is discharged to Strathcona Sound via Twin Lakes Creek.

The WTDA internal dykes are a frozen core construction and were last inspected in July, 2001 by BGC Engineering Inc. No significant stability issues were identified during the annual inspection.

Nanisivik holds an "Interim" Closure and Reclamation Plan approved by the Nunavut Water Board. As required under the current water license, this plan is updated annually and submitted to the Nunavut Water Board. The "Final" Closure and Restoration Plan is currently being prepared. The main components of the interim closure plan are summarized below.

- Use the waste rock currently on surface as backfill in the underground workings and open pits.
- Cover and contour the backfill in the open pits and allow permafrost to develop within the backfill.
- Seal the mine portals.

- Cover the WTDA area and allow permafrost to develop.
- Re-contour ore and waste rock pads and borrow pits.
- Remove the settled solids from the sludge pond at the East Adit water treatment facility and put them underground, then cover and contour the pond area.
- Control surface run-off.
- Decommission the site infrastructure.
- Monitor the site for five years following closure.

Based on information available to SRK, the closure plan seems reasonable and the major issues have been addressed. BWR believes that the site will be relatively straightforward to restore to a near natural condition.

Under the terms of the present water license, BWR is required to post security for reclamation in an aggregate amount of $7.0 million over the five-year term of the water license. To date, security of $4.0 million has been posted. BWR indicates the financial commitment at closure is approximately $12.02 million, which includes an annual post-closure expenditure for anticipated water treatment and monitoring. Once reclamation of the East Open Pit has been completed, BWR does not anticipate a requirement for water treatment at the site.

8.4 Bougrine Mine

The Bougrine Mine operates under an Environmental Agreement (August, 1991) with the Tunisian Environmental Protection Agency that stipulates operational guidelines and closure requirements for the tailings pond.

The mine is situated in an arid (semi-desert) climate zone with low and irregular precipitation and a high rate of water evaporation. As a result, the majority of the water in the tailings pond evaporates and the pond is managed as a zero discharge facility. Water required for mineral processing is recycled from the tailings pond and thickener overflows via the polishing pond, but must also be supplemented by two groundwater wells located approximately two and four kilometres from the mill. The environment in which the mine operates requires efficient water management.

The tailings facility consists of two earth embankments constructed of clay fill. Both dams were raised one metre in 1999. An additional raise of the tailings dam was completed in early 2001, and will have sufficient storage capacity to meet life of mine requirements. The dams are inspected annually by a geotechnical consultant. SRK (Cardiff UK office) inspected the site in December, 2001 and did not find any significant stability issues with the dams.

It is reported that the tailings facility leaks water into the underground workings and into the groundwater system. Some of this seepage reaches the surface via a spring approximately 500 metres to the east. Groundwater in the area is saline, therefore not likely used as a drinking water source by people in the area. Consequently, the seepage is unlikely to represent a public health risk. The Tunisian Environmental Authority has apparently not raised any concerns about this seepage. As the tailings facility dries out during closure, seepage from the facility should cease. With proper covering and surface run-off control, mobilization of secondary oxidation products, such as acidity and metals, outside of the tailings facility during the post-closure period, is not expected.

The host rock in the mine is limestone. Acidic drainage issues would not be expected from any waste rock that may be on surface or from the underground workings.

Tunisia does not have closure requirements. However, BWR has committed in the Environmental Agreement with the Tunisian EPA (1991) to reclaim the tailings facility. BWR also intends to dismantle the surface infrastructure and close off all openings to the underground to prevent public access.

SRK's experience indicates the financial commitment at closure could be in the order of \$1.5 to \$2.0 million in capital. The Tunisian Government does not require security to be posted.

8.5 El Mochito Mine

The El Mochito Mine operates under the Honduras Law of the Environment (1993). This law has a very limited section referring specifically to mining, for which the standards are for the most part World Bank standards. BWR believes the mine meets relevant North American standards as the relevant guidelines for its mining practices. In 1998, a new Mining Code was published, which calls for the creation of a "Manual of Environmental Policy" for establishing environmental standards. This manual was recently published in draft form and is now undergoing review by all stakeholders. The mining industry is participating in the review through their Mining Association.

Mill tailings are discharged to the Pozo Azul tailings facility where they are hydro-cycloned into coarse and fine fractions. Coarse material is utilized for upstream dam construction while the fine fraction reports to the reservoir portion of the pond. Dam construction and geotechnical evaluations are contracted to an independent consulting firm. Stage one of a two-stage construction schedule is complete. Stage two is now underway and is expected to contain tailings resulting from an additional five years production. Dam inspection reports prepared by Olsson and Associates (Denver, Colorado) do not indicate any significant stability issues associated with the dams on site.

The reservoir portion of Pozo Azul provides retention time to naturally treat water prior to discharge (along with mine water) to the Quebrada Raices, and subsequently to Lago (Lake) de Yojoa.

The closure plan focuses on the decommissioning of mine and surface infrastructure and the reclamation of the tailings facilities. Closure of the Pozo Azul tailings facility will begin once mining operations have ceased. Final and long term reclamation of El Bosque tailings facility remains to be completed and will consist mainly of geotechnical stabilization and erosion protection.

Based on the site geology and reports from SRK personnel who visited the site, acidic drainage and associated metal leaching do not appear to be an issue at El Mochito.

SRK's experience indicates the financial commitment at closure could be in the order of \$3.0 to \$4.0 million in capital. The Honduran government does not require security to be posted.

8.6 El Toqui Mine

The El Toqui Mine operates under various Chilean permits and authorities related to the environment. Site discharge waters are regulated under the "Norma Technica Relativa a Descargas de Residuos Industriales Liquidos Directamente a Cursos y Masa de Aguas Superficioles" (standards for industrial discharge into surface waters). These limits come under the authority of the Ministry of Health.

Mill tailings are discharged to the Confluencia tailings impoundment where they are separated via hydro-cyclone into coarse and fine fractions. The coarse fraction is utilized to construct the tailings dams in a centerline configuration in accordance with Chilean regulations. The fine fraction reports to the interior of the pond where it is deposited subaqueously. Reclaim water is pumped back to the mill for mineral processing while any excess is discharged via two sedimentation ponds to the Toqui River.

The Confluencia impoundment is expected to have sufficient storage capacity to contain all tailings produced until 2004. E.C. Rowe Inc. (Santiago, Chile) has been contracted for dam design and on-going geotechnical evaluations. Dam inspection reports were not provided to SRK for review.

There are three additional tailings facilities that are out of service and at various stages of reclamation.

A study of the acid generation potential of cycloned tailings was completed by Lorax Environmental in July, 2000. The study concluded that both the underflow and overflow tailings have the potential to generate acidic drainage. Zinc release is significant at circum-neutral pH levels.

The closure plan provided by BWR is incomplete but, in its current form, lists the closure elements and commits to capping mine openings and raises to prevent access, and covering and revegetating waste rock dumps and tailings facilities. Uncertainty is expressed regarding the presence of potential acidic drainage from the mine adits. It is unclear how the acid generation potential of the cycloned tailings will be addressed at closure.

The financial commitment at closure could be in the order of $3.0 to $5.0 million in capital, based on the elements presented in the closure plan. The financial commitment could increase significantly depending on whether Chilean regulators choose to insist on a more stringent closure plan, and whether the closure measures prevent acid generation, and the need to treat excess water and seepage. The Chilean government does not require security to be posted.

8.7 Langlois Mine

The mine operates under required Certificates of Authorization as issued by the Québec Ministry of Environment under the Loi sur la qualité de l'environnement (L.R.Q., chapitre Q-2). In addition, all approvals/leases required for land use (tailings pond, quarries and pits) have been issued by the Québec Ministry of Natural Resources in conformance with the Loi sur les mines (L.R.Q., c. M-13.1).

There are two surface dumps for the storage of waste rock. One dump is for the storage of non-acid generating rock and the other is for material with a low potential for acid generation.

Approximately 60% of the mill tailings produced is used for paste backfill underground with the remainder discharged subaqueously at the tailings impoundment. Retaining dykes are constructed of a sand and gravel mass with slope protection comprised of clean, non-acid generating mine rockfill. Seepage control within the dykes is achieved with a low permeability geosynthetic clay liner (thin layer of bentonite sandwiched between two layers of geotextile). The pond has the capacity to store all of the tailings produced in the feasibility study. At the end of mine life, there will be approximately 837,000 tonnes of tailings in the tailings pond. The tailings dams will not require raising for the tonnes mined in the feasibility study.

During operations, water from the tailings pond is reclaimed and used in the mill. Excess water is discharged to the Wedding River after being treated with caustic soda to elevate the pH to compliance levels.

During the current temporary closure period (which began December 2000) the mine is being kept on a care and maintenance program. The underground workings continue to be dewatered and all water is treated with lime before being sent to the tailings pond. All required environmental sampling, monitoring, and reporting continues.

Langlois holds a Restoration Plan (the "Plan") that received approval from the Québec Ministry of Natural Resources, in consultation with the Québec Ministry of the Environment, in August 1996. The Plan was subsequently updated, as required, and approved in January 2000. It comprises the following main elements:

- Place non-acid generating waste rock underground and the acid-generating waste rock will be placed under 1 metre of water in the tailings pond to avoid acid generation.
- Following closure of the tailings pond, direct surplus water through the emergency discharge and build an alternate discharge point on the south-eastern side of the pond.
- Dismantle all the buildings and infrastructure, move equipment to another mining operation, and dispose of any remaining hazardous materials off-site.
- Re-vegetate the site.

The next update of the Plan is required by August 2004. The cost to reclaim the mine site is currently estimated to be $1.7 million.

8.8 Caribou Mine

The Caribou operations are regulated under the New Brunswick Mining Act (Chapter M-14.1), Clean Environment Act (Chapter C-6) and Clean Water Act (Chapter C-6.1). Certificates of Approval ("COA") to operate have been issued by the New Brunswick Department of Environment for both the Caribou Mine underground site and the Restigouche open pit.

During the most recent operational period (1997 – 1998), tailings were discharged to the South Tributary Tailings Pond (STTP) facility. Tailings were deposited subaqueously via floating line and stored under one metre of water cover in accordance with the COA to operate. Retention time in the tailings pond averaged 200 days and served as water treatment. Water required for the mill is reclaimed while any excess was discharged to a polishing pond and subsequently to Forty Mile Brook. The tailings pond dam is a glacial till structure with an inverted filter drain on the downstream face. It is reported that there is currently sufficient storage capacity to contain tailings produced for three additional years of mine production. As part of the re-opening plan, a 1.4 metre lift will be placed on the dam to accommodate tailings produced from the entire current mineral reserves. Based on the 1998 Dam Inspection report prepared by Golder Associates, information provided by BWR, and discussions with Golder Associates, there does not appear to be any significant stability issues with the dams on site.

The Minister of the Department of Natural Resources and Energy ("DNRE") approved a Mine Reclamation Plan for the Caribou Mine on 6 March, 1996. The cornerstone of the closure plan is progressive reclamation, namely relocating the acidic drainage generating materials from the previous open pit operation to the underground during the course of mining. At the time the closure plan was prepared, the mine life was estimated to be six years. For economic reasons, the mine only operated for one year. Consequently, much of the geochemically problematic waste rock and tailings remain on surface. An older, smaller tailings facility (Anaconda) is located near the mill. The tailings dam is constructed of waste rock, some of which is reported to be potentially acid generating in a document prepared by BWR in 1992. This document has not been reviewed by SRK as it has been archived. The tailings are acid generating. This tailings facility is no longer used but has not been remediated.

Approximately 159,000 tonnes of waste rock of the 1.1 million tonnes produced during the former open pit operations were returned to the underground mine as backfill during the most recent operational period, as per the closure plan.

In year 2000, Golder Associates reviewed options for the remediation of the Anaconda tailings pond and recommended that they be reslurried, limed and pumped to the current STTP for subaqueous deposition. An agreement in principal has been received from DNRE for this remediation plan.

The closure plan is unclear regarding the acidic drainage discharging from the mine adit during the post-closure period, but it is likely to require treatment prior to discharge. Based on the information provided in the closure plan, the closure plan seems reasonable and takes into consideration the major issues.

BWR and CanZinco have an agreement with the Province of New Brunswick whereby the liability of East West Caribou Mining for reclamation work, rehabilitation work, other environmental work or environmental liabilities of any nature related to exploration and development activities at the Caribou Mine that took place prior to 29 October, 1993 be limited to $3 million. The Province of New Brunswick is responsible for liabilities exceeding the $3 million. As a result of this agreement, most of the current closure issues will be the responsibility of the Province of New Brunswick.

The cost estimate provided herein represents SRK's estimate of what is reasonable based on the provided information and the current situation at the site.

Security in the amount of $2,949,350 has been posted with the DNRE towards these costs in accordance with an agreed schedule. An additional amount of $1.1 million has been posted for environmental protection with the Minister of Environment as required under Regulation 87-83, the Environmental Assessment Regulation of the Clean Environment Act. The financial commitment at closure could be in the order of $3.5 million in capital. BWR estimates the closure cost to be $3,580,525.

An open pit operation is located at the Restigouche property. Ore extracted from the open pit was trucked to the Caribou Mine mill for processing. Consequently, the Restigouche site has a minimal amount of infrastructure, two waste rock storage pads and a small water treatment plant.

A mine reclamation plan was approved on 6 March, 1996 for the Restigouche open pit site. The following summarizes the key elements of the closure plan:

- Move potentially acidic drainage producing waste rock off the storage pad and into the open pit.
- Dredge the hydroxide sludge from the sludge pond, dewater and dispose into the Caribou open pit.
- Allow the pit to flood, which will be an oxygen barrier.
- Fill in all ponds.
- Recontour the inert waste rock.
- Regrade the site surface to improve site runoff and minimize erosion.
- Revegetate the site.
- Decommission infrastructure.
- Monitor the site for three years after closure.

Based on the information provided in the closure plan, the closure plan seems reasonable and takes into consideration the major issues. If the development of the pit and the storage of waste rock was implemented as described in the closure plan, there is unlikely to be a significant issue regarding poor quality seepage now and in the future.

The closure plan also specifies a three year post-closure monitoring period.

The cost estimate provided herein represents SRK's estimate of what is reasonable based on the provided information and the current situation at the site.

Security in the amount of $1.9 million has been posted with the DNRE towards these costs in accordance with an agreed schedule. An additional amount of $1.7 million has been posted for environmental protection with the Minister of Environment as required under Regulation 87-83, the Environmental Assessment Regulation of the Clean Environment Act. BWR estimates the closure cost to be $3,715,267.

9.0 ECONOMIC ANALYSIS
9.1 Introduction

BWR has developed an internal long-range plan technical model (the BWR LRP) for its operating mines (Bouchard-Hébert, Bougrine, El Mochito, El Toqui, Nanisivik) and non-operating mine properties (Langlois, Caribou). BWR produces a detailed 5 year plan for each operation on an annual basis. The current 5 year plans have been extended to mine out all of the currently-delineated proven and probable reserves to produce the LRP.

As mentioned earlier in this report, the LoM Plans and the technical economic projections include forward-looking statements that are not historical facts and are required in accordance with the reporting requirements of the OSC. These forward-looking statements are estimates and involve a number of risks and uncertainties that could cause actual results to differ materially.

SRK's review of the BWR LRP, with specific emphasis on fiscal year 2001 forecast and year 2002 budget, is presented in this section. This technical report contains details regarding the first two years of the BWR LRP. SRK audited all information contained in the BWR LRP and finds it reasonable.

9.2 Validation of BWR Model Inputs

The structure of the BWR LRP is comprised of three basic modules; (1) Assumptions & Model Inputs, (2) Mine Analysis, and (3) Summary Results. It appears that all links between these spreadsheets are properly made and that values have been properly calculated.

The Langlois and Caribou projects are currently under care and maintenance. The BWR LRP therefore does not include the potential revenues or costs from these operations in its net present value (NPV) calculation.

SRK finds no reason to question the integrity of the BWR LRP.

9.2.1 Economic Parameters

General modeling parameters used in the analysis are summarized in Table 9.1. Net smelter return (NSR) costs are presented in US dollars and capital and operating costs are presented in Canadian dollars. All values are in year 2001 basis with no escalation/inflation. The model uses actual metal prices realized for year 2001 and forecast metal prices for the remaining analysis.

Concentrate treatment, freight and marketing costs, summarized in Table 9.2, are based upon concentrate attributes (grade, metal content, etc.) and metal prices as are typical for any smelter contract. The values presented in the table are in the range of historical rates at these properties.

SRK is of the opinion that the economic modeling parameters used by BWR in its LRP are fair and reasonably reflect current metal and financial market trends.

Table 9.1: BWR LRP, Economic Modeling Parameters

Model Parameter	2001Forecast	2002 Budget	BWR LRP Average 2002 -2008
Metal Price (US$)			
Zinc	0.393/lb	0.397/lb	0.467/lb
Lead	0.212/lb	0.215/lb	0.215/lb
Copper	0.713/lb	0.680/lb	0.756/lb
Silver	4.34/oz	4.20/oz	4.44/oz
Gold	270.00/oz	275.00/oz	275.00/oz
Exchange Rate			
US$:CDN$	0.651	0.640	0.641
CDN$:US$	1.536	1.563	1.560

Table 9.2: BWR LRP, Concentrate Costs[1] (US$)

	2001 Forecast	2002 Budget	BWR LRP Average 2002 - 2008
Smelter Treatment Charge	164.29/dmt	162.35/dmt	188.05/dmt
Freight & Marketing	28.06/dmt	27.22/dmt	28.96/dmt
Total Charges	**192.35/dmt**	**189.57/dmt**	**217.01/dmt**

(1) Reflects average cost for all zinc, lead and copper concentrates processed in that year.

9.2.2 Technical Model Inputs

Technical model inputs used in the BWR LRP are shown in Table 9.3. For modeling purposes, only currently-delineated proven and probable reserves that total 11.1Mt of ore are mined through the term of the LRP. Production rates, ore grades and metallurgical recoveries over this period have been reviewed by SRK and found to be within those historically achieved at each respective operation. Based upon current market forecasts, BWR should expect a run-of-mine (RoM) zinc grade of 7.29%, with other byproducts increasing the grade to 8.89% zinc-equivalent.

All operating mines in the LRP exhaust their mineral reserves with the exception of the Nanisivik Mine. Due to low market prices, BWR has elected to cease mining operations at Nanisivik prior to the complete depletion of its ore reserves. As such, only 1.36Mt of the existing 2.9Mt of RoM ore available at Nanisivik will be processed before the mine goes into final reclamation. The RoM zinc grade to be mined will average 7.8% Zn against the published LoM reserve average of 6.9% Zn. The mine plan will exploit 106kt, or 54% of the mineable 198kt of zinc metal reserve.

The Bouchard-Hébert and Langlois Mines were purchased by BWR in 2000. An operating plan for the Bouchard-Hébert Mine has been completed and is included in the operating estimates presented in Table

9.3. The Langlois and Caribou Mines remain on Care & Maintenance at this time and therefore have not been included in the table. SRK are of the opinion that BWR will realize additional value once the two mines go into production.

Table 9.3: BWR LRP, Summary of Technical Inputs (all Operating Mines 2002 - 2008)

Model Parameter	Zinc	Lead	Copper	Gold	Silver
Ore Milled (all mines)	11,136Mt				
RoM Grade	7.29%	0.64%	0.17%	0.70g/t	32.66g/t
Zn Equivalent	8.89%				
Contained Metals	811.5kt	71.5kt	18.7kt	249koz	11,694koz
Metallurgical Recovery	89%	90%	79%	54%	92%
Metal In Concentrates	1,591Mlb	141.8Mlb	32.5Mlb	134koz	10,758koz
Payable Metals	1,347Mlb	133Mlb	30.5Mlb	116koz	7,403koz

Projected revenues from operations are based upon the payable values listed in Table 9.3. The projected revenues are in line with historical achievements, and SRK is of the opinion that the values presented in the model are reasonable.

9.2.3 Operating Costs

Historic mine operating costs for fiscal years 1998 to 2000, and estimated mine operating costs for fiscal years 2001 and 2002, are summarized in Table 9.4. BWR projects unit operating costs for combined operations to be US$26.93/t-milled and US$26.64/t-milled in 2001 and 2002, respectively. El Mochito and Nanisivik presently represent the highest unit operating cost mines, while Bouchard-Hébert and El Toqui are currently the lowest unit cost mines.

Operating cost projections for each mine reflect historical achievements. SRK is of the opinion that these operating cost projections are achievable given that historical mine production rates will be maintained during the period of this analysis, thus keeping fixed costs under control (on a unit cost basis).

The major components of the operating costs are all aspects of mining, milling and assaying, plant maintenance, environmental, site administration, port and concentrate handling, off-site administration, and non-capital mine development.

Table 9.4: Historical and Projected Operating Costs (US$)

Mine	1998 Actual	1999 Actual	2000 Actual	2001 Forecast	2002 Budget
Bouchard-Hébert					
Ore Milled (kt)	-	-	543	1,045	1,038
Operating Cost ($000)	-	-	12,000	23,223	24,107
Unit Cost ($/t-milled)	-	-	22.10	22.23	23.22
Nanisivik					
Ore Milled (kt)	792	803	811	773	589
Operating Cost ($000)	21,083	19,930	21,224	23,523	16,470
Unit Cost ($/t-milled)	26.62	24.82	26.17	30.43	27.98
Bougrine					
Ore Milled (kt)	276	430	422	420	436
Operating Cost ($000)	10,557	12,276	11,601	11,630	13,097
Unit Cost ($/t-milled)	38.25	28.55	27.49	27.66	30.07
El Mochito					
Ore Milled (kt)	569	612	638	646	650
Operating Cost ($000)	19,010	19,321	20,946	20,154	19,106
Unit Cost ($/t-milled)	33.41	31.57	32.83	31.19	29.39
El Toqui					
Ore Milled (kt)	258	392	416	425	420
Operating Cost ($000)	10,243	10,686	11,207	10,620	10,674
Unit Cost ($/t-milled)	39.70	27.26	26.94	24.98	25.43
Combined Operations					
Ore Milled (kt)	1,895	2,237	3,246	3,310	3,132
Operating Cost ($000)	60,893	62,213	76,978	89,149	83,455
Unit Cost ($/t-milled)	32.13	27.81	23.71	26.93	26.64

9.2.4 Capital Costs

Historic capital costs for each operating mine for fiscal years 1998 to 2000, and estimated capital costs for fiscal years 2001 and 2002 for each operating mine, are shown in Table 9.5.

Table 9.5: BWR Historical and Projected Capital Cost (US$000)

Mine	1998 Actual	1999 Actual	2000 Actual	2001 Forecast	2002 Budget
Bouchard-Hébert	-	-	1,355	1,302	1,588
Nanisivik	1,414	1,521	5,533	3,293	1,565
Bougrine	8,058	2,610	4,285	1,752	1,176
El Mochito	3,568	2,401	4,137	1,733	1,555
El Toqui	4,210	2,858	6,804	3,491	1,558
Total	**17,250**	**9,390**	**22,114**	**11,571**	**7,442**

In BWR's LRP the operating mines are planned to exhaust their currently defined mineral reserves. For modeling purposes the mines are assumed to close when the present mineral reserves run out. The actual timing of the closure will ultimately depend on the success of each operation to replenish mineral reserves. Several of BWR's mines have a history of identifying additional mineral resources and subsequently converting these mineral resources to mineral reserves, particularly El Mochito and El Toqui. However, as a condition of National Instrument 43-101, only proven and probable reserves can be considered in a model such as the LRP. As such, capital expenditure profiles call for completion of the underground mine development programs, mine and mill equipment requirements and closure and reclamation costs. Also included in the capital cost estimate are estimated salvage value credits. Ongoing exploration programs account for a small portion of planned capital expenditures. BWR LRP capital cost projections for fiscal years 2002 to 2008 are summarized in Table 9.6.

Table 9.6: BWR LRP, Capital Cost Projections 2002 -2008 (US$000)

Mine	Net Total	Mine Development	Mine & Mill Equipment	Reclamation	Exploration	Salvage Credits
Bouchard-Hebert	**3,789**	1,637	1,399	6,106	-	(5,352)
Nanisivik	**7,463**	-	1,447	6,976	-	(960)
Bougrine	**2,368**	833	1,131	977	387	(960)
El Mochito	**10,622**	3,331	9,443	1,954	394	(4,500)
El Toqui	**16,492**	4,012	9,291	2,540	1,149	(500)
Total	**40,734**	**9,813**	**22,711**	**18,553**	**1,930**	**(12,272)**
% of total		**24.1%**	**55.8%**	**45.5%**	**4.7%**	**-30.1%**

9.3 Cash Flow Projections

9.3.1 Fiscal Years 2001 and 2002

Cash flow projections for fiscal years 2001 and 2002 are summarized in Table 9.7. All cash flow projections referred to here are based upon the economic parameters, technical model inputs, estimated operating costs, and estimated capital costs referred to elsewhere in this section 9.0. Mining operations are expected to continue operating under historic operating parameters, and given projected market prices an undiscounted cash flow of US$13.3 million is projected over the 2-year review period.

Net cash outflow in 2001 is projected to be US$6.2 million. Although revenues roughly equal operating expenses in 2001, it is the estimate of capital requirements that results in a negative cash flow for the year. The increase in receivables are projected to exceed the increase in payables in 2002, which is projected to result in a cash flow of approximately US$19.5 million in 2002.

Table 9.7: BWR FY2001-02, Projected Minesite Cash Flow (US$000)

	2001 Forecast	2002 Budget
Total Revenue	94,490	111,101
Minesite Operating Cost	(89,150)	(83,455)
Operating Cash Flow	5,340	27,647
Taxes and Royalties	(9)	(700)
Capital	(11,570)	(7,442)
NET CASH FLOW	**(6,240)**	**19,505**

9.3.2 BWR Long-Range Plan Cash Flow Projections

SRK has completed its review of BWR's LRP and concludes that projections and results presented by BWR are fair and reasonable. LRP cash flow projections for fiscal years 2002 to 2008 are shown in Table 9.8.

Table 9.8: BWR LRP 2002 – 2008 Minesite Cash Flow (US$000)

Discount Rate	BWR LRP NPV
0%	$68,901
8%	$57,511
10%	$55,162
12%	$52,974

Sensitivities on zinc price, operating costs and capital costs were performed on the BWR LRP and are shown in Table 9.9. The project is as sensitive to zinc price as it is to operating costs. The mature nature of these operations requires relatively low capital requirements. As such, there is little sensitivity to capital cost variation.

Table 9.9: BWR LRP 2002 –2008, Cash Flow Model Sensitivity $NPV_{10\%}$(US$million)

Sensitivity Range	Zinc Price	Capital Cost	Operating Cost
-30%	(14.3)	N/A	N/A
-20%	10.6	61.3	102.3
-10%	35.5	58.2	78.7
Base Case	57.5	57.5	57.5
+10%	79.9	52.1	31.6
+20%	102.3	49.0	8.0

It is important to note that the Langlois Mine, as it is not operating, has not been included in the LRP, but based on SRK's feasibility study of August, 2001, the mine yields a total pre-tax cash flow of US$39.0 million at a zinc price of US$0.50/lb. The internal rate of return is 24.0% and the NPV at 10% is US$13.4 million. The project is sensitive to the price of zinc. A +/- 10% change in the zinc price causes a change of US$18.2 million in the pre-tax cashflow.

9.4 Material Issues and Opportunities

SRK finds no material issues associated with technical-economic aspects of the BWR LRP and concludes that the results shown are fair and reasonable. However, SRK suggests that BWR consider the following:

- There is potential to add additional resources at El Mochito and El Toqui. SRK expects that reserves at these two mines will be replaced each year, as has been done in the past.
- There is good exploration potential in the vicinity of the Bouchard-Hébert and Langlois mines.
- The Bougrine Mine may recover less ore than planned from pillars, particularly from the F1 Zone.
- BWR is highly leveraged to the price of zinc.

10. CONCLUDING REMARKS

The views expressed by SRK in this technical report have been based on the fundamental assumptions that the required management resources and pro-active management skills and access to adequate capital necessary to achieve the LoM Plan projections for the mining assets are provided as projected.

SRK has conducted a comprehensive review and assessment of all material issues likely to influence the future operations of BWR's mining assets. The LoM Plans for the mining assets as provided to SRK have been reviewed in detail for appropriateness, reasonableness and viability, including the existence of, and justification for, any departures from historical performance. Where material differences were found, these were discussed with BWR and adjusted where considered appropriate. SRK considers that the resulting projections have been based upon sound reasoning, engineering judgement and achievable mine plans, within the context of the risks associated with the global mining industry.

"Signed"

"Neal Rigby"

Dr. Neal Rigby,

Corporate Mining Consultant,

For and Behalf of

Steffen, Robertson and Kirsten (Canada) Inc.

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GLOSSARY OF TERMS, ABBREVIATIONS AND UNITS

GLOSSARY

The mineral resources and mineral reserves have been classified according to the "*CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines*" (August, 2000). Accordingly, the Resources have been classified as Measured, Indicated or Inferred and the Reserves have been classified as Proven and Probable based on the Measured and Indicated Resources as defined below.

Mineral Resources

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

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

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

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

Mineral Reserves

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

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

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

assay	the chemical analysis of mineral samples to determine the metal content
capital expenditure	all other expenditures not classified as operating costs
composite	combining more than one sample result to give an average result over a larger distance
concentrate	a metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
crushing	initial process of reducing ore particle size to render it more amenable for further processing
cutoff grade	the grade of mineralized rock which determines as to whether or not it is economic to recover its gold content by further concentration
desalination	chemical process of removing salt from contaminated water
dilution	waste which is unavoidably mined with ore
dip	angle of inclination of a geological feature/rock from the horizontal
fault	the surface of a fracture along which movement has occurred
flotation	the process by which the surface chemistry of the desired mineral particles is chemically modified such that they preferentially attach themselves to bubbles and float to the pulp surface in specially designed machines. The gangue or waste minerals are chemically depressed and do not float, thus allowing the valuable minerals to be concentrated and separated from the undesired material
footwall	the underlying side of an orebody or stope
gangue	non-valuable components of the ore
grade	the measure of concentration of gold within mineralized rock
hangingwall	the overlying side of an orebody or slope
haulage	a horizontal underground excavation which is used to transport mined ore;
hydrocyclone	a process whereby material is graded according to size by exploiting centrifugal forces of particulate materials
igneous	primary crystalline rock formed by the solidification of magma
kriging	an interpolation method of assigning values from samples to blocks that minimises the estimation error
lenticular	in the form of elongated lenses
Level	horizontal tunnel the primary purpose is the transportation of personnel and materials
lithological	geological description pertaining to different rock types
LoM Plans	Life-of-Mine plans
LRP	Long Range Plan
Material Properties	mine properties

milling	a general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral / Mining Lease	a lease area for which mineral rights are held
Mining Assets	the Material Properties and Significant Exploration Properties
on-going capital	capital estimates of a routine nature which are necessary for sustaining operations
ore reserve	see Mineral Reserve
pillar	rock left behind to help support the excavations in an underground mine
RoM	Run-of-Mine
sedimentary	pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks
shaft	an opening cut downwards from the surface for transporting personnel, equipment, supplies, ore and waste
sill	a thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness
smelting	a high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase.
stope	underground void created by mining
stratigraphy	study of stratified rocks in terms of time and space
strike	direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction
sulfide	sulfur bearing mineral
tailings	finely ground waste rock from which valuable minerals or metals have been extracted
thickening	process of concentrating solid particles in suspension
total expenditure	all expenditures including those of a operating and capital nature
variogram	statistical representation of the characteristics (usually grade)

UNITS

cm	a centimetre
g	grammes
g/t	grammes per metric tonne – gold concentration
Ha	a Hectare
hrs	hours
k	one thousand units
kg	a kilogram
km	a kilometer
koz	one thousand fine troy ounces
kt	one thousand metric tonnes
m	a metre
m^2	a square metre – measure of area
m^3	a cubic metre
mm	a millimetre
Moz	a million troy ounces
Mt	a million metric tonnes
Mtpa	a million metric tonnes per annum
MW	a million watts
oz	a fine troy ounce equalling 31.10348 grammes
t	a metric tonne
tm-3	density measured as metric tonnes per cubic metre
US$m	a million United States Dollars
US$	United States Dollar
US$/oz	United States Dollars per fine troy ounce
US$/t	United States Dollars per tonne
°	degrees
°C	degrees centigrade
%	percentage

APPENDIX 1

CLAIM INFORMATION

Bouchard-Hebert

Disp Name	Disp Type	Clm Area	Prot to	Project Name	Owner	Twp-County
5016797	Mining Claim	40HA	3/28/05	CLERICY	BWR - 100%	CLERICY
5016798	Mining Claim	40HA	3/28/05	CLERICY	BWR - 100%	CLERICY
5016799	Mining Claim	40HA	3/28/05	CLERICY	BWR - 100%	CLERICY
5016800	Mining Claim	40HA	3/28/05	CLERICY	BWR - 100%	CLERICY
5016801	Mining Claim	40HA	3/28/05	CLERICY	BWR - 100%	CLERICY
5016802	Mining Claim	40HA	3/28/05	CLERICY	BWR - 100%	CLERICY
5016803	Mining Claim	40HA	3/28/05	CLERICY	BWR - 100%	CLERICY
5016804	Mining Claim	40HA	3/28/05	CLERICY	BWR - 100%	CLERICY
TOTAL	8 claims	320HA				
3378831	Mining Claim	40HA	6/30/05	DUFRESNOY	BWR - 100%	DUFRESNOY
3378841	Mining Claim	40HA	6/30/05	DUFRESNOY	BWR - 100%	DUFRESNOY
3382311	Mining Claim	40HA	8/18/03	DUFRESNOY	BWR - 100%	DUFRESNOY
3504082	Mining Claim	40HA	3/13/05	DUFRESNOY	BWR - 100%	DUFRESNOY
3504091	Mining Claim	40HA	3/13/05	DUFRESNOY	BWR - 100%	DUFRESNOY
3504092	Mining Claim	40HA	3/13/05	DUFRESNOY	BWR - 100%	DUFRESNOY
3504101	Mining Claim	40HA	3/13/05	DUFRESNOY	BWR - 100%	DUFRESNOY
3504102	Mining Claim	40HA	3/13/05	DUFRESNOY	BWR - 100%	DUFRESNOY
3504441	Mining Claim	40HA	3/19/05	DUFRESNOY	BWR - 100%	DUFRESNOY
3504442	Mining Claim	40HA	3/19/05	DUFRESNOY	BWR - 100%	DUFRESNOY
4446381	Mining Claim	40HA	2/16/05	DUFRESNOY	BWR - 100%	DUFRESNOY
4446401	Mining Claim	40HA	2/16/05	DUFRESNOY	BWR - 100%	DUFRESNOY
843232	Mining Claim	40HA	11/24/02	DUFRESNOY	BWR - 100%	DUFRESNOY
843281	Mining Claim	36.3HA	11/25/02	DUFRESNOY	BWR - 100%	DUFRESNOY
843282	Mining Claim	40HA	11/25/02	DUFRESNOY	BWR - 100%	DUFRESNOY
843291	Mining Claim	40HA	11/25/02	DUFRESNOY	BWR - 100%	DUFRESNOY
843292	Mining Claim	40HA	11/25/02	DUFRESNOY	BWR - 100%	DUFRESNOY
843511	Mining Claim	40HA	11/25/02	DUFRESNOY	BWR - 100%	DUFRESNOY
843512	Mining Claim	40HA	11/25/02	DUFRESNOY	BWR - 100%	DUFRESNOY
843521	Mining Claim	40HA	11/25/02	DUFRESNOY	BWR - 100%	DUFRESNOY
843522	Mining Claim	20.4HA	11/25/02	DUFRESNOY	BWR - 100%	DUFRESNOY
844501	Mining Claim	20.4HA	11/24/02	DUFRESNOY	BWR - 100%	DUFRESNOY
844502	Mining Claim	18.67HA	11/27/05	DUFRESNOY	BWR - 100%	DUFRESNOY
844511	Mining Claim	18.78HA	11/27/05	DUFRESNOY	BWR - 100%	DUFRESNOY
844512	Mining Claim	18HA	11/27/05	DUFRESNOY	BWR - 100%	DUFRESNOY
844961	Mining Claim	40HA	1/8/03	DUFRESNOY	BWR - 100%	DUFRESNOY
BM767	Mining Lease	53.44HA	6/30/02	DUFRESNOY	BWR - 100%	DUFRESNOY
BM821	Mining Lease	53.46HA	5/16/02	DUFRESNOY	BWR - 100%	DUFRESNOY
P.R.813481	Surface Lease	0.4HA	1/31/02	DUFRESNOY	BWR - 100%	DUFRESNOY
P.R.816030	Surface Lease	62HA	4/30/02	DUFRESNOY	BWR - 100%	DUFRESNOY
P.R.99980	Surface Lease	5HA	12/31/01	DUFRESNOY	BWR - 100%	DUFRESNOY
TOTAL	26 mining claims	932.55HA				
	2 mining leases	106.9HA				
	3 surface leases	67.4HA				
Disp Name	**Disp Type**	**Clm Area**	**Prot to**	**Project Name**	**Owner**	**Twp-County**

4490031	Mining Claim	40HA	4/11/05	JEVIS	BWR - 100%	DUFRESNOY
4490032	Mining Claim	40HA	4/12/05	JEVIS	BWR - 100%	DUFRESNOY
4490041	Mining Claim	40HA	4/11/05	JEVIS	BWR - 100%	DUFRESNOY
4490051	Mining Claim	40HA	4/11/05	JEVIS	BWR - 100%	DUFRESNOY
4490061	Mining Claim	40HA	4/11/05	JEVIS	BWR - 100%	DUFRESNOY
4507601	Mining Claim	40HA	8/27/05	JEVIS	BWR - 100%	DUFRESNOY
4507602	Mining Claim	40HA	8/27/05	JEVIS	BWR - 100%	DUFRESNOY
4507611	Mining Claim	40HA	8/27/05	JEVIS	BWR - 100%	DUFRESNOY
4507612	Mining Claim	40HA	8/27/05	JEVIS	BWR - 100%	DUFRESNOY
4507621	Mining Claim	40HA	8/27/05	JEVIS	BWR - 100%	DUFRESNOY
5052393	Mining Claim	40HA	9/26/05	JEVIS	BWR - 100%	DUFRESNOY
5128912	Mining Claim	40HA	4/20/05	JEVIS	BWR - 100%	DUFRESNOY
5128913	Mining Claim	40HA	4/20/05	JEVIS	BWR - 100%	DUFRESNOY
5141484	Mining Claim	40HA	3/12/05	JEVIS	BWR - 100%	DUFRESNOY
5141485	Mining Claim	40HA	3/12/05	JEVIS	BWR - 100%	DUFRESNOY
5141486	Mining Claim	40HA	3/12/05	JEVIS	BWR - 100%	DUFRESNOY
5141487	Mining Claim	40HA	3/12/05	JEVIS	BWR - 100%	DUFRESNOY
5141488	Mining Claim	40HA	3/12/05	JEVIS	BWR - 100%	DUFRESNOY
TOTAL	18 mining claims	720HA				
3070262	Mining Claim	40HA	7/22/05	KINO	BWR - 100%	CLERICY
3070263	Mining Claim	40HA	7/22/05	KINO	BWR - 100%	CLERICY
3070273	Mining Claim	40HA	8/28/05	KINO	BWR - 100%	CLERICY
3075034	Mining Claim	40HA	8/28/05	KINO	BWR - 100%	CLERICY
3111311	Mining Claim	40HA	8/28/05	KINO	BWR - 100%	CLERICY
3111931	Mining Claim	40HA	9/28/05	KINO	BWR - 100%	CLERICY
3111932	Mining Claim	40HA	9/28/05	KINO	BWR - 100%	CLERICY
3111941	Mining Claim	40HA	9/28/05	KINO	BWR - 100%	CLERICY
3111942	Mining Claim	40HA	9/28/05	KINO	BWR - 100%	CLERICY
3111951	Mining Claim	40HA	9/28/05	KINO	BWR - 100%	CLERICY
3111952	Mining Claim	40HA	9/28/05	KINO	BWR - 100%	CLERICY
3111961	Mining Claim	40HA	9/28/05	KINO	BWR - 100%	CLERICY
3111962	Mining Claim	40HA	9/28/05	KINO	BWR - 100%	CLERICY
3111971	Mining Claim	40HA	9/28/05	KINO	BWR - 100%	CLERICY
3111972	Mining Claim	40HA	9/28/05	KINO	BWR - 100%	CLERICY
3111981	Mining Claim	40HA	9/28/05	KINO	BWR - 100%	CLERICY
3111982	Mining Claim	40HA	9/28/05	KINO	BWR - 100%	CLERICY
3517041	Mining Claim	40HA	3/23/05	KINO	BWR - 100%	CLERICY
3517042	Mining Claim	40HA	3/23/05	KINO	BWR - 100%	CLERICY
3517221	Mining Claim	40HA	3/23/05	KINO	BWR - 100%	CLERICY
3519511	Mining Claim	40HA	3/23/05	KINO	BWR - 100%	CLERICY
3679421	Mining Claim	40HA	10/6/05	KINO	BWR - 100%	CLERICY
3679422	Mining Claim	40HA	10/6/05	KINO	BWR - 100%	CLERICY
3680101	Mining Claim	30HA	10/4/05	KINO	BWR - 100%	CLERICY
3680102	Mining Claim	40HA	10/4/05	KINO	BWR - 100%	CLERICY
3680441	Mining Claim	40HA	10/4/05	KINO	BWR - 100%	CLERICY
3680442	Mining Claim	40HA	10/4/05	KINO	BWR - 100%	CLERICY
3680451	Mining Claim	40HA	10/4/05	KINO	BWR - 100%	CLERICY
Disp Name	**Disp Type**	**Clm Area**	**Prot to**	**Project Name**	**Owner**	**Twp-County**

3680452	Mining Claim	40HA	10/4/05	KINO	BWR - 100%	CLERICY
3682671	Mining Claim	40HA	10/5/05	KINO	BWR - 100%	CLERICY
3682672	Mining Claim	40HA	10/5/05	KINO	BWR - 100%	CLERICY
3682681	Mining Claim	40HA	10/5/05	KINO	BWR - 100%	CLERICY
3682682	Mining Claim	40HA	10/5/05	KINO	BWR - 100%	CLERICY
3682691	Mining Claim	40HA	10/5/05	KINO	BWR - 100%	CLERICY
3682692	Mining Claim	40HA	10/5/05	KINO	BWR - 100%	CLERICY
3682951	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3682952	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3682961	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3682962	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3682971	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3682972	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3705051	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3705052	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3705061	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3705062	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3705071	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3705072	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3705301	Mining Claim	20HA	10/7/05	KINO	BWR - 100%	CLERICY
3705302	Mining Claim	23HA	10/7/05	KINO	BWR - 100%	CLERICY
3705303	Mining Claim	34HA	10/7/05	KINO	BWR - 100%	CLERICY
3705304	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3705305	Mining Claim	20HA	10/7/05	KINO	BWR - 100%	CLERICY
3705311	Mining Claim	20HA	10/7/05	KINO	BWR - 100%	CLERICY
3705312	Mining Claim	20HA	10/7/05	KINO	BWR - 100%	CLERICY
3705313	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3705321	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3705322	Mining Claim	40HA	10/7/05	KINO	BWR - 100%	CLERICY
3801313	Mining Claim	28HA	3/24/05	KINO	BWR - 100%	DUFRESNOY
3801314	Mining Claim	28HA	3/24/05	KINO	BWR - 100%	DUFRESNOY
4046464	Mining Claim	18HA	6/22/05	KINO	BWR - 100%	CLERICY
4046471	Mining Claim	15HA	6/22/05	KINO	BWR - 100%	CLERICY
4046592	Mining Claim	20HA	6/21/05	KINO	BWR - 100%	CLERICY
4092891	Mining Claim	20HA	6/21/05	KINO	BWR - 100%	CLERICY
4092901	Mining Claim	20HA	6/21/05	KINO	BWR - 100%	CLERICY
5028974	Mining Claim	40HA	9/1/05	KINO	BWR - 100%	CLERICY
5050411	Mining Claim	40HA	11/25/02	KINO	BWR - 100%	CLERICY
5050412	Mining Claim	40HA	11/25/02	KINO	BWR - 100%	CLERICY
5050413	Mining Claim	40HA	11/25/02	KINO	BWR - 100%	CLERICY
5050414	Mining Claim	40HA	11/25/02	KINO	BWR - 100%	CLERICY
5050415	Mining Claim	40HA	11/25/02	KINO	BWR - 100%	CLERICY
5050416	Mining Claim	40HA	11/25/02	KINO	BWR - 100%	CLERICY
5135018	Mining Claim	40HA	1/10/06	KINO	BWR - 100%	CLERICY
5135019	Mining Claim	40HA	1/10/06	KINO	BWR - 100%	CLERICY
5135020	Mining Claim	40HA	1/10/06	KINO	BWR - 100%	CLERICY
5135021	Mining Claim	40HA	1/10/06	KINO	BWR - 100%	CLERICY
Disp Name	Disp Type	Clm Area	Prot to	Project Name	Owner	Twp-County

5137581	Mining Claim	40HA	2/21/05	KINO	BWR - 100%	CLERICY
5137582	Mining Claim	40HA	2/21/05	KINO	BWR - 100%	CLERICY
5137583	Mining Claim	40HA	2/21/05	KINO	BWR - 100%	CLERICY
5137584	Mining Claim	40HA	2/21/05	KINO	BWR - 100%	CLERICY
5137585	Mining Claim	40HA	2/21/05	KINO	BWR - 100%	CLERICY
5158831	Mining Claim	40HA	4/16/06	KINO	BWR - 100%	CLERICY
5158832	Mining Claim	40HA	4/16/06	KINO	BWR - 100%	CLERICY
5158833	Mining Claim	40HA	4/16/06	KINO	BWR - 100%	CLERICY
5158834	Mining Claim	40HA	4/16/06	KINO	BWR - 100%	CLERICY
5158835	Mining Claim	40HA	4/16/06	KINO	BWR - 100%	CLERICY
5158836	Mining Claim	40HA	4/16/06	KINO	BWR - 100%	CLERICY
5218941	Mining Claim	40HA	8/27/06	KINO	BWR - 50%	CLERICY
5218942	Mining Claim	40HA	8/27/06	KINO	BWR - 50%	CLERICY
5218944	Mining Claim	40HA	8/27/06	KINO	BWR - 50%	CLERICY
TOTAL	89 mining claims	3316HA				
3951382	Mining Claim	33HA	11/30/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
3955791	Mining Claim	40HA	12/2/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
3955792	Mining Claim	40HA	12/2/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
3955801	Mining Claim	40HA	12/2/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
3955802	Mining Claim	40HA	12/2/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
3955811	Mining Claim	40HA	12/2/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
3955812	Mining Claim	40HA	12/2/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
3955821	Mining Claim	40HA	12/1/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
3955822	Mining Claim	40HA	12/1/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
3955831	Mining Claim	40HA	12/1/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
3955832	Mining Claim	40HA	12/1/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
3955841	Mining Claim	33HA	12/1/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
3955842	Mining Claim	33HA	12/1/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
3955851	Mining Claim	33HA	11/30/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
3955852	Mining Claim	33HA	11/30/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
3955861	Mining Claim	40HA	12/3/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4276401	Mining Claim	40HA	4/10/05	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4276402	Mining Claim	28HA	4/10/05	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4276411	Mining Claim	34HA	4/10/05	RIVIERE DUFRESNOY	BWR - 100%	CLERICY
4276412	Mining Claim	40HA	4/10/05	RIVIERE DUFRESNOY	BWR - 100%	CLERICY
4276421	Mining Claim	40HA	4/11/05	RIVIERE DUFRESNOY	BWR - 100%	CLERICY
4276422	Mining Claim	40HA	4/11/05	RIVIERE DUFRESNOY	BWR - 100%	CLERICY
4276431	Mining Claim	40HA	4/11/05	RIVIERE DUFRESNOY	BWR - 100%	CLERICY
4276432	Mining Claim	40HA	4/11/05	RIVIERE DUFRESNOY	BWR - 100%	CLERICY
4349591	Mining Claim	33HA	10/30/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4349601	Mining Claim	40HA	9/27/05	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4349611	Mining Claim	40HA	9/27/05	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4349612	Mining Claim	33HA	10/30/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4352621	Mining Claim	33HA	10/30/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4352622	Mining Claim	33HA	10/31/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4352631	Mining Claim	33HA	10/30/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
Disp Name	**Disp Type**	**Clm Area**	**Prot to**	**Project Name**	**Owner**	**Twp-County**
4352632	Mining Claim	33HA	10/31/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY

4352641	Mining Claim	33HA	10/30/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4352642	Mining Claim	33HA	10/31/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4352651	Mining Claim	33HA	10/30/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4352652	Mining Claim	33HA	10/31/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4352661	Mining Claim	40HA	10/31/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4354671	Mining Claim	40HA	11/27/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4354672	Mining Claim	40HA	11/27/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4354681	Mining Claim	40HA	11/27/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4354682	Mining Claim	40HA	11/27/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4354691	Mining Claim	40HA	11/28/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4354692	Mining Claim	40HA	11/28/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4354711	Mining Claim	40HA	11/27/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4354712	Mining Claim	40HA	11/27/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4354721	Mining Claim	40HA	11/27/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4385931	Mining Claim	40HA	7/24/03	RIVIERE DUFRESNOY	BWR - 100%	DESTOR
4446341	Mining Claim	40HA	2/16/05	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4446352	Mining Claim	40HA	2/15/05	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4446511	Mining Claim	40HA	2/15/05	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4446512	Mining Claim	28HA	2/15/05	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4446521	Mining Claim	40HA	2/19/05	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4446551	Mining Claim	40HA	2/16/05	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4446561	Mining Claim	40HA	2/15/05	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4446582	Mining Claim	40HA	2/19/05	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
4505931	Mining Claim	40HA	7/24/03	RIVIERE DUFRESNOY	BWR - 100%	DESTOR
4505932	Mining Claim	40HA	7/24/03	RIVIERE DUFRESNOY	BWR - 100%	DESTOR
4505941	Mining Claim	40HA	7/24/03	RIVIERE DUFRESNOY	BWR - 100%	DESTOR
5067642	Mining Claim	35HA	2/17/05	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
5094237	Mining Claim	40HA	4/15/03	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
5094238	Mining Claim	40HA	4/15/03	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
5218558	Mining Claim	40HA	12/21/01	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
5218559	Mining Claim	40HA	12/21/01	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
5218560	Mining Claim	40HA	12/21/01	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
5218861	Mining Claim	40HA	12/21/01	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
5218864	Mining Claim	40HA	12/21/01	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
5219470	Mining Claim	16HA	3/1/02	RIVIERE DUFRESNOY	BWR - 100%	DESTOR
5219524	Mining Claim	40HA	12/21/01	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
5219525	Mining Claim	40HA	12/21/01	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
5223446	Mining Claim	33HA	2/18/02	RIVIERE DUFRESNOY	BWR - 100%	DESTOR
5223447	Mining Claim	40HA	2/18/02	RIVIERE DUFRESNOY	BWR - 100%	DESTOR
5223448	Mining Claim	40HA	2/18/02	RIVIERE DUFRESNOY	BWR - 100%	DESTOR
5223449	Mining Claim	38HA	2/18/02	RIVIERE DUFRESNOY	BWR - 100%	DESTOR
5223531	Mining Claim	29HA	2/18/02	RIVIERE DUFRESNOY	BWR - 100%	DESTOR
5223532	Mining Claim	24HA	2/18/02	RIVIERE DUFRESNOY	BWR - 100%	DESTOR
5223533	Mining Claim	34HA	2/18/02	RIVIERE DUFRESNOY	BWR - 100%	DESTOR
5223534	Mining Claim	40HA	2/18/02	RIVIERE DUFRESNOY	BWR - 100%	DESTOR
5223535	Mining Claim	40HA	2/18/02	RIVIERE DUFRESNOY	BWR - 100%	DESTOR
Disp Name	**Disp Type**	**Clm Area**	**Prot to**	**Project Name**	**Owner**	**Twp-County**
5223540	Mining Claim	16HA	3/1/02	RIVIERE DUFRESNOY	BWR - 100%	DESTOR

5223541	Mining Claim	27 HA	2/18/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
5223542	Mining Claim	38 HA	2/18/02	RIVIERE DUFRESNOY	BWR - 100%	DUFRESNOY
5226958	Mining Claim	5 HA	4/30/02	RIVIERE DUFRESNOY	BWR - 100%	DESTOR
TOTAL	81 mining claims	3000 HA				

Nanisivik

Disp Name	Disp Type	Clm Area	Project Name	Owner
2451	Mining Lease	6833.88AC	NANISIVIK	CANZINCO LTD.
2452	Mining Lease	1060.94AC	NANISIVIK	CANZINCO LTD.
2799	Mining Lease	609AC	NANISIVIK	CANZINCO LTD.
2800	Mining Lease	51AC	NANISIVIK	CANZINCO LTD.
2801	Mining Lease	370AC	NANISIVIK	CANZINCO LTD.
2802	Mining Lease	66.2AC	NANISIVIK	CANZINCO LTD.
2803	Mining Lease	407AC	NANISIVIK	CANZINCO LTD.
2804	Mining Lease	278AC	NANISIVIK	CANZINCO LTD.
2875	Mining Lease	132.7AC	NANISIVIK	CANZINCO LTD.
2876	Mining Lease	342AC	NANISIVIK	CANZINCO LTD.
2877	Mining Lease	372.4AC	NANISIVIK	CANZINCO LTD.
2905	Mining Lease	861.1AC	NANISIVIK	CANZINCO LTD.
3268	Mining Lease	359.61AC	NANISIVIK	CANZINCO LTD.
3269	Mining Lease	227.42AC	NANISIVIK	CANZINCO LTD.
3317	Mining Lease	1356.9AC	NANISIVIK	CANZINCO LTD.
3379	Mining Lease	1853AC	NANISIVIK	CANZINCO LTD.
3383	Mining Lease	1038AC	NANISIVIK	CANZINCO LTD.
TOTAL	17 Min. leases	16219.15AC		

Disp Name	Disp Type	Clm Area	Project Name	Owner
48-C/1-10-2	Surface Lease		NANISIVIK	CANZINCO LTD.
48-C/1-5-2	Surface Lease		NANISIVIK	CANZINCO LTD.
48-C/1-6-2	Surface Lease		NANISIVIK	CANZINCO LTD.
48-C/1-7-2	Surface Lease		NANISIVIK	CANZINCO LTD.
48-C/1-8-3	Surface Lease		NANISIVIK	CANZINCO LTD.
48-C/1-9-3	Surface Lease		NANISIVIK	CANZINCO LTD.
8008T	Surface Lease		NANISIVIK	CANZINCO LTD.
8677T	Surface Lease		NANISIVIK	CANZINCO LTD.
DL-40041T	Surface Lease		NANISIVIK	CANZINCO LTD.
DL-40042T	Surface Lease		NANISIVIK	CANZINCO LTD.
DL-40043T	Surface Lease		NANISIVIK	CANZINCO LTD.
DL-40044T	Licence		NANISIVIK	CANZINCO

				LTD.
DL-40163T	Licence		NANISIVIK	CANZINCO LTD.

TOTAL 13 Sur. leases

Disp Name	Disp Type	Clm Area	Project Name	Owner
BB 1	Mineral Claim	2479.2AC	NANISIVIK	CANZINCO LTD.
BB 2	Mineral Claim	2479.2AC	NANISIVIK	CANZINCO LTD.
EB 1	Mineral Claim	2479.2AC	NANISIVIK	CANZINCO LTD.
EB 2	Mineral Claim	2479.2AC	NANISIVIK	CANZINCO LTD.
GULL	Mineral Claim	1267AC	NANISIVIK	CANZINCO LTD.
KL 1	Mineral Claim	1291.3AC	NANISIVIK	CANZINCO LTD.
KL 2	Mineral Claim	425AC	NANISIVIK	CANZINCO LTD.
NB 2	Mineral Claim	2479.2AC	NANISIVIK	CANZINCO LTD.
NB 4	Mineral Claim	2479.2AC	NANISIVIK	CANZINCO LTD.

TOTAL 9 Mineral claim 17858.5AC

Bougrine

Breakwater Tunisia S.A.: Property Description

Mining Lease of the Bougrine Mine

All Mineral Resources and Mineral Reserves reported for the Bougrine orebody and all underground mine openings fall into the limits of a single mining title, the so-called Bougrine mining concession (*Concession de substances minérales du 3ème groupe No. 598.321 au lieu dite 'Jebel Bougrine'*).

The mining concession is located in the township (*délégation*) of Sers, province (*gouvernorat*) of Kef, Republic of Tunisia. The centre of the mining lease is at about 36°06'10"N, 8°56'40"E.

Register number with the Tunisian "*Directeur General des Mines*" is 598.321.

The concession has a surface of 400 hectares.

The shape is a square of 2kilometres by 2kilometres defined as follows:

Reference point: Topographic point "Koudiat El Guenaoua" on topographic map sheet "Les Salines", scale 1:50000. Latitude = $40^{G}08'30''$, longitude = $7^{G}29'30''$ [old "French" longitude as marked on map sheet], elevation = 775m.

Northern limit: line W-E passing 4550m north of the reference point,
Eastern limit: line N-S passing 5250m east of reference point,
Southern limit: line E-W passing 2550m north of reference point,
Western limit: line S-N passing 3250m east of reference point.

This translates approximately[1] into the following corner co-ordinates:

North-West corner: N 312,750 / E 413,000 North-East corner:N 312,750 / E 415,000
South-West corner: N 310,750 / E 413,000 South-East corner:N 310,750 / E 415,000

[1] The word "approximately" is used as the location of the reference point is simply read of a 1:50,000 scale map sheet and reported in the original documents by longitude and latitude. It is quite likely that this reference point does not fall accurately on the assumed Lambert co-ordinates N 308,500 /
E 409,750, but deviates by a few metres from these round figures.

The regional grid (which is also used as mine grid) is the "*Quadrilage kilometrique Lambert Nord Tunisie (CARTHAGE)*". Origin is 40^G North (=36°N), 11^G East of Greenwich (=10°E) with x_0 = 500,000 and y_0 = 300,000.

The concession was issued on the name of Société Minière de Bougrine by the Minister of National Economy on 6 May 1992 and published in the "*Journal Officiel de la République Tunisienne*" (JORT) from 19 May 1992. It has a validity of 50 years.

This mining lease was transferred to Breakwater Tunisia S.A. upon signature of a contract between BWT and the Tunisian State on 21 July 1997 ("*Concession de Jebel Bougrine – Convention et Annexes entre l'Etat Tunisien et la Société Breakwater S.A.*").

The *Convention* was approved by the Tunisian Government as law 97-73 on 18 November 1997 and has by itself a validity of 50 years. It set basic rules for import and export of funds, materials and the produced goods, for the employment of expatriate personnel, describes the tax regime and governs the possible change of ownership of the mining lease.

Exploitation of the deposit is governed by a set of rules ("*Cahier de Charges*") forming part of the above-mentioned contract and by the Mining Act from 1953 (as long as not in contradiction with the *Convention*, which overrules the Mining Act in case of conflict).

The "*Cahier de Charges*" calls for a minimum yearly production rate of 200,000mt for the first five years following the re-activation of the mine by Breakwater and subsequently for a minimum "yearly tonnage adapted to the reserve situation". Unless certain conditions are fulfilled, the "*Cahier de Charges*" obliges the operator also to undertake minimum yearly exploration efforts.

State royalty and tax payments are governed by the *Convention* between BWT and the Tunisian State. There are no third-party royalties or payments to be made.

Environmental liabilities and environmental obligations are controlled by a protocol signed on 30 August 1991 between the previous operator (Société Minière de Bougrine) and the Tunisian *Agence National de Protection de l'Environnement* (A.N.P.E.): "*Mesures Obligatoires pour la Protection de l'Environnement à mettre en œuvre pour l'Exploitation de la Mine de Bougrine*".

Apart from the *Convention*, a number of permits are required to effectively operate the Bougrine mine, namely for the storage and use of explosives, for the exploitation of groundwater resources and for the operation of the backfill quarry. Quarry and water wells are outside of the mining concession. All required permits have been obtained and are current.

<u>Exploration Permits</u>

Including the Bougrine mining lease, the Company holds 80km² of permit areas for exploration in two diapir structures (Djebel Lorbeus and Djebel Kebbouch) in a 15km-radius around the mine site in NW Tunisia.

All permits are located in the province (*gouvernorat*) of Kef and fall into three different townships (*délégations*): Sers, Dahmani and Nebeur.

The "Jebel Kebbouch" permit (site of the former Kebbouch-Sud mine, surface = 400 hectares) was obtained by BWT in connection with the sale of the SMB assets in order to prevent a third party from applying for what was described by O.N.M. as a potential source of additional mill feed for the Bougrine plant.

In 1999, an application was filed for twenty contiguous permits covering the entire Djebel Lorbeus diapir with exception of the Bougrine mining concession and a 4km²-area at the Lorbeus mine already obtained by a British "promoter". Total permit area is 7200 hectares.

The following exploration permits ("*Permis de recherche de substances minérales du 3ème groupe*") are held by Breakwater Tunisia S.A. as of 31 October 2001:

permit designation	surface [hectares]	register no.	application	issued	published [1]	validity
1.) Lorbeus Area						
Koudiat en Nouesser	325	642.090	01-Sep-99	08-Dec-99	21-Dec-99	07-Dec-02
Argoub ez Zebbouz	325	642.091	01-Sep-99	08-Dec-99	21-Dec-99	07-Dec-02
Koudiat el Houireb	400	642.092	01-Sep-99	08-Nov-99	26-Nov-99	07-Nov-02
Koudiat Tabet Serdouk	275	642.093	01-Sep-99	08-Dec-99	21-Dec-99	07-Dec-02
Koudiat Mellaha	275	642.094	01-Sep-99	08-Dec-99	21-Dec-99	07-Dec-02
Ain el Messai	400	642.095	01-Sep-99	08-Nov-99	26-Nov-99	07-Nov-02
Koudiat el Guenaoua	400	642.096	01-Sep-99	08-Nov-99	26-Nov-99	07-Nov-02
Koudiat el Guenaoua - Est	301	642.097	01-Sep-99	05-Jan-00	21-Jan-00	04-Jan-03
Ain el Morra	302	642.098	01-Sep-99	05-Jan-00	21-Jan-00	04-Jan-03
Les Salines	400	642.099	01-Sep-99	08-Nov-99	26-Nov-99	07-Nov-02
Sidi Bou Sradig	400	642.100	01-Sep-99	08-Nov-99	26-Nov-99	07-Nov-02
Sidi M'Sid	400	642.101	01-Sep-99	08-Nov-99	26-Nov-99	07-Nov-02
Koudiat el Kechrid - Est	400	642.102	01-Sep-99	08-Nov-99	26-Nov-99	07-Nov-02
Koudiat Sfaia	298	642.103	01-Sep-99	05-Jan-00	21-Jan-00	04-Jan-03
Sidi Bou Hadjadja	299	642.104	01-Sep-99	05-Jan-00	21-Jan-00	04-Jan-03
Bled el Bania	400	642.105	01-Sep-99	08-Nov-99	26-Nov-99	07-Nov-02
Sidi Bou Sliâa	400	642.106	01-Sep-99	08-Nov-99	26-Nov-99	07-Nov-02
Djebel Argoub er Ras	400	642.107	01-Sep-99	08-Nov-99	26-Nov-99	07-Nov-02
Koudiat Hanich	400	642.108	01-Sep-99	08-Nov-99	26-Nov-99	07-Nov-02
Koudiat Hanich - Est	400	642.109	01-Sep-99	08-Nov-99	26-Nov-99	07-Nov-02
2.) Kebbouch Area						
Jebel Kebbouch	400	636.020	24-Nov-97	03-Feb-98	13-Feb-98	02-Feb-01
	2nd 3yr-term:	647.357	01-Dec-00	27-Feb-01	09-Mar-01	02-Feb-04

[1] Date of the relevant "*Journal Officiel de la République Tunisienne*" (JORT).

The official definition of the permit areas in relation to the reference points is given in the issues of the "*Journal Officiel de la République Tunisienne*" dated 13 February 1998, 26 November 1999, 21 December 1999 and 21 January 2000 (as indicated above)

Reference point for the "Jebel Kebbouch" permit is the topographic point "Signal du Djebel Kebbouch", map sheet "Nebeur", scale 1:50000. Latitude = $40^{G}23'26''$ North, longitude = $7^{G}32'68''$ East (old French system). For the permits in the Lorbeus are, the same reference point as for the Bougrine mining lease has been used: Topographic point

"Koudiat El Guenaoua", map sheet "Les Salines", scale 1:50000. Latitude = $40^G08'30''$ North, longitude = $7^G29'30''$ East (old French system).

These definitions translate approximately[1] into the following corner co-ordinates ("*Lambert Nord Tunisie*", clock-wise, starting at the upper left (north-western) corner):

permit designation	register no.	1st point	2nd point	3rd point	4th point	5th point	6th point
Koudiat en Nouesser	642.090	N 314,000 E 412,000	N 314,000 E 414,000	N 312,750 E 414,000	N 312,750 E 415,000	N 312,000 E 413,000	N 312,000 E 412,000
Argoub ez Zebbouz	642.091	N 314,000 E 414,000	N 314,000 E 416,000	N 312,000 E 416,000	N 312,000 E 415,000	N 312,750 E 415,000	N 312,750 E 414,000
Koudiat el Houireb	642.092	N 312,000 E 410,000	N 312,000 E 412,000	N 310,000 E 412,000	N 310,000 E 410,000		
Koudiat Tabet Serdouk	642.093	N 312,000 E 412,000	N 312,000 E 413,000	N 310,750 E 413,000	N 310,750 E 414,000	N 310,000 E 414,000	N 310,000 E 412,000
Koudiat Mellaha	642.094	N 310,750 E 414,000	N 310,750 E 415,000	N 312,000 E 415,000	N 312,000 E 416,000	N 310,000 E 416,000	N 310,000 E 414,000
Ain el Messai	642.095	N 310,000 E 406,000	N 310,000 E 408,000	N 308,000 E 408,000	N 308,000 E 406,000		
Koudiat el Guenaoua	642.096	N 310,000 E 408,000	N 310,000 E 410,000	N 308,000 E 410,000	N 308,000 E 408,000		
Koudiat el Guenaoua - Est	642.097	N 310,000 E 410,000	N 310,000 E 412,000	N 308,984 E 412,000	N 308,984 E 410,993	N 308,000 E 410,993	N 308,000 E 410,000
Ain el Morra	642.098	N 310,000 E 412,000	N 310,000 E 414,000	N 308,000 E 414,000	N 308,000 E 412,993	N 308,984 E 412,993	N 308,984 E 412,000
Les Salines	642.099	N 310,000 E 414,000	N 310,000 E 416,000	N 308,000 E 416,000	N 308,000 E 414,000		
Sidi Bou Sradig	642.100	N 308,000 E 404,000	N 308,000 E 406,000	N 306,000 E 406,000	N 306,000 E 404,000		
Sidi M'Sid	642.101	N 308,000 E 406,000	N 308,000 E 408,000	N 306,000 E 408,000	N 306,000 E 406,000		
Koudiat el Kechrid - Est	642.102	N 308,000 E 408,000	N 308,000 E 410,000	N 306,000 E 410,000	N 306,000 E 408,000		
Koudiat Sfaia	642.103	N 308,000 E 410,000	N 308,000 E 410,993	N 306,984 E 410,993	N 306,984 E 412,000	N 306,000 E 412,000	N 306,000 E 410,000
Sidi Bou Hadjadja	642.104	N 306,984 E 412,000	N 306,984 E 412,993	N 308,000 E 412,993	N 308,000 E 414,000	N 306,000 E 414,000	N 306,000 E 412,000
Bled el Bania	642.105	N 308,000 E 414,000	N 308,000 E 416,000	N 306,000 E 416,000	N 306,000 E 414,000		
Sidi Bou Sliâa	642.106	N 306,000 E 404,000	N 306,000 E 406,000	N 304,000 E 406,000	N 304,000 E 404,000		
Djebel Argoub er Ras	642.107	N 306,000 E 406,000	N 306,000 E 408,000	N 304,000 E 408,000	N 304,000 E 406,000		
Koudiat Hanich	642.108	N 306,000 E 408,000	N 306,000 E 410,000	N 304,000 E 410,000	N 304,000 E 408,000		
Koudiat Hanich - Est	642.109	N 306,000 E 410,000	N 306,000 E 412,000	N 304,000 E 412,000	N 304,000 E 410,000		
Jebel Kebbouch	1st: 636.020 now: 636.021	N 323,400 E 411,500	N 323,400 E 413,500	N 321,400 E 413,500	N 321,400 E 411,500		

(1)The word "approximately" is used as the location of the reference points is read of 1:50,000 scale map sheets and reported in the original documents by longitude and latitude. It is quite likely that this reference point does not fall accurately on the assumed Lambert co-ordinates, but deviates by a few metres from the round figures.

Standard permit size is 2 by 2 kilometres, unless the permit overlaps with another earlier issued permit. Permits are valid for three years and can be renewed for further 3-year terms (at last twice, in practice probably indefinitely). However, to obtain a prolongation, compliance with the minimum work programme indicated in the initial application has to be demonstrated. There are no acreage fees, only very modest fees for the application (and the application for the renewal) itself (about US$ 12 per permit).

Upon demand, the exploration permit is transferred into a mining lease, which has the same extend as the exploration permit. The conditions of a mining permit would be negotiated on a case to case basis and fixed in a "*Convention*" and a "*Cahier de Charge*". The current *Convention* for the Bougrine project indicates that additional mining leases would have initially a duration of 30 years.

El Mochito

The claim group documents are in Spanish and have not been reproduced here. The following was provided by BWR:

Fees were paid on 10,836 hectares in 2000 for the "El Mochito" Claim Group.

The Mochito claim block is an irregular shaped group of claims with approximate dimensions of 12 x 9 kilometres, centered on UTM coordinates 383085E-1642087N. In addition, there are 10 claims in 4 groups, to the west of the Mochito claims that are considered part of the El Mochito concessions.

The claims are classified in Honduras as exploitation concessions. In 1999 a new Mining Code was enacted, greatly simplifying the concession acquisition process and eliminates the 20 Ha limit to the size of the individual claims; however it is advantageous to maintain our original claims and their unique status.

AMPAC SA de CV has sole title to these exploitation concessions, which are 40-yr concessions expiring (however subject to renewal) in 2027. In addition, AMPAC has legal title to extensive surface lands covering a portion of the El Mochito zone of claim concessions; these surface lands being necessary for worker camps, surface mine and mill facilities, present and future tails dams, exploration activity and water springs. In 2000, American Pacific Honduras paid US$2,709 as 'canon territorial' for the El Mochito concession. Additionally, AMPAC must submit a report annually detailing activities undertaken during the year on the claims, including production, employment, capital investment and accounting balance sheets.

Our attorneys consider that Ampac is "grandfathered" with regard to the new (1999) Mining Code, and not retroactively subject to environmental legislation enacted in 1995. Future tailings dams or mining activity which is not an expansion of existing facilities would be subject to EIS and closure bonds. Exploration activity, whatever the phase, is exempt by executive decree of any environmental permitting.

Concession	Municipality	Area (hectares)	Secretary of State - Register of Mining Zones / Natural Resources #	Property Register #
El Mochito Group				
El Mochito	Zacapa	199	31	7663
El Caliche	San Pedro de Zacapa	199	67	8220
El Porvenir	Zacapa	200	66	418
Los Andes	Zacapa	199	35	
Moreno	Zacapa	199		12440
Yojoa	Zacapa	199		12436
Novillo	Zacapa	99		12438
Soledad	Zacapa	85		12437
Sauce	Zacapa	199		12439
Lopez	Zacapa	199		423
Nispero	Zacapa	200		424
Concepcion	Zacapa	200		419
Dantillo	Zacapa	200		425
Fortuna	Zacapa	199		420
Paraiso	Zacapa	199		417
El Cedral	Zacapa	200		422
El Palmar	Zacapa	199		427
El Robledal	Zacapa	199		421
El Ocotal	Zacapa	199		426
Pacifico	Zacapa	198		436
Tina	Zacapa	197	NR 437	
Zona Hondo	Zacapa	199	NR 439	
Silencio	Zacapa	200	NR 439	
Las Vegas	Zacapa	200	NR 441	
La Pina	Zacapa	187	NR 440	
El Plan	Zacapa	199	NR 258	
Escarpado	Zacapa	200	NR 256	
Quebrada	Zacapa	99	NR 257	
El Aguilar	Zacapa	199	NR 225	
Atlantico	Zacapa	199	NR 254	
Caminos	Zacapa	199	NR 249	
Transmission	Zacapa	199	NR 250	
La Garrapata	Zacapa	201	NR 251	
America	Zacapa	199	NR 252	
Caribe	Zacapa	200	NR 253	
El Lago	Zacapa	199	NR 259	
Agua Blanca	Zacapa	199	NR 260	
El Verde	Zacapa	156	NR 261	
Calcita	Zacapa	199	NR 262	
Barranca	Zacapa	200	NR 263	
El Triste	Zacapa	199	NR 248	
La Muralla	Zacapa	200	NR 247	
Laguna Verde	Concepción del Sur	172	NR 314	
Los Santos	Concepción del Sur	199	NR 315	
Santa Ana	Concepción del Sur	200	NR 316	
Los Bancos	Santa Barbara	200	NR 317	

Concession	Municipality	Area (hectares)	Secretary of State - Register of Mining Zones / Natural Resources #	Property Register #
Monte Picado	Santa Barbara	199	NR 318	
Los Anices	Santa Barbara	199	NR 319	
Quinientos	Santa Barbara	200	NR 323	
Taixiguat	Concepción del Sur	200	NR 322	
Los Sarritos	Santa Barbara	200	NR 321	
Rio Chiquito	Santa Barbara	199	NR 320	
Lucia	Zacapa & Concepción del Sur	194	NR 1043	319
El Nispero Group				
Patricia	Santa Barbara	199	NR 1126	
Viviana	Santa Barbara	199	NR 1125	
Joconal	Santa Rita & El Nispero	199	NR 1127	
Luisa	Santa Barbara	197	NR 1130	
Leticia	Lempira	199	NR 1129	
Rosario & Aurora Group	San Juancito	999		Executive Power of May 12, 1909
Minera Escobales	San Juancito	950		Executive Power of July 22, 1909
Minera La Esperanza #6	Valle de Angeles	200	#1778 Executive Power of July 22, 1909	222
Remedida de Zonas 1939, including:		2403	Title delivered in Tegucigalpa Folio 124 to 154	
Guadalupe San Juan de Caridad		24 87		

El Toqui

Project Area	Exploitation Concessions		Exploration Concessions	
	Number	Hectares	Number	Hectares
Toquí District	19	13,792	29	8,000
Lago Gral. Carrera (incl. Puerto Sanchez)	15	7,271	37	9,900
Cerro Domo			14	4,200
Malito	11	3,100		
Limite			6	1,800
TOTAL	45	24,163	86	23,900

PATENTES MINERAS CONCESIONES DE EXPLOTACION
SOCIEDAD CONTRACTUAL MINERA EL TOQUI
2001

PERTENENCIAS	ROL NACIONAL	CONSECIONARIO	UBICACION	HÁS
Mañihuales 1/800	11101-0002-3	S.C.M. El Toqui	Co. Estatuas	1,750
Mañihuales Alto 1/800	11101-0001-5	S.C.M. El Toqui	Río Toqui	1,490
Arroyo Concordia 1/250	11101-0003-1	S.C.M. El Toqui	Río Toqui	930
Carlota 1/500	11101-0004-K	S.C.M. El Toqui	Río Toqui	60
Angelika 1/500	11103-0030-8	S.C.M. El Toqui	Río Toqui	930
Angelika 1/501	11101-0005-8	S.C.M. El Toqui	Río Toqui	420
Toqui 1/700	11101-0006-6	S.C.M. El Toqui	Río Toqui	60
Machi 16 1 al 21	11401-0213-6	S.C.M. El Toqui	Est. Sn. Antonio	196
Machi 17 1 al 30	11401-0214-4	S.C.M. El Toqui	Est. Sn. Antonio	300
Machi 18 1 al 7	11401-0215-2	S.C.M. El Toqui	Est. Porvenir	70
Machi 19 1 al 28	11401-0216-0	S.C.M. El Toqui	Est. Sn. Antonio	260
Machi 20 1 al 30	11401-0217-9	S.C.M. El Toqui	Est. Porvenir	298
Machi 21 1 al 30	11401-0218-7	S.C.M. El Toqui	Est. Sn. Antonio	300
Machi 29 1 al 21	11401-0219-5	S.C.M. El Toqui	Est. Porvenir	208
Machi 30 1 al 30	11401-0220-9	S.C.M. El Toqui	Est. Sn. José	300
Gato 1/1380	11101-0009-0	S.C.M. El Toqui	Río Gato	315
Gato 1/1381	11103-0040-5	S.C.M. El Toqui	Río Gato	105
Gato Negro 28 1/40	11101-0024-4	S.C.M. El Toqui	Río E. Guillermo	400
Gato Negro 29 1/20	11101-0025-2	S.C.M. El Toqui	Río E. Guillermo	200
Gato Negro 30 1/20	11101-0026-0	S.C.M. El Toqui	Río E. Guillermo	200
Gato Negro 35 1/20	11101-0027-9	S.C.M. El Toqui	Río E. Guillermo	200
Gato Negro 36 1/50	11101-0028-7	S.C.M. El Toqui	Río E. Guillermo	500
Gato Negro 37 1/80	11101-0029-5	S.C.M. El Toqui	Río E. Guillermo	800
Gato Negro 41 1/20	11101-0030-9	S.C.M. El Toqui	Río E. Guillermo	200
PERTENENCIAS	ROL NACIONAL	CONSECIONARIO	UBICACION	HÁS

Gato Negro 47 1/10	11101-0031-7	S.C.M. El Toqui	Río E. Guillermo	100
Gato Negro 48 1/10	11101-0032-5	S.C.M. El Toqui	Río E. Guillermo	100
Alfa 1/30	11101-0019-8	S.C.M. El Toqui	Río E. Guillermo	300
Muriel 11/15 y 26/30	11101-0020-1	S.C.M. El Toqui	Río E. Guillermo	100
Katerfeld 1/1000	11103-0032-4	S.C.M. El Toqui	Río Ñirehuao	825
Condor 1/995	11402-0003-0	S.C.M. El Toqui	La Tapera	4,975
TOTAL			Hás	16,892

Exploration Consessions

VER	NORT	EST	TIPO	NOMROL	BOLSENT	FECHSENT	JUGA	NOMMAN	FECHCERT
V-1	5014000.000	268000.000	P	NUEVO 10 1.311	5099	18-10-99	II -COYHAIQUE	S.C.M. EL TOQUI	11-06-99
V-2	5014000.000	270000.000	P	NUEVO 10 1.311	5099	18-10-99	II -COYHAIQUE	S.C.M. EL TOQUI	11-06-99
V-3	5013000.000	270000.000	P	NUEVO 10 1.311	5099	18-10-99	II -COYHAIQUE	S.C.M. EL TOQUI	11-06-99
V-4	5013000.000	268000.000	P	NUEVO 10 1.311	5099	18-10-99	II -COYHAIQUE	S.C.M. EL TOQUI	11-06-99
V-1	5012000.000	297000.000	P	NUEVO 11 1.312	5099	18-10-99	II -COYHAIQUE	S.C.M. EL TOQUI	11-06-99
V-2	5012000.000	298000.000	P	NUEVO 11 1.312	5099	18-10-99	II -COYHAIQUE	S.C.M. EL TOQUI	11-06-99
V-3	5010000.000	298000.000	P	NUEVO 11 1.312	5099	18-10-99	II -COYHAIQUE	S.C.M. EL TOQUI	11-06-99
V-4	5010000.000	297000.000	P	NUEVO 11 1.312	5099	18-10-99	II -COYHAIQUE	S.C.M. EL TOQUI	11-06-99
V-1	5017000.000	264000.000	P	NUEVO 20 1.321	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-2	5017000.000	267000.000	P	NUEVO 20 1.321	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-3	5016000.000	267000.000	P	NUEVO 20 1.321	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-4	5016000.000	264000.000	P	NUEVO 20 1.321	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-1	5016000.000	264000.000	P	NUEVO 21 1.322	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-2	5016000.000	267000.000	P	NUEVO 21 1.322	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-3	5015000.000	267000.000	P	NUEVO 21 1.322	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-4	5015000.000	264000.000	P	NUEVO 21 1.322	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-1	5015000.000	268000.000	P	NUEVO 22 1.323	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-2	5015000.000	270000.000	P	NUEVO 22 1.323	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-3	5014000.000	270000.000	P	NUEVO 22 1.323	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-4	5014000.000	268000.000	P	NUEVO 22 1.323	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-1	5015000.000	295000.000	P	NUEVO 31 1.332	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-2	5015000.000	296000.000	P	NUEVO 31 1.332	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-3	5013000.000	296000.000	P	NUEVO 31 1.332	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-4	5013000.000	295000.000	P	NUEVO 31 1.332	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-1	5015000.000	297000.000	P	NUEVO 32 1.333	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-2	5015000.000	298000.000	P	NUEVO 32 1.333	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-3	5012000.000	298000.000	P	NUEVO 32 1.333	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-4	5012000.000	297000.000	P	NUEVO 32 1.333	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-1	5016000.000	295000.000	P	NUEVO 33 1.334	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-2	5016000.000	298000.000	P	NUEVO 33 1.334	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
V-3	5015000.000	298000.000	P	NUEVO 33 1.334	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99
VER	NORT	EST	TIPO	NOMROL	BOLSENT	FECHSENT	JUGA	NOMMAN	FECHCERT
V-4	5015000.000	295000.000	P	NUEVO 33 1.334	5115	09-12-99	II -COYHAIQUE	S.C.M. EL TOQUI	17-08-99

V-1	4972000.000	282000.000	P	NUEVO CERRO DOMO UNO 1.344	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-2	4972000.000	285000.000	P	NUEVO CERRO DOMO UNO 1.344	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-3	4971000.000	285000.000	P	NUEVO CERRO DOMO UNO 1.344	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-4	4971000.000	282000.000	P	NUEVO CERRO DOMO UNO 1.344	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-1	4971000.000	282000.000	P	NUEVO CERRO DOMO DOS 1.345	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-2	4971000.000	285000.000	P	NUEVO CERRO DOMO DOS 1.345	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-3	4970000.000	285000.000	P	NUEVO CERRO DOMO DOS 1.345	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-4	4970000.000	282000.000	P	NUEVO CERRO DOMO DOS 1.345	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-1	4970000.000	282000.000	P	NUEVO CERRO DOMO TRES 1.346	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-2	4970000.000	285000.000	P	NUEVO CERRO DOMO TRES 1.346	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-3	4969000.000	285000.000	P	NUEVO CERRO DOMO TRES 1.346	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-4	4969000.000	282000.000	P	NUEVO CERRO DOMO TRES 1.346	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-1	4969000.000	282000.000	P	NUEVO CERRO DOMO CUATRO 1.347	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-2	4969000.000	285000.000	P	NUEVO CERRO DOMO CUATRO 1.347	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-3	4968000.000	285000.000	P	NUEVO CERRO DOMO CUATRO 1.347	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-4	4968000.000	282000.000	P	NUEVO CERRO DOMO CUATRO 1.347	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-1	4968000.000	282000.000	P	NUEVO CERRO DOMO CINCO 1.348	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-2	4968000.000	285000.000	P	NUEVO CERRO DOMO CINCO 1.348	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-3	4967000.000	285000.000	P	NUEVO CERRO DOMO CINCO 1.348	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-4	4967000.000	282000.000	P	NUEVO CERRO DOMO CINCO 1.348	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-1	4967000.000	282000.000	P	NUEVO CERRO DOMO SEIS 1.349	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-2	4967000.000	285000.000	P	NUEVO CERRO DOMO SEIS 1.349	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-3	4966000.000	285000.000	P	NUEVO CERRO DOMO SEIS 1.349	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-4	4966000.000	282000.000	P	NUEVO CERRO DOMO SEIS 1.349	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-1	4966000.000	282000.000	P	NUEVO CERRO DOMO SIETE 1.350	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-2	4966000.000	285000.000	P	NUEVO CERRO DOMO SIETE 1.350	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-3	4965000.000	285000.000	P	NUEVO CERRO DOMO SIETE 1.350	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-4	4965000.000	282000.000	P	NUEVO CERRO DOMO SIETE 1.350	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-1	4972000.000	285000.000	P	NUEVO CERRO DOMO OCHO 1.351	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-2	4972000.000	288000.000	P	NUEVO CERRO DOMO OCHO 1.351	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-3	4971000.000	288000.000	P	NUEVO CERRO DOMO OCHO 1.351	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-4	4971000.000	285000.000	P	NUEVO CERRO DOMO OCHO 1.351	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-1	4971000.000	285000.000	P	NUEVO CERRO DOMO NUEVE 1.352	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-2	4971000.000	288000.000	P	NUEVO CERRO DOMO NUEVE 1.352	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-3	4970000.000	288000.000	P	NUEVO CERRO DOMO NUEVE 1.352	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-4	4970000.000	285000.000	P	NUEVO CERRO DOMO NUEVE 1.352	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-1	4970000.000	285000.000	P	NUEVO CERRO DOMO DIEZ 1.353	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-2	4970000.000	288000.000	P	NUEVO CERRO DOMO DIEZ 1.353	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-3	4969000.000	288000.000	P	NUEVO CERRO DOMO DIEZ 1.353	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-4	4969000.000	285000.000	P	NUEVO CERRO DOMO DIEZ 1.353	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-1	4969000.000	285000.000	P	NUEVO CERRO DOMO ONCE 1.354	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-2	4969000.000	288000.000	P	NUEVO CERRO DOMO ONCE 1.354	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
VER	**NORT**	**EST**	**TIPO**	**NOMROL**	**BOLSENT**	**FECHSENT**	**JUGA**	**NOMMAN**	**FECHCERT**
V-3	4968000.000	288000.000	P	NUEVO CERRO DOMO ONCE 1.354	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-4	4968000.000	285000.000	P	NUEVO CERRO DOMO ONCE 1.354	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-1	4968000.000	285000.000	P	NUEVO CERRO DOMO DOCE 1.355	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99

V-2	4968000.000	288000.000	P	NUEVO CERRO DOMO DOCE 1.355	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-3	4967000.000	288000.000	P	NUEVO CERRO DOMO DOCE 1.355	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-4	4967000.000	285000.000	P	NUEVO CERRO DOMO DOCE 1.355	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-1	4967000.000	285000.000	P	NUEVO CERRO DOMO TRECE 1.356	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-2	4967000.000	288000.000	P	NUEVO CERRO DOMO TRECE 1.356	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-3	4966000.000	288000.000	P	NUEVO CERRO DOMO TRECE 1.356	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-4	4966000.000	285000.000	P	NUEVO CERRO DOMO TRECE 1.356	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-1	4966000.000	285000.000	P	NUEVO CERRO DOMO CATORCE 1.357	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-2	4966000.000	288000.000	P	NUEVO CERRO DOMO CATORCE 1.357	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-3	4965000.000	288000.000	P	NUEVO CERRO DOMO CATORCE 1.357	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-4	4965000.000	285000.000	P	NUEVO CERRO DOMO CATORCE 1.357	5148	14-04-00	II -COYHAIQUE	S.C.M. EL TOQUI	21-12-99
V-1	4862500.000	287000.000	P	NUEVO LIMITE UNO 1.748	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-2	4862500.000	288000.000	P	NUEVO LIMITE UNO 1.748	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-3	4859500.000	288000.000	P	NUEVO LIMITE UNO 1.748	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-4	4859500.000	287000.000	P	NUEVO LIMITE UNO 1.748	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-1	4864000.000	284500.000	P	NUEVO LIMITE DOS 1.749	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-2	4864000.000	287500.000	P	NUEVO LIMITE DOS 1.749	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-3	4863000.000	287500.000	P	NUEVO LIMITE DOS 1.749	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-4	4863000.000	284500.000	P	NUEVO LIMITE DOS 1.749	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-1	4863000.000	284000.000	P	NUEVO LIMITE TRES 1.750	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-2	4863000.000	287000.000	P	NUEVO LIMITE TRES 1.750	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-3	4862000.000	287000.000	P	NUEVO LIMITE TRES 1.750	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-4	4862000.000	284000.000	P	NUEVO LIMITE TRES 1.750	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-1	4862000.000	284000.000	P	NUEVO LIMITE CUATRO 1.751	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-2	4862000.000	287000.000	P	NUEVO LIMITE CUATRO 1.751	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-3	4861000.000	287000.000	P	NUEVO LIMITE CUATRO 1.751	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-4	4861000.000	284000.000	P	NUEVO LIMITE CUATRO 1.751	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-1	4861000.000	284000.000	P	NUEVO LIMITE CINCO 1.752	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-2	4861000.000	287000.000	P	NUEVO LIMITE CINCO 1.752	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-3	4860000.000	287000.000	P	NUEVO LIMITE CINCO 1.752	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-4	4860000.000	284000.000	P	NUEVO LIMITE CINCO 1.752	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-1	4860000.000	282500.000	P	NUEVO LIMITE SEIS 1.753	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-2	4860000.000	285500.000	P	NUEVO LIMITE SEIS 1.753	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-3	4859000.000	285500.000	P	NUEVO LIMITE SEIS 1.753	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-4	4859000.000	282500.000	P	NUEVO LIMITE SEIS 1.753	5148	08-04-00	-CHILE CHICO	S.C.M. EL TOQUI	27-12-99
V-1	5015000.000	264000.000	P	NUEVO TREINTA Y CUATRO 1.359	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00
V-2	5015000.000	265000.000	P	NUEVO TREINTA Y CUATRO 1.359	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00
V-3	5012000.000	265000.000	P	NUEVO TREINTA Y CUATRO 1.359	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00
V-4	5012000.000	264000.000	P	NUEVO TREINTA Y CUATRO 1.359	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00
V-1	5015000.000	265000.000	P	NUEVO TREINTA Y CINCO 1.360	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00
VER	**NORT**	**EST**	**TIPO**	**NOMROL**	**BOLSENT**	**FECHSENT**	**JUGA**	**NOMMAN**	**FECHCERT**
V-2	5015000.000	266000.000	P	NUEVO TREINTA Y CINCO 1.360	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00
V-3	5012000.000	266000.000	P	NUEVO TREINTA Y CINCO 1.360	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00
V-4	5012000.000	265000.000	P	NUEVO TREINTA Y CINCO 1.360	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00
V-1	5015000.000	266000.000	P	NUEVO TREINTA Y SEIS 1.361	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00
V-2	5015000.000	267000.000	P	NUEVO TREINTA Y SEIS 1.361	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00

V-3	5012000.000	267000.000	P	NUEVO TREINTA Y SEIS 1.361	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00
V-4	5012000.000	266000.000	P	NUEVO TREINTA Y SEIS 1.361	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00
V-1	5015000.000	267000.000	P	NUEVO TREINTA Y SIETE 1.362	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00
V-2	5015000.000	268000.000	P	NUEVO TREINTA Y SIETE 1.362	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00
V-3	5012000.000	268000.000	P	NUEVO TREINTA Y SIETE 1.362	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00
V-4	5012000.000	267000.000	P	NUEVO TREINTA Y SIETE 1.362	5232	22-03-01	II -COYHAIQUE	S.C.M. EL TOQUI	06-11-00

Exploitation Concessions

NOMMAN	DIR	TIPO	COMU	NORT	EST	BOLCERT	FECHCERT
MRA. LAC CHILE S.A.	LAS URBINAS Nº53 P.13	M	P. COYHAIQUE	4975500	729500	4293	26-02-92
MRA. LAC CHILE S.A.	LAS URBINAS Nº53 P.13	M	P. AYSEN	4979500	728500	4293	26-02-92
BOLMENS	**FECHMENS**	**XLAR**	**XANC**	**SIE**	**NOMROL**		
4361	05-11-92	3000	3000	SEC. RIO GUILLERMO	GATO BLANCO 61 1-90 738		
4361	05-11-92	3000	1000	RIO EMPER.GUILLERMO	MAURA 1-30 5.099		
JUGA	**OBSER**	**ANO**					
II =COYHAIQUE	ERA PEDIMENTO "GATO BLANCO 61"	1992					
=PUERTO AYSEN	ERA PEDIMENTO "MAURA"	1992					

Langlois

Disp Name	Disp Type	Clm Area	Prot to	Project Name	Owner	Twp-County
3658811	Mining Claim	16 HA	2/1/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3658812	Mining Claim	16 HA	2/1/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3658821	Mining Claim	16 HA	2/2/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3658822	Mining Claim	16 HA	2/2/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3658841	Mining Claim	16 HA	6/7/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3658842	Mining Claim	16 HA	6/7/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3804543	Mining Claim	16 HA	3/26/03	BP-NOREX	BWR-82.7%	GREVET
3804544	Mining Claim	16 HA	3/26/03	BP-NOREX	BWR-82.7%	GREVET
3804545	Mining Claim	16 HA	3/26/03	BP-NOREX	BWR-82.7%	GREVET
3804571	Mining Claim	16 HA	3/26/03	BP-NOREX	BWR-82.7%	GREVET
3804572	Mining Claim	16 HA	3/26/03	BP-NOREX	BWR-82.7%	GREVET
3804573	Mining Claim	9 HA	3/26/03	BP-NOREX	BWR-82.7%	GREVET
3804703	Mining Claim	16 HA	3/29/03	BP-NOREX	BWR-82.7%	GREVET
3804704	Mining Claim	16 HA	3/29/03	BP-NOREX	BWR-82.7%	GREVET
3804705	Mining Claim	16 HA	3/29/03	BP-NOREX	BWR-82.7%	GREVET
3804714	Mining Claim	16 HA	3/30/03	BP-NOREX	BWR-82.7%	GREVET
3804715	Mining Claim	16 HA	3/30/03	BP-NOREX	BWR-82.7%	GREVET
3804725	Mining Claim	16 HA	3/29/03	BP-NOREX	BWR-82.7%	GREVET
3804734	Mining Claim	16 HA	3/30/03	BP-NOREX	BWR-82.7%	GREVET
3804735	Mining Claim	16 HA	3/30/03	BP-NOREX	BWR-82.7%	GREVET
3822662	Mining Claim	16 HA	4/5/03	BP-NOREX	BWR-82.7%	GREVET
3822663	Mining Claim	16 HA	4/5/03	BP-NOREX	BWR-82.7%	GREVET
3822664	Mining Claim	16 HA	4/5/03	BP-NOREX	BWR-82.7%	GREVET
3822665	Mining Claim	16 HA	4/5/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3822695	Mining Claim	16 HA	4/2/03	BP-NOREX	BWR-82.7%	GREVET
3822781	Mining Claim	16 HA	4/5/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3822811	Mining Claim	16 HA	4/5/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3822812	Mining Claim	16 HA	4/5/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3822813	Mining Claim	16 HA	4/5/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3822814	Mining Claim	16 HA	4/5/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3822821	Mining Claim	16 HA	4/4/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3822822	Mining Claim	16 HA	4/4/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3822823	Mining Claim	16 HA	4/4/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3822831	Mining Claim	10 HA	4/3/03	BP-NOREX	BWR-82.7%	GREVET
3822834	Mining Claim	16 HA	4/3/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3822835	Mining Claim	16 HA	4/3/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3822843	Mining Claim	16 HA	4/6/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3822844	Mining Claim	16 HA	4/6/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3822845	Mining Claim	16 HA	4/6/03	BP-NOREX	BWR-82.7%	MOUNTAIN
3822853	Mining Claim	16 HA	4/7/03	BP-NOREX	BWR-82.7%	MOUNTAIN
TOTAL	40 mining claims	627 HA				

Disp Name	Disp Type	Clm	Prot to	Project Name	Owner	Twp-County

		Area				
5012461	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012462	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012463	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012464	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012465	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012466	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012467	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012468	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012469	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012470	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012471	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012472	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012473	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012474	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012475	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012476	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012477	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012478	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012479	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012480	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012481	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012482	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012483	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012484	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012485	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012486	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012487	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012488	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012489	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012490	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012491	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012492	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012493	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012494	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
5012495	Mining Claim	16HA	2/20/03	CODA	BWR-75%	GREVET
TOTAL	35 mining claims	560HA				
5011955	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5011956	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5011957	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5011958	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5011959	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5011960	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5011961	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5011964	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5015163	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016368	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
Disp Name	**Disp Type**	**Clm**	**Prot to**	**Project Name**	**Owner**	**Twp-County**

		Area				
5016369	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016370	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016371	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016372	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016373	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016375	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016389	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016390	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016391	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016392	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016401	Mining Claim	16HA	4/17/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016404	Mining Claim	16HA	4/17/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016405	Mining Claim	16HA	4/17/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016406	Mining Claim	16HA	4/17/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016407	Mining Claim	16HA	4/17/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016408	Mining Claim	16HA	4/17/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016409	Mining Claim	16HA	4/17/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016410	Mining Claim	16HA	4/17/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016411	Mining Claim	16HA	4/17/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016412	Mining Claim	16HA	4/17/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016413	Mining Claim	16HA	4/17/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016414	Mining Claim	16HA	4/17/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016415	Mining Claim	16HA	4/17/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016416	Mining Claim	16HA	4/17/03	FANCAMP	BWR-18.75%	MOUNTAIN
5016417	Mining Claim	16HA	4/17/03	FANCAMP	BWR-18.75%	MOUNTAIN
5017021	Mining Claim	16HA	4/26/03	FANCAMP	BWR-18.75%	MOUNTAIN
TOTAL	36 mining claims	576HA				
3723543	Mining Claim	16HA	4/28/03	GREVET	BWR-100%	GREVET
3723544	Mining Claim	16HA	4/28/03	GREVET	BWR-100%	GREVET
3723545	Mining Claim	16HA	4/28/03	GREVET	BWR-100%	MOUNTAIN
3723551	Mining Claim	16HA	4/28/03	GREVET	BWR-100%	GREVET
3723552	Mining Claim	16HA	4/28/03	GREVET	BWR-100%	GREVET
3723553	Mining Claim	16HA	4/28/03	GREVET	BWR-100%	MOUNTAIN
3808374	Mining Claim	16HA	5/27/03	GREVET	BWR-100%	MOUNTAIN
3808375	Mining Claim	16HA	5/27/03	GREVET	BWR-100%	MOUNTAIN
3808401	Mining Claim	16HA	5/28/03	GREVET	BWR-100%	MOUNTAIN
3808402	Mining Claim	16HA	5/28/03	GREVET	BWR-100%	MOUNTAIN
3808403	Mining Claim	16HA	5/28/03	GREVET	BWR-100%	MOUNTAIN
3808404	Mining Claim	16HA	5/28/03	GREVET	BWR-100%	MOUNTAIN
3808405	Mining Claim	16HA	5/28/03	GREVET	BWR-100%	MOUNTAIN
3808415	Mining Claim	16HA	5/29/03	GREVET	BWR-100%	MOUNTAIN
3820143	Mining Claim	16HA	5/22/03	GREVET	BWR-100%	MOUNTAIN
3820144	Mining Claim	16HA	5/22/03	GREVET	BWR-100%	MOUNTAIN
3820145	Mining Claim	16HA	5/22/03	GREVET	BWR-100%	MOUNTAIN
3820151	Mining Claim	16HA	5/23/03	GREVET	BWR-100%	MOUNTAIN
3820153	Mining Claim	16HA	5/23/03	GREVET	BWR-100%	MOUNTAIN
Disp Name	Disp Type	Clm	Prot to	Project Name	Owner	Twp-County

		Area				
3820154	Mining Claim	16HA	5/23/03	GREVET	BWR-100%	MOUNTAIN
3820155	Mining Claim	16HA	5/23/03	GREVET	BWR-100%	MOUNTAIN
3820161	Mining Claim	16HA	5/24/03	GREVET	BWR-100%	MOUNTAIN
3820162	Mining Claim	16HA	5/24/03	GREVET	BWR-100%	MOUNTAIN
3820172	Mining Claim	15HA	5/25/03	GREVET	BWR-100%	MOUNTAIN
3820173	Mining Claim	17HA	5/25/03	GREVET	BWR-100%	MOUNTAIN
3820174	Mining Claim	16HA	5/25/03	GREVET	BWR-100%	MOUNTAIN
3820182	Mining Claim	16HA	5/26/03	GREVET	BWR-100%	MOUNTAIN
3820183	Mining Claim	17HA	5/26/03	GREVET	BWR-100%	MOUNTAIN
3820184	Mining Claim	15HA	5/26/03	GREVET	BWR-100%	MOUNTAIN
3820205	Mining Claim	15HA	5/28/03	GREVET	BWR-100%	MOUNTAIN
3820211	Mining Claim	16HA	5/29/03	GREVET	BWR-100%	MOUNTAIN
3820212	Mining Claim	16HA	5/29/03	GREVET	BWR-100%	MOUNTAIN
3820252	Mining Claim	16HA	5/23/03	GREVET	BWR-100%	MOUNTAIN
3820253	Mining Claim	16HA	5/23/03	GREVET	BWR-100%	MOUNTAIN
3820254	Mining Claim	16HA	5/23/03	GREVET	BWR-100%	MOUNTAIN
3820255	Mining Claim	16HA	5/23/03	GREVET	BWR-100%	MOUNTAIN
3820261	Mining Claim	16HA	5/24/03	GREVET	BWR-100%	MOUNTAIN
3820264	Mining Claim	16HA	5/24/03	GREVET	BWR-100%	MOUNTAIN
3820265	Mining Claim	16HA	5/24/03	GREVET	BWR-100%	MOUNTAIN
3820271	Mining Claim	16HA	5/25/03	GREVET	BWR-100%	MOUNTAIN
3820272	Mining Claim	16HA	5/25/03	GREVET	BWR-100%	MOUNTAIN
3820273	Mining Claim	16HA	5/25/03	GREVET	BWR-100%	MOUNTAIN
3820301	Mining Claim	16HA	5/28/03	GREVET	BWR-100%	MOUNTAIN
3820302	Mining Claim	16HA	5/28/03	GREVET	BWR-100%	MOUNTAIN
3820303	Mining Claim	16HA	5/28/03	GREVET	BWR-100%	MOUNTAIN
3820304	Mining Claim	16HA	5/28/03	GREVET	BWR-100%	MOUNTAIN
3820305	Mining Claim	16HA	5/28/03	GREVET	BWR-100%	MOUNTAIN
3820313	Mining Claim	16HA	5/29/03	GREVET	BWR-100%	MOUNTAIN
3820314	Mining Claim	16HA	5/29/03	GREVET	BWR-100%	MOUNTAIN
3820315	Mining Claim	16HA	5/29/03	GREVET	BWR-100%	MOUNTAIN
3820321	Mining Claim	16HA	5/30/03	GREVET	BWR-100%	MOUNTAIN
3820322	Mining Claim	16HA	5/30/03	GREVET	BWR-100%	MOUNTAIN
3820335	Mining Claim	16HA	5/31/03	GREVET	BWR-100%	MOUNTAIN
3828132	Mining Claim	16HA	5/22/03	GREVET	BWR-100%	GREVET
3828133	Mining Claim	16HA	5/22/03	GREVET	BWR-100%	GREVET
3828134	Mining Claim	16HA	5/22/03	GREVET	BWR-100%	GREVET
3828135	Mining Claim	16HA	5/22/03	GREVET	BWR-100%	GREVET
3828171	Mining Claim	16HA	5/21/03	GREVET	BWR-100%	GREVET
3828184	Mining Claim	16HA	5/20/03	GREVET	BWR-100%	GREVET
3828185	Mining Claim	16HA	5/20/03	GREVET	BWR-100%	GREVET
3828344	Mining Claim	16HA	5/20/03	GREVET	BWR-100%	GREVET
3828345	Mining Claim	16HA	5/20/03	GREVET	BWR-100%	GREVET
3828351	Mining Claim	16HA	5/21/03	GREVET	BWR-100%	GREVET
3828352	Mining Claim	16HA	5/21/03	GREVET	BWR-100%	GREVET
3923651	Mining Claim	4.48HA	5/5/03	GREVET	BWR-100%	GREVET
Disp Name	Disp Type	Clm Area	Prot to	Project Name	Owner	Twp-County

3923652	Mining Claim	4.09HA	5/5/03	GREVET	BWR-100%	GREVET
3949531	Mining Claim	9.68HA	5/5/03	GREVET	BWR-100%	GREVET
3949532	Mining Claim	16HA	2/3/03	GREVET	BWR-100%	GREVET
3990521	Mining Claim	16HA	4/1/03	GREVET	BWR-100%	MOUNTAIN
3990522	Mining Claim	16HA	4/1/03	GREVET	BWR-100%	MOUNTAIN
3990523	Mining Claim	16HA	4/1/03	GREVET	BWR-100%	MOUNTAIN
3990524	Mining Claim	16HA	4/1/03	GREVET	BWR-100%	MOUNTAIN
3990525	Mining Claim	16HA	4/1/03	GREVET	BWR-100%	MOUNTAIN
3990535	Mining Claim	16HA	4/1/03	GREVET	BWR-100%	MOUNTAIN
3997341	Mining Claim	0.19HA	5/5/03	GREVET	BWR-100%	GREVET
3997351	Mining Claim	16.06HA	5/5/03	GREVET	BWR-100%	GREVET
3997352	Mining Claim	16HA	2/3/03	GREVET	BWR-100%	GREVET
3997361	Mining Claim	0.51HA	5/5/03	GREVET	BWR-100%	GREVET
3997362	Mining Claim	15.85HA	5/5/03	GREVET	BWR-100%	GREVET
4230874	Mining Claim	16HA	5/17/03	GREVET	BWR-100%	GREVET
4484461	Mining Claim	16HA	5/10/03	GREVET	BWR-100%	GREVET
4484462	Mining Claim	16HA	5/10/03	GREVET	BWR-100%	GREVET
4484463	Mining Claim	16HA	5/10/03	GREVET	BWR-100%	GREVET
4484464	Mining Claim	16HA	5/10/03	GREVET	BWR-100%	GREVET
4484465	Mining Claim	16HA	5/10/03	GREVET	BWR-100%	GREVET
4484671	Mining Claim	16HA	5/9/03	GREVET	BWR-100%	GREVET
4484672	Mining Claim	16HA	5/9/03	GREVET	BWR-100%	GREVET
4484673	Mining Claim	16HA	5/9/03	GREVET	BWR-100%	GREVET
4484674	Mining Claim	16HA	5/9/03	GREVET	BWR-100%	GREVET
4484675	Mining Claim	16HA	5/9/03	GREVET	BWR-100%	GREVET
4527871	Mining Claim	16HA	5/9/03	GREVET	BWR-100%	GREVET
4527872	Mining Claim	16HA	5/9/03	GREVET	BWR-100%	GREVET
4527873	Mining Claim	16HA	5/9/03	GREVET	BWR-100%	GREVET
4550401	Mining Claim	16HA	12/9/02	GREVET	BWR-100%	GREVET
4550402	Mining Claim	16HA	12/9/02	GREVET	BWR-100%	GREVET
4550403	Mining Claim	16HA	12/9/02	GREVET	BWR-100%	GREVET
4550404	Mining Claim	16HA	12/9/02	GREVET	BWR-100%	GREVET
4550411	Mining Claim	16HA	12/8/02	GREVET	BWR-100%	GREVET
4550412	Mining Claim	16HA	12/8/02	GREVET	BWR-100%	GREVET
4550413	Mining Claim	16HA	12/8/02	GREVET	BWR-100%	GREVET
4550414	Mining Claim	16HA	12/8/02	GREVET	BWR-100%	GREVET
4703911	Mining Claim	16HA	4/30/03	GREVET	BWR-100%	GREVET
4703912	Mining Claim	16HA	4/30/03	GREVET	BWR-100%	GREVET
4703913	Mining Claim	16HA	4/30/03	GREVET	BWR-100%	GREVET
4703914	Mining Claim	16HA	4/30/03	GREVET	BWR-100%	GREVET
4703915	Mining Claim	16HA	4/30/03	GREVET	BWR-100%	GREVET
4703921	Mining Claim	16HA	5/1/03	GREVET	BWR-100%	GREVET
4703922	Mining Claim	7.69HA	7/31/03	GREVET	BWR-100%	GREVET
4703923	Mining Claim	3.87HA	7/31/03	GREVET	BWR-100%	GREVET
4703924	Mining Claim	1.01HA	7/31/03	GREVET	BWR-100%	GREVET
4703925	Mining Claim	6.78HA	7/31/03	GREVET	BWR-100%	GREVET
Disp Name	**Disp Type**	**Clm Area**	**Prot to**	**Project Name**	**Owner**	**Twp-County**
4703931	Mining Claim	16HA	5/2/03	GREVET	BWR-100%	GREVET

4703932	Mining Claim	15.29HA	8/1/03	GREVET	BWR-100%	GREVET
4703933	Mining Claim	16HA	5/2/03	GREVET	BWR-100%	GREVET
4703934	Mining Claim	16HA	5/2/03	GREVET	BWR-100%	GREVET
4703935	Mining Claim	16HA	5/2/03	GREVET	BWR-100%	GREVET
4703942	Mining Claim	16HA	5/3/03	GREVET	BWR-100%	GREVET
4703943	Mining Claim	16HA	5/3/03	GREVET	BWR-100%	MOUNTAIN
4703944	Mining Claim	10.32HA	8/1/03	GREVET	BWR-100%	MOUNTAIN
4703951	Mining Claim	16HA	4/30/03	GREVET	BWR-100%	GREVET
4703952	Mining Claim	16HA	7/30/03	GREVET	BWR-100%	GREVET
4703953	Mining Claim	7.44HA	7/30/03	GREVET	BWR-100%	GREVET
4703954	Mining Claim	16HA	4/30/03	GREVET	BWR-100%	GREVET
4703961	Mining Claim	16HA	5/2/03	GREVET	BWR-100%	GREVET
4703962	Mining Claim	8.88HA	8/1/03	GREVET	BWR-100%	GREVET
4703963	Mining Claim	16HA	5/2/03	GREVET	BWR-100%	GREVET
4703964	Mining Claim	11HA	8/1/03	GREVET	BWR-100%	GREVET
4703965	Mining Claim	7HA	8/1/03	GREVET	BWR-100%	GREVET
4703971	Mining Claim	16HA	5/3/03	GREVET	BWR-100%	GREVET
4703972	Mining Claim	16HA	5/3/03	GREVET	BWR-100%	MOUNTAIN
4703973	Mining Claim	16HA	5/3/03	GREVET	BWR-100%	MOUNTAIN
4703974	Mining Claim	16HA	5/3/03	GREVET	BWR-100%	MOUNTAIN
4703975	Mining Claim	16HA	5/3/03	GREVET	BWR-100%	MOUNTAIN
4704001	Mining Claim	10HA	8/2/03	GREVET	BWR-100%	MOUNTAIN
4704002	Mining Claim	16HA	5/4/03	GREVET	BWR-100%	MOUNTAIN
4704003	Mining Claim	16HA	5/4/03	GREVET	BWR-100%	MOUNTAIN
4704004	Mining Claim	16HA	5/4/03	GREVET	BWR-100%	MOUNTAIN
4704005	Mining Claim	16HA	5/4/03	GREVET	BWR-100%	MOUNTAIN
4704011	Mining Claim	16HA	5/5/03	GREVET	BWR-100%	MOUNTAIN
4704012	Mining Claim	16HA	5/5/03	GREVET	BWR-100%	MOUNTAIN
4704024	Mining Claim	16HA	5/4/03	GREVET	BWR-100%	MOUNTAIN
4704025	Mining Claim	16HA	5/4/03	GREVET	BWR-100%	MOUNTAIN
4704031	Mining Claim	16HA	5/4/03	GREVET	BWR-100%	MOUNTAIN
4704032	Mining Claim	16HA	5/4/03	GREVET	BWR-100%	MOUNTAIN
4704033	Mining Claim	16HA	5/4/03	GREVET	BWR-100%	MOUNTAIN
4704035	Mining Claim	16HA	5/4/03	GREVET	BWR-100%	MOUNTAIN
4704951	Mining Claim	16HA	5/25/03	GREVET	BWR-100%	MOUNTAIN
4704952	Mining Claim	16HA	5/25/03	GREVET	BWR-100%	MOUNTAIN
4704953	Mining Claim	16HA	5/25/03	GREVET	BWR-100%	MOUNTAIN
4704954	Mining Claim	16HA	5/25/03	GREVET	BWR-100%	MOUNTAIN
4704955	Mining Claim	16HA	5/25/03	GREVET	BWR-100%	MOUNTAIN
4704961	Mining Claim	16HA	5/26/03	GREVET	BWR-100%	MOUNTAIN
5013867	Mining Claim	16HA	3/27/03	GREVET	BWR-100%	MOUNTAIN
5014129	Mining Claim	16HA	3/27/03	GREVET	BWR-100%	MOUNTAIN
5236611	Mining Claim	16HA	5/9/03	GREVET	BWR-100%	GREVET
5236612	Mining Claim	16HA	5/9/03	GREVET	BWR-100%	GREVET
BM831	Mining Lease	132.9HA	10/18/02	GREVET	BWR-100%	GREVET-MOUNTAIN
C.F.813008	RAILWAY LEASE	0HA	7/31/02	GREVET	BWR-100%	GREVET
P.R.90321	Surface Lease	295HA	1/31/02	GREVET	BWR-100%	GREVET
TOTAL	156 mining claims	2347.14HA				

1 mining lease	132.9HA	
2 surface leases	295HA	

Disp Name	Disp Type	Clm Area	Prot to	Project Name	Owner	Twp-County
5013869	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013870	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013871	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013872	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013873	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013874	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013875	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013876	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013877	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013878	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013879	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013880	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013881	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013882	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013883	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013884	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013889	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013890	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013891	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013892	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013893	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013894	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013895	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013896	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013897	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013901	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013902	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013903	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013904	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013905	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013906	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013907	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013908	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013909	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013930	Mining Claim	16HA	4/26/03	LAROCHE	BWR-75%	MOUNTAIN
5013931	Mining Claim	16HA	4/26/03	LAROCHE	BWR-75%	MOUNTAIN
5013932	Mining Claim	16HA	4/26/03	LAROCHE	BWR-75%	MOUNTAIN
5013933	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013934	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013935	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013936	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
Disp Name	Disp Type	Clm Area	Prot to	Project Name	Owner	Twp-County
5013939	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013940	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN

5013941	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013942	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013943	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013944	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013947	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013948	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013954	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	MOUNTAIN
5013955	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013956	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013962	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013963	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013964	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013965	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013966	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013970	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013971	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013972	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013973	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013974	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013975	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013977	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
5013978	Mining Claim	16HA	3/19/03	LAROCHE	BWR-75%	GREVET
TOTAL	64 mining claims	1040HA				
4067453	Mining Claim	16HA	11/4/02	LIGNERIS	BWR-70%	GREVET
4160873	Mining Claim	16HA	5/20/03	LIGNERIS	BWR-70%	GREVET
4160874	Mining Claim	16HA	5/20/03	LIGNERIS	BWR-70%	GREVET
4160875	Mining Claim	16HA	5/20/03	LIGNERIS	BWR-70%	GREVET
4160881	Mining Claim	16HA	5/20/03	LIGNERIS	BWR-70%	GREVET
4438263	Mining Claim	16HA	1/14/03	LIGNERIS	BWR-70%	GREVET
4438275	Mining Claim	16HA	1/15/03	LIGNERIS	BWR-70%	GREVET
5059973	Mining Claim	16HA	12/18/01	LIGNERIS	BWR-70%	GREVET
5059974	Mining Claim	16HA	12/18/01	LIGNERIS	BWR-70%	GREVET
5059975	Mining Claim	16HA	12/18/01	LIGNERIS	BWR-70%	GREVET
5059976	Mining Claim	16HA	12/18/01	LIGNERIS	BWR-70%	GREVET
5059977	Mining Claim	16HA	12/18/01	LIGNERIS	BWR-70%	GREVET
5059978	Mining Claim	16HA	12/18/01	LIGNERIS	BWR-70%	GREVET
5059979	Mining Claim	16HA	12/18/01	LIGNERIS	BWR-70%	GREVET
TOTAL	13 mining claims	224HA				

Disp Name	Disp Type	Clm Area	Prot to	Project Name	Owner	Twp-County
4552202	Mining Claim	16HA	2/10/03	ORPHEE	BWR-50%	MOUNTAIN
4552203	Mining Claim	16HA	2/10/03	ORPHEE	BWR-50%	MOUNTAIN
4552204	Mining Claim	16HA	2/10/03	ORPHEE	BWR-50%	MOUNTAIN

4552215	Mining Claim	16HA	2/11/03	ORPHEE	BWR-50%	MOUNTAIN
4552221	Mining Claim	16HA	2/12/03	ORPHEE	BWR-50%	MOUNTAIN
4552222	Mining Claim	16HA	2/12/03	ORPHEE	BWR-50%	MOUNTAIN
4552225	Mining Claim	16HA	2/12/03	ORPHEE	BWR-50%	MOUNTAIN
4552232	Mining Claim	16HA	2/13/03	ORPHEE	BWR-50%	MOUNTAIN
4552233	Mining Claim	16HA	2/13/03	ORPHEE	BWR-50%	MOUNTAIN
4552234	Mining Claim	16HA	2/13/03	ORPHEE	BWR-50%	MOUNTAIN
4552235	Mining Claim	16HA	2/13/03	ORPHEE	BWR-50%	MOUNTAIN
4552492	Mining Claim	16HA	2/10/03	ORPHEE	BWR-50%	MOUNTAIN
4552493	Mining Claim	16HA	2/10/03	ORPHEE	BWR-50%	MOUNTAIN
4552494	Mining Claim	16HA	2/10/03	ORPHEE	BWR-50%	MOUNTAIN
4552562	Mining Claim	16HA	2/10/03	ORPHEE	BWR-50%	MOUNTAIN
4552563	Mining Claim	16HA	2/10/03	ORPHEE	BWR-50%	MOUNTAIN
4552564	Mining Claim	16HA	2/10/03	ORPHEE	BWR-50%	MOUNTAIN
4552572	Mining Claim	16HA	2/11/03	ORPHEE	BWR-50%	MOUNTAIN
4552921	Mining Claim	16HA	2/15/03	ORPHEE	BWR-50%	MOUNTAIN
4552922	Mining Claim	16HA	2/15/03	ORPHEE	BWR-50%	MOUNTAIN
4552923	Mining Claim	16HA	2/15/03	ORPHEE	BWR-50%	MOUNTAIN
4552932	Mining Claim	16HA	2/13/03	ORPHEE	BWR-50%	MOUNTAIN
4552933	Mining Claim	16HA	2/13/03	ORPHEE	BWR-50%	MOUNTAIN
4552934	Mining Claim	16HA	2/13/03	ORPHEE	BWR-50%	MOUNTAIN
4552944	Mining Claim	16HA	2/12/03	ORPHEE	BWR-50%	MOUNTAIN
4552945	Mining Claim	16HA	2/12/03	ORPHEE	BWR-50%	MOUNTAIN
4552951	Mining Claim	16HA	2/11/03	ORPHEE	BWR-50%	MOUNTAIN
4552954	Mining Claim	16HA	2/11/03	ORPHEE	BWR-50%	MOUNTAIN
4552955	Mining Claim	16HA	2/11/03	ORPHEE	BWR-50%	MOUNTAIN
TOTAL	29 mining claims	464HA				

Caribou

PROPERTY	AREA (ha)	TOTAL CLAIMS	CROWN GRANT	MINING LEASE	SURFACE LEASE	OWNERSHIP
Caribou Mine	3106			ML 246		100% CanZinco
Crown Grant	509		142686			
Woodside Brook (Tailings)	528	33			SIML 2271	100% CanZinco
Restigouche Deposit	816	43		ML 255	SIML 2473	100% CanZinco

The Caribou underground and open pit mine is located in Restigouche County, New Brunswick. Mineral tenure for the underground mine is held through surveyed Mining Lease 246. This covers an area of 3,106 hectares. Within this Lease, and centred over the Caribou deposit is a Land Grant of 509 hectares where East West Caribou Mining Limited controls all surface rights. On the southern boundary of the Mining Lease, the 528 hectare Woodside Brook property is the site of the current tailings pond area. This property consists of Industrial Surface Mining Lease # 2271 totaling 246 hectares and 33 mineral claims totaling 282 hectares.

Mineral tenure for the open pit mine, which is located 30km west of the underground mine is held through surveyed Mining Lease 255 totaling 128 hectares as well as Industrial Surface Mining Lease # 2473 (same area) and 43 mineral claims totaling 688 hectares.

APPENDIX 2

CERTIFICATE AND CONSENT LETTERS

CERTIFICATE AND CONSENT

To Accompany the Independent Technical Report on the Mining Assets of Breakwater Resources Ltd.

I, Neal Rigby residing at 2728 S Newcombe St, Lakewood, Colorado, USA do hereby certify that:

1. I am a Corporate Mining Consultant with the firm of Steffen Robertson and Kirsten (US) Inc. (SRK) with an office at Suite 3000, 7175 W Jefferson Ave, Lakewood, Colorado, USA;
2. I am a graduate of the University of Wales, UK with a BSc. in Mining Engineering (1974), a PhD. in Mining Engineering from the University of Wales (1977) and have practiced my profession continuously since 1974;
3. I am a Chartered Engineer, a Member of the Institute of Mining and Metallurgy and a Member of the American Institute of Mining Engineers;
4. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the securities of Breakwater Resources Ltd.
5. I am not aware of any material fact or material change with respect to the subject matter of the technical report, which is not reflected in the technical report, the omission to disclose which makes the technical report misleading.
6. I, as the qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
7. I have not had any prior involvement with the property that is subject to the technical report.
8. I have read National Instrument 43-101 and Form 43-101F1 and the technical report has been prepared in compliance with this Instrument and Form 43-101F1.
9. Steffen Robertson and Kirsten (Canada) Inc. supported by Steffen Robertson and Kirsten (US) Inc. was retained by Breakwater Resources Ltd. to prepare an Independent Technical Report on Breakwater in accordance with National Instrument 43-101 and to assist in the preparation of the technical components of the prospectus. The following report is based on our review of project files, operations data, discussions with Breakwater personnel, and discussions with SRK employees who visited the mine sites in 2001.
10. I was a co-author of the report.
11. I hereby consent to use of this report and our name in the preparation of a prospectus for submission to any Provincial regulatory authority.

"SEAL"
"Neal Rigby"

Denver, Colorado, USA
January 11, 2002

Neal Rigby, C.Eng.
Group Chairman, Corporate Mining Consultant

CERTIFICATE AND CONSENT

To Accompany the Independent Technical Report on the Mining Assets of Breakwater Resources Ltd.

I, Christopher H. Page, residing at 102 Deep Dene Road, Vancouver, BC do hereby certify that:

1. I am a Corporate Consultant with the firm of Steffen Robertson and Kirsten (Canada) Inc. (SRK) with an office at Suite 800, 580 Hornby Street, Vancouver, Canada.
2. I am a graduate of the University of Leeds with a BSc. in Mining Engineering in 1967, Ph.D from Leeds University in 1971, and have practiced my profession continuously since 1971;
3. I am a member of the CIM; and a Professional Engineer registered with the Association of Professional Engineers and Geoscientists of the province of British Columbia, North West Territories and Ontario;
4. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the securities of Breakwater Resources Ltd.
5. I am not aware of any material fact or material change with respect to the subject matter of the technical report, which is not reflected in the technical report, the omission to disclose which makes the technical report misleading.
6. I, as the qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
7. I have not had any prior involvement with the property that is subject to the technical report.
8. I have read National Instrument 43-101 and Form 43-101F1 and the technical report has been prepared in compliance with this Instrument and Form 43-101F1.
9. Steffen Robertson and Kirsten (Canada) Inc. was retained by Breakwater Resources Ltd. to prepare an Independent Technical Report on Breakwater in accordance with National Instrument 43-101 and to assist in the preparation of the technical components of a prospectus. The following report is based on our review of project files, operations data, discussions with Breakwater personnel, and observations made during mine site visits in 2001. Chris Page visited the El Toqui Mine for three days from December 15 to 17, 2001. He visited the El Mochito Mine for four days from December 18 to December 21, 2001. Chris Page visited the Nanisivik Mine several times over the past three years, most recently for 2 days in March, 2001. He has also worked on all of the sites at one time or another.
10. I was a co-author of the report.
11. I hereby consent to use of this report and our name in the preparation of a prospectus for submission to any Provincial regulatory authority.

"SEAL"
"Christopher H. Page"

Vancouver, BC, Canada
January 11, 2002

Christopher H. Page, P.Eng.
Corporate Consultant

CERTIFICATE AND CONSENT

To Accompany the Independent Technical Report on the Mining Assets of Breakwater Resources Ltd.

I, Michael J. Michaud, residing at 43 Eastlawn Street, Oshawa, Ontario do hereby certify that:

1. I am a Senior Geologist with the firm of Steffen Robertson and Kirsten (Canada) Inc. (SRK) with an office at Suite 602, 357 Bay Street, Toronto, Canada.

2. I am a graduate of the University of Waterloo with a HBSc. in Earth Science, MSc. from Lakehead University in 1998, and have practiced my profession continuously since 1987;

3. I am a fellow with the Geological Association of Canada; is a Professional Geoscientist registered with the Association of Professional Engineers and Geoscientists of the province of British Columbia;

4. I have not received, nor do I expect to receive, any interest, directly or indirectly, in any of Breakwater Resources Ltd. mines or securities of Breakwater Resources Ltd.

5. I am not aware of any material fact or material change with respect to the subject matter of the technical report, which is not reflected in the technical report, the omission to disclose which makes the technical report misleading.

6. I, as the qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.

7. I have not had any prior involvement with the property that is subject to the technical report.

8. I have read National Instrument 43-101 and Form 43-101F1 and the technical report has been prepared in compliance with this Instrument and Form 43-101F1.

9. Steffen Robertson and Kirsten (Canada) Inc. was retained by Breakwater Resources Ltd. to prepare a report on their mining assets in accordance with National Instrument 43-101 and to assist in the preparation of the technical components of a prospectus. The following report is based on our review of project files, discussions with Breakwater Resources Ltd personnel, and discussions with SRK employees who visited the mine sites in 2001. Michael Michaud visited the Nanisivik Mine from June 24-25, 1999, Langlois Mine from January 16-19, 2001, Bouchard-Hebert Mine from May 6-10, 2001, and the Caribou Mine from December 16-18, 2001.

10. I was the co-author of the report.

11. I hereby consent to use of this report and our name in the preparation of a prospectus for submission to any Provincial regulatory authority.

"SEAL"
"Michael J. Michaud"

Toronto, Ontario, Canada
January 11, 2002

Michael J. Michaud, P.Geo.
Senior Geologist

CERTIFICATE AND CONSENT

To Accompany the Independent Technical Report on the Mining Assets of Breakwater Resources Ltd.

I, Allan McCracken, residing at 36 Duffryn Crescent, Peterston-Super-Ely, Vale of Glamorgan, Wales, UK do hereby certify that:

1. I am a Principal Geologist with the firm of Steffen Robertson and Kirsten (UK) Ltd. (SRK) with an office at 1-3 Windsor Place Cardiff, CF10 3BX, Wales, UK.

2. I am a graduate of the University of Strathclyde, UK., with a BSc.(Honours) in Applied Geology 1974, and have practiced my profession continuously since 1974;

3. I am a UK Chartered Engineer, member of the Institution of Mining and Metallurgy; and a member of the Institution of Civil Engineers all registered in the UK.

4. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the securities of Breakwater Resources Ltd.

5. I am not aware of any material fact or material change with respect to the subject matter of the technical report, which is not reflected in the technical report, the omission to disclose which makes the technical report misleading.

6. I, as the qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.

7. I have not had any prior involvement with the property that is subject to the technical report.

8. I have read National Instrument 43-101 and Form 43-101F1 and the technical report has been prepared in compliance with this Instrument and Form 43-101F1.

9. Steffen Robertson and Kirsten (Canada) Inc. supported by Steffen Robertson and Kirsten (UK) Inc. was retained by Breakwater Resources Ltd. to prepare an Independent Technical Report on Breakwater in accordance with National Instrument 43-101 and to assist in the preparation of the technical components of a prospectus. The following report is based on our review of project files, operations data, discussions with Breakwater personnel, and observations made during mine site visits in 2001. Allan McCracken visited the Bougrine Mine for three days between December 10 and December 12, 2001.

10. I was a co-author of the report.

11. I hereby consent to use of this report and our name in the preparation of a prospectus for submission to any Provincial regulatory authority.

"SEAL"
"Allan McCracken"

Cardiff, UK
January 11, 2002

Allan McCracken, C.Eng.
Principal Geologist

CERTIFICATE and CONSENT

To Accompany the Independent Technical Report of the Mining Assets of Breakwater Resources Ltd.

I, Louis M. Bernard, residing at 331 Maplehurst Ave., Oakville, Ontario do hereby certify that:

1. I am an Associate Metallurgical Engineer with the firm of Steffen Robertson and Kirsten (Canada) Inc. (SRK) whose office is at Suite 602, 357 Bay Street, Toronto, Canada.

2. I am a graduate of DalTech University (previously Nova Scotia Tech) with a B.Sc in Mining Engineering in 1962 and a Masters of Engineering (M.Eng) degree in Extractive Metallurgy from DalTech in 1963, and have practiced my profession continuously since 1963;

3. I am a member of the Canadian Institute of Mining and Metallurgy (CIM), a member of the Society of Mining, Metallurgy and Exploration (SME); and am a Professional Engineer registered with the Association of Professional Engineers of the province of Ontario (PEO);

4. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the securities of Breakwater Resources Ltd.

5. I am not aware of any material fact or material change with respect to the subject matter of the technical report, which is not reflected in the technical report, the omission to disclose which makes the technical report misleading.

6. I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.

7. I have not had any prior involvement with the property that is subject to the technical report.

8. I have read National Instrument 43-101 and Form 43-101F1 and the technical report has been prepared in compliance with this Instrument and Form 43-101F1.

9. Steffen Robertson and Kirsten (Canada) Inc. was retained by Breakwater Resources Ltd. to prepare an Independent Technical Report on Breakwater in accordance with National Instrument 43-101 and to assist in the preparation of the technical components of the prospectus. The following report is based on our review of project files, operations data, discussions with Breakwater personnel, and discussions with SRK employees who visited the mine sites in 2001.

10. I, Louis M. Bernard, personally visited two of the Breakwater operations ; Bougrine in Tunisia for 90 days in February-May, 2001 and Caribou in Canada for 90 days in 1998.

11. I was a co-author of the report.

12. I hereby consent to use of this report and our name in the preparation of a prospectus for submission to any Provincial regulatory authority.

"SEAL"
"Louis M. Bernard"

Oakville, Ontario, Canada
January 11, 2002

Louis M. Bernard, P.Eng.
Associate Metallurgical Engineer

CERTIFICATE and CONSENT

To Accompany the Independent Technical Report on the Mining Assets of Breakwater Resources Ltd.

I, Andrew S. Bradfield, residing at 5 Patrick Drive, Erin, Ontario do hereby certify that:

1. I am a Principal Mining Engineer with the firm of Steffen Robertson and Kirsten (Canada) Inc. (SRK) with an office at Suite 602, 357 Bay Street, Toronto, Canada.

2. I am a graduate of Queen's University with a B.Sc in Mining Engineering in 1982, and have practiced my profession continuously since 1982;

3. I am a member of the Canadian Institute of Mining and Metallurgy (CIM), and am a Professional Engineer registered with the Association of Professional Engineers of the province of Ontario (PEO);

4. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the securities of Breakwater Resources Ltd.

5. I am not aware of any material fact or material change with respect to the subject matter of the technical report, which is not reflected in the technical report, the omission to disclose which makes the technical report misleading.

6. I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.

7. I have not had any prior involvement with the property that is subject to the technical report.

8. I have read National Instrument 43-101 and Form 43-101F1 and the technical report has been prepared in compliance with this Instrument and Form 43-101F1.

9. Steffen Robertson and Kirsten (Canada) Inc. was retained by Breakwater Resources Ltd. to prepare an Independent Technical Report on Breakwater in accordance with National Instrument 43-101 and to assist in the preparation of the technical components of a prospectus. The following report is based on our review of project files, operations data, discussions with Breakwater personnel, and discussions with SRK employees who visited the mine sites in 2001.

10. I was a co-author of the report.

11. I hereby consent to use of this report and our name in the preparation of a prospectus for submission to any Provincial regulatory authority.

"SEAL"
"Andrew S. Bradfield"

Toronto, Ontario, Canada
January 11, 2002

Andrew S. Bradfield, P.Eng.
Principal Mining Engineer

CERTIFICATE AND CONSENT

To Accompany the Independent Technical Report on the Mining Assets of Breakwater Resources Ltd.

I, Nicholas Michael, residing at 6200 So. Vivian St., Littleton Colorado 80127 USA do hereby certify that:

1. I am a Senior Mining Engineer with the firm of Steffen Robertson and Kirsten (US) Inc. (SRK) with an office at 7175 West Jefferson Avenue, Suite 3000, Lakewood, Colorado 80235 USA.
2. I am a graduate of the Colorado School of Mines with a BSc. in Mining Engineering, MBA. from Willamette University, Geo. H Atkinson Graduate School of Management in 1986, and have practiced my profession continuously since 1987;
3. I am a member of the Society of Mining Engineers (SME);
4. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the securities of Breakwater Resources Ltd.
5. I am not aware of any material fact or material change with respect to the subject matter of the technical report, which is not reflected in the technical report, the omission to disclose which makes the technical report misleading.
6. I, as the qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
7. I have not had any prior involvement with any property that is subject to the technical report.
8. I have read National Instrument 43-101 and Form 43-101F1 and the technical report has been prepared in compliance with this Instrument and Form 43-101F1.
9. Steffen Robertson and Kirsten (Canada) Inc. supported by Steffen Robertson and Kirsten (US) Inc. was retained by Breakwater Resources Ltd. to prepare an Independent Technical Report on Breakwater in accordance with National Instrument 43-101 and to assist in the preparation of the technical components of a prospectus. The following report is based on our review of project files, operations data, discussions with Breakwater personnel, and discussions with SRK employees who visited the mine sites in 2001.
10. I was a co-author of the report.
11. I hereby consent to use of this report and our name in the preparation of a prospectus for submission to any Provincial regulatory authority.

"SEAL"
"Nicholas Michael"

Denver, Colorado, USA
January 11, 2002

Nicholas Michael, MBA
Senior Mining Engineer

CERTIFICATE and CONSENT

To Accompany the Independent Technical Report on the Mining Assets of Breakwater Resources Ltd.

I, Ken S. Reipas, residing at 43 Deverell Street, Whitby, Ontario do hereby certify that:

1. I am a Principal Mining Engineer with the firm of Steffen Robertson and Kirsten (Canada) Inc. (SRK) with an office at Suite 602, 357 Bay Street, Toronto, Canada.

2. I am a graduate of Queen's University with a B.Sc in Mining Engineering in 1981, and have practiced my profession continuously since 1981;

3. I am a Professional Engineer registered with the Association of Professional Engineers of the province of Ontario (PEO);

4. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the securities of Breakwater Resources Ltd.

5. I am not aware of any material fact or material change with respect to the subject matter of the technical report, which is not reflected in the technical report, the omission to disclose which makes the technical report misleading.

6. I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.

7. I have not had any prior involvement with the property that is subject to the technical report.

8. I have read National Instrument 43-101 and Form 43-101F1 and the technical report has been prepared in compliance with this Instrument and Form 43-101F1.

9. Steffen Robertson and Kirsten (Canada) Inc. was retained by Breakwater Resources Ltd. to prepare an Independent Technical Report on Breakwater in accordance with National Instrument 43-101 and to assist in the preparation of the technical components of a prospectus. The following report is based on our review of project files, operations data, discussions with Breakwater personnel, and discussions with SRK employees who visited the mine sites in 2001.

10. I was a co-author of the report.

11. I hereby consent to use of this report and our name in the preparation of a prospectus for submission to any Provincial regulatory authority.

"SEAL"
"Ken S. Reipas"

Toronto, Ontario, Canada
January 11, 2002

Ken S. Reipas, P.Eng.
Principal Mining Engineer

CERTIFICATE AND CONSENT

To Accompany the Independent Technical Report on the Mining Assets of Breakwater Resources Ltd.

I, Diana D. Sollner, residing at #306-308 West 2nd Street, Vancouver, BC do hereby certify that:

1. I am an Environmental Engineer with the firm of Steffen Robertson and Kirsten (Canada) Inc. (SRK) with an office at Suite 800, 580 Hornby Street, Vancouver, Canada.
2. I am a graduate of the University of British Columbia with a B.A.Sc. in Mining and Mineral Process Engineering in 1993, and have practiced my profession continuously since 1993;
3. I am a Professional Engineer registered with the Association of Professional Engineers and Geoscientists of the province of British Columbia;
4. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the securities of Breakwater Resources Ltd.
5. I am not aware of any material fact or material change with respect to the subject matter of the technical report, which is not reflected in the technical report, the omission to disclose which makes the technical report misleading.
6. I, as the qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
7. I have not had any prior involvement with the property that is subject to the technical report.
8. I have read National Instrument 43-101 and Form 43-101F1 and the technical report has been prepared in compliance with this Instrument and Form 43-101F1.
9. Steffen Robertson and Kirsten (Canada) Inc. was retained by Breakwater Resources Ltd. to prepare an Independent Technical Report on Breakwater in accordance with National Instrument 43-101 and to assist in the preparation of the technical components of a prospectus. The following report is based on our review of project files, operations data, discussions with Breakwater personnel, and discussions with SRK employees who visited the mine sites in 2001.
10. I was a co-author of the report.
11. I hereby consent to use of this report and our name in the preparation of a prospectus for submission to any Provincial regulatory authority.

"SEAL"
"Diana D. Sollner"

Vancouver, BC, Canada
January 11, 2002

Diana D. Sollner, P.Eng.
Environmental Engineer