EXPLORATION
OREX Inc.

AR 90-174

A re-appraisal of the Goldboro property
(Upper Seal Harbour district)
Guysborough County, Nova Scotia
Geology, metallurgy and ore reserves

(Revised edition)

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August 1990

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INTRODUCTION

This report deals with the analysis of geological, metallurgical and ore reserve calculation studies undertaken by Exploration Orex Inc. with respect to its 100% owned Goldboro property located in Guysborough County, Nova Scotia (see Figure 1). The property covers the historical, past-producing Upper Seal Harbour district - the second largest gold district of Nova Scotia. The district contains the Boston-Richardson mine (1892-1912) from which approximately 375,000 tons of ore yielded a little over 50,000 ounces of gold. Gold was mined from the Richardson belt, a "saddle type" orebody straddling the Upper Seal Harbour anticline.

In 1987-88, exploration below the Richardson belt led to the discovery of a major orebody bordered by significant quantities of low-grade mineralization. Exploration Orex Inc. then re-evaluated its exploration strategies and considered some type of surface or underground bulk mining scenario that would circumvent the problems of detailed correlations and erratic gold content.

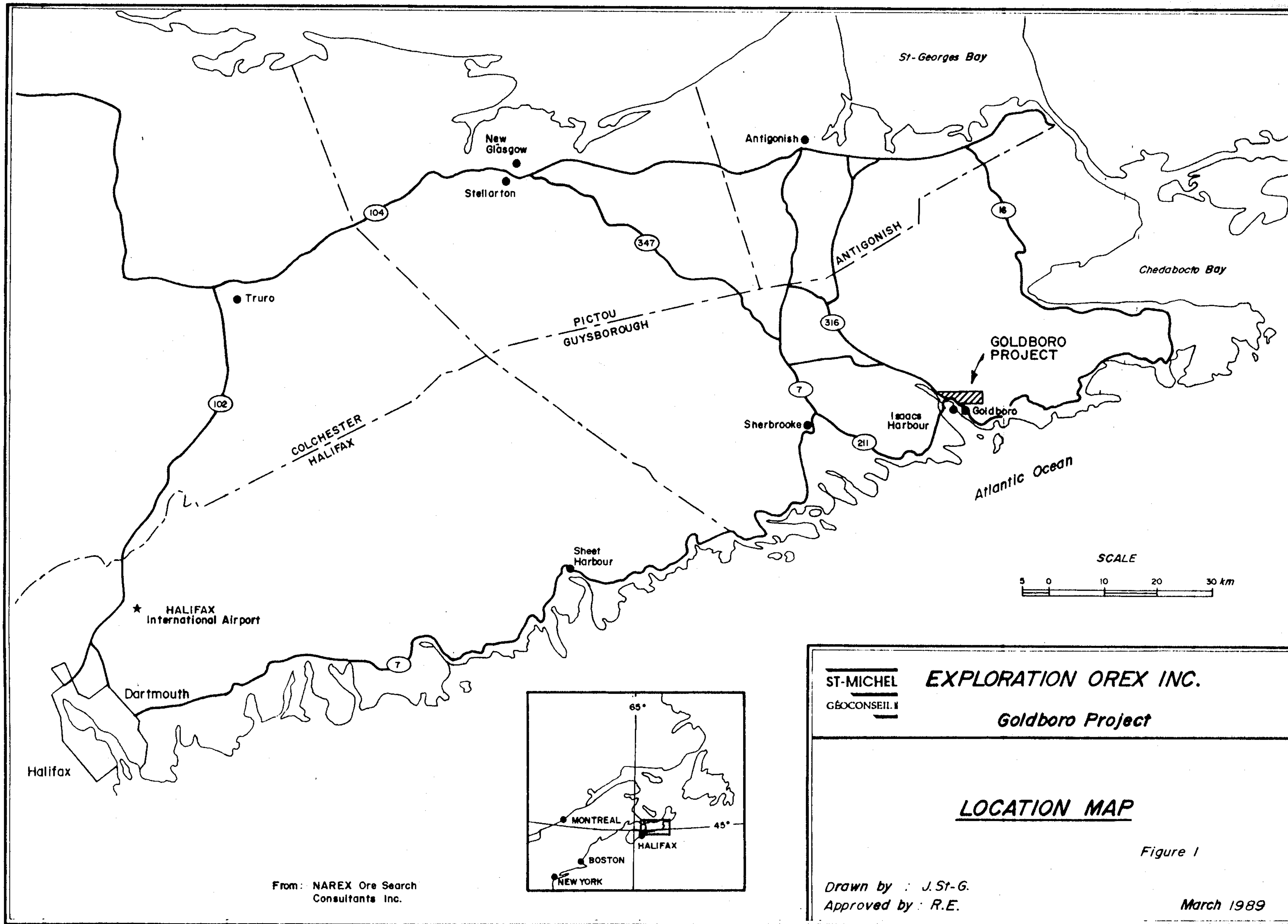
Geological, analytical, and metallurgical parameters of the project are reviewed in order to reassess its economic potential as a large open-pit operation. Property geology and economic potential were reviewed with respect to current metallogenic models (Smith and Kontak, 1988c). Analytical results of core sampling programs were reevaluated according to interpretation of gold grade determination mill tests. A grade correction factor was calculated and applied to the ore reserve estimation of an independent consulting group. Revised ore reserves are presented and an open pit mining scenario briefly reviewed.

EXPLORATION AND MINING HISTORY

Discovery of gold in the district goes back to 1892 when the Richardson belt, an auriferous shale-quartz horizon was discovered. Mining of the Richardson orebody was initiated the same year, giving birth to the Boston-Richardson mine. At the mine closure in 1912, approximately 375,000 tons of ore had yielded a little over 50,000 ounces of gold. From 1926 to 1927, attempts were made to recover the auriferous arsenopyrite from the tailings.

Sporadic development and mining operations were undertaken at neighboring sites along the same anticline: Dolliver Mountain, 2 km west of the Boston-Richardson mine; East Goldbrook located east of the Boston-Richardson property; West Goldbrook located west of the Boston-Richardson mine. In 1981, Patino Mines (Quebec) Ltd. completed a detailed geophysical program on a large tract of ground covering most of the Upper Seal Harbour District. In 1984, Onitap Resources Inc. acquired 37 claims (see Figure 2) including the Boston-Richardson, East Goldbrook, West Goldbrook and Dolliver Mountain properties. From 1984 to early 1988, Onitap investigated new mineralized belts stacked below the Richardson orebody by drilling 44 holes totalling 48,247 feet below the workings of the Boston-Richardson and East Goldbrook mines.

In 1988, Exploration Orex Inc. optioned the property from Onitap. Orex drilled 44 holes (BR-39 to BR-82) totalling 35,135 feet, below the old Boston-Richardson and West Goldbrook mine workings. This led to the discovery of a major auriferous belt package, traced continually for 500 m along strike and over 300 m at depth. Two exploration cross-cuts were driven through the zone on the 125 and 250 foot levels in an area located approximately 350 m west of the Boston-Richardson shaft which was rehabilitated. In early 1989, Orex acquired Onitap's remaining interest before 9,257 feet were drilled in 26 holes (BR-83 to BR-108) in the West Goldbrook area. A complete mill test (see test No. 2) was undertaken on 760 kg of muck in an attempt to compare initial sampling results with the grade generated by thorough metallurgical testing. Results from this test led to the undertaking of 10 more tests from mid 1989 to mid 1990. Finally, in the winter of 1989-1990, 15,555 feet of underground definition drilling was completed in the Ramp Area, on a selected portion of the new vein package to outline a proven ore block.



ST-MICHEL GÉOCONSEIL

EXPLORATION OREX INC.

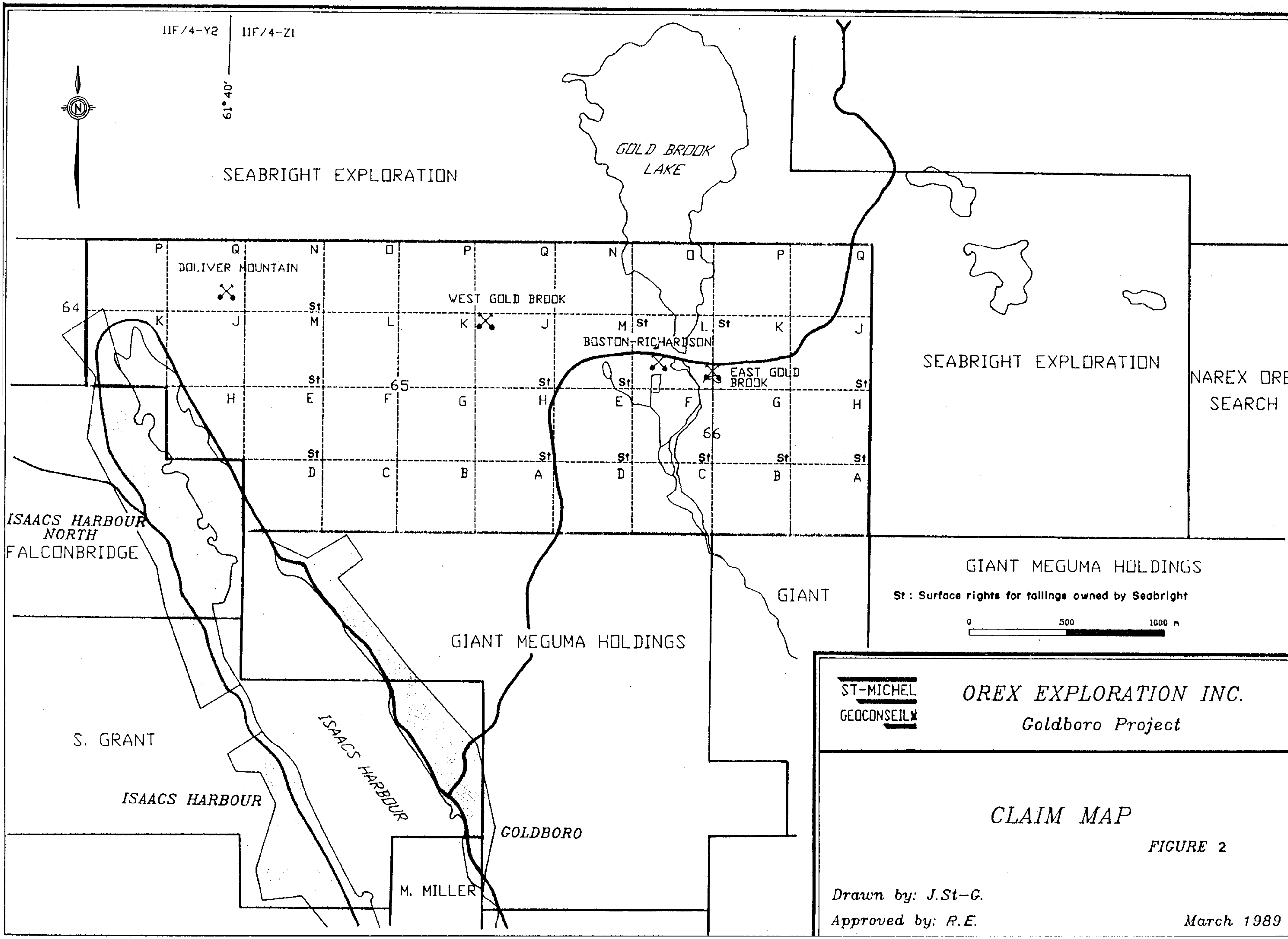
Goldboro Project

LOCATION MAP

Figure 1

Drawn by : J. St-G.
Approved by : R.E.

March 1989



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CLAIM MAP

FIGURE 2

Drawn by: J.St-G.

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PART I: GEOLOGY

REGIONAL SETTING

The regional geology of the Country Harbour area, eastern Nova Scotia, is dealt with only briefly (see Figure 3). For more information of that subject, the reader is referred to publications by Haynes (1983), Keppie and Smith (1978), Keppie (1983), O'Brien (1983), Poole (1967), Schenk (1970), Smith (1981, 1983). The geology of the Boston-Richardson mine was described by Faribault in Malcolm (1976), and more recently by Naert (1987), Parent and Ethier (1989), Parent (1990).

The Goldboro property is situated in the Meguma Terrane, a series of Cambro-Ordovician clastic sediments intruded by Devonian granitoids. Gold mineralization is hosted by silicified sandy turbidites. The Meguma Group sediments are divided into the lower Goldenville Formation, consisting of thinly to thickly bedded sandstones and greywackes with thin mudstones interbeds and the upper Halifax Formation consisting dominantly of thinly bedded mudstones with minor sandstones units (Keppie, 1983). Stratigraphic continuity of the host rocks and many of the semi-concordant auriferous quartz vein packages allows for excellent geological correlation and predictability of auriferous zones (Smith and Kontak, 1987).

Five phases of deformation are recognized in the Country Harbour area (Keppie, 1983). The earliest deformation, D_1, is typically expressed by grain alignment cleavage and rare folds. D_2, the main phase of Acadian deformation, precede regional metamorphism and produced subhorizontal to gently plunging, east-west, regional folds with upright slaty and pressure solution cleavages. D_3 is syn-plutonism (370 Ma) and is characterized by a pervasive subhorizontal crenulation cleavage preferentially developed in shales. D_4 is associated with ductile shear zones, numerous folds and strong upright cleavage, it deforms metamorphic porphyroblasts and has a progressive development. D_5, the last events recorded in the area, are expressed by several sets of kink bands, major northwest-southeast sinistral faults and late crosscutting quartz veins.

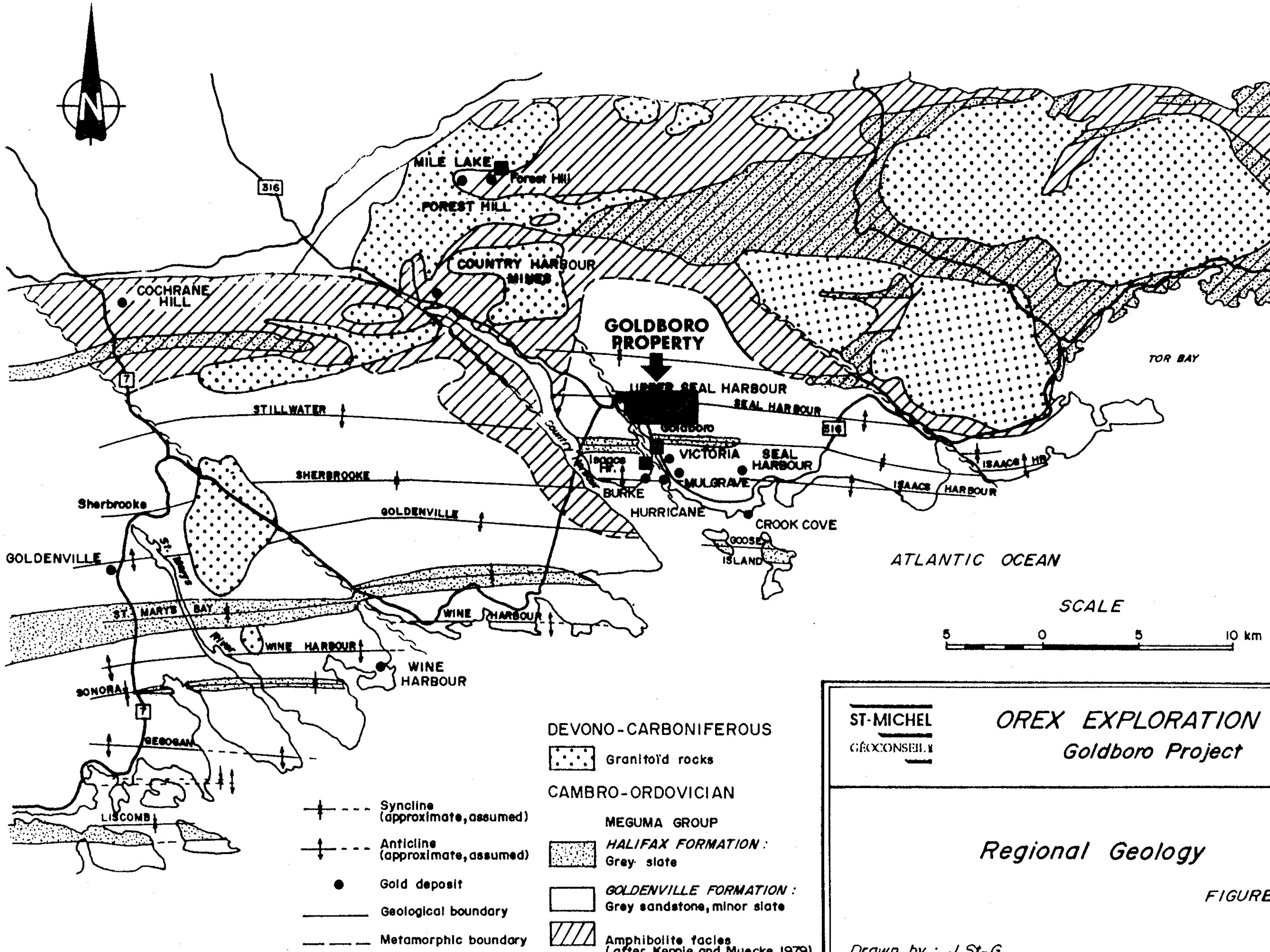
Regional metamorphism rises from the greenschist facies in the Isaacs Harbour-Goldboro area, to the amphibolite facies northward in the Forest Hill area, to the west along the Country Harbour fault and eastward along the New Harbour fault (Keppie, 1979). This metamorphism has risen from the greenschist facies during D_1, D_2 and D_3 into the amphibolite facies immediately following D_3 (Keppie, 1983).

BOSTON-RICHARDSON MINE AREA

Lithology

The stratigraphic section of the mine area can be described in general as turbiditic sediments of the Goldenville Formation, tightly folded in an upright regional fold, the Upper Seal Harbour anticline. With recent underground development and detailed drilling in the ramp area, stratigraphic relations within the section hosting mineralization are becoming evident. Correlation of individual stratigraphic horizons through the structurally complex zone of mineralization would require much more work, but results of preliminary work are summarized here.

Gold mineralization is recognized across a repetitive sequence of interbedded sandstones (arenites), greywackes and shales (argillites). The relative order of lithological packages from base to top across the anticline is given below. Shale units are preferentially thickened at the apex by a ratio of 1: 2.8 (Parent, 1989).



FROM : HAYNES, S.J., 1983

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Goldboro Project

Regional Geology

FIGURE 3

Drawn by : J. St-G.

Approved by : G.P.

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a) the *N1-N14* package, hosting the Main orebody, consists of at least 14 shale horizons or "belts", labeled N1 to N14, interbedded with arenites and greywackes (at least 100 m thick increasing to more than 300 m at the apex). Belt "A" previously recognized only on the north limb on the anticline (Parent,1990) is correlated to N1 by the writer.
b) the *Arenite Marker* unit, a major arenite-greywacke unit, containing occasional gold in large quartz stockwork and minor cross-cutting ac veins (40 m thick).
c) the *Richardson* belt, the originally discovered (2.5 m thick increasing to 6 m at the apex, 6.7 m thick in the "pay-shoots").
d) the East Goldbrook package, dominantly arenitic at the base, progressively evolving upward into a thinly bedded argillite and arenite sequence of unknown thickness, contains major auriferous apex [1] veins in the East Goldbrook and Boston-Richardson mines workings and well mineralized sections as far as 120 m from the apex in drill core and ramp workings of the south limb.

Structure

The dominant structural feature in host lithologies of the mine area is a penetrative fabric defined by pressure solution cleavage in the sandstones and slate cleavage in the shales. This fabric is related to the Upper Seal Harbour anticline, a regional F_2 fold generally striking ESE, with an axial plane dipping steeply south and an eastward plunge increasing from 10° at the west end of the property to 32° at the east end. A break in the linear continuity of the anticline occurs between the West Goldbrook and Boston-Richardson mines. Structural work by Keppie (1983) in the neighboring Isaacs Harbour district led to the recognition of an en echelon arrangement of minor anticlines which together form a major anticline. This leads to postulate that different anticlines underlie the West Goldbrook and Boston-Richardson Areas.

Numerous strike parallel and well mineralized narrow graphitic shears are hosted in the shale belts. Local stratigraphic discontinuity and thickness variation in the ramp workings may result from undetected faulting. Shear or fold related (?) flexures controlling high grade pay-shoots, oriented at shallow angles in the plane of stratification, are reported on both limbs of the Richardson belt (Faribault *in* Malcolm, 1976). A 2 m to 5 m wide, steep ENE fault cuts the anticline axis at a low angle in the vicinity of section 8650E and apparently truncates the north shoot of the Richardson belt on the 400 foot level.

Mineralization and alteration

The Goldboro orebodies consists of belts, or stacks of closely spaced belts, of shale, quartz veins, sulphides, and gold in a pervasively silicified and mineralized section of Cambro-Ordovician turbidites. Ore minerals include native gold and auriferous arsenopyrite, the native gold occuring in veins and wall rock. Voluminous quartz vein assemblages and alteration zones are an important feature of the orebodies. The various veins types encountered are typical of most Meguma deposits, their description is dealt with only briefly. For more information on that subject, the reader is referred to Smith and Kontak (1988b). Five vein types are described below.

a) ribbon veins: auriferous, nearly bedding parallel geometry and laminar texture, representing more than one episode of quartz emplacement, restricted to shale belts.
b) stratabound veins: auriferous, massive white quartz, confined within a particular strata and generally parallel to bedding.
c) irregular veins: gold less common, various geometry including en echelon.
d) bull crosscutting veins: trace amount of gold, discordant geometry and large size.
e) ac veins: high angle crosscutting orientation, gold common in small undeformed veins.

The highest grades of mineralization occur in shoots, the location of which appears to be determined by some irregularity of rock structure such as a subordinate flexure in the anticline or a fracturing of the strata. Two shoots, linear swollen zones in the belt, were identified in the original Richardson orebody. They plunged eastward at approximately 25°, averaging 6 m in width and 70 m to

---

1 Apex of the Upper Seal Harbour anticline.

100 m in length on the level, and were mined from surface to respectively the 525 and 700 foot levels. The shoots were characterized by strong arsenopyrite mineralization embedded in velvety black slate with relatively minor quartz. The hanging-wall, within the shoots, was "shattered" was injected by angular cross-cutting veins (Faribault *in* Malcolm, 1976). The location of the two Richardson shoots as well as mineralized and altered zones lying between them, within the N1-N14 package, might be controlled by the same structural plane. Further studies led to the outlining of four additional similar parallel planes which preliminary interpretation shows as a set of NW striking parallel planes with a 45° dip to the NE. Individual thickness and spacing of the planes varies from 5 m to 40 m (Hatchette, D. & Smith, P.K., personal communications).

Four distinctive orebodies or mineralized areas are described below.

a) the ***Main*** orebody is stratabound within the N1-N14 package. It has the shape of an upward thinning wedge averaging, on section, 45 m thick on the 250 foot level increasing to 80 m on the 400 foot level. It closely parallels the anticline geometry, thus plunging eastward. The north and south ore boundaries usually follow the contact with the arenitic Marker unit, except to the northwest where it is cut by the ENE fault. Mineralization is also reported within and north of the fault but information is scarce. The Main orebody is open to the east, the west, and at depth.
b) the ***Richardson*** orebody is stratabound within the Richardson belt. It is still open at depth.
c) the ***East Goldbrook*** package has a few significant mineralized zones of unknown extent, as indicated by a few exploration holes and underground developments on the south limb. Hole BR-35A collared on section 9012.5E returned 2.8 g/t Au over 110 m.
d) the ***West Goldbrook*** mine area is mineralized with gold along the apex, west of section 8600E, but without direct correlations with the Main orebody.

Pervasive alteration zones characterized by carbonate, sericite, sulphide, tourmaline and chlorite, postdating growth of porphyroblasts and development of regional cleavage, are common around the gold districts of Nova Scotia (Smith and Kontak, 1987). Little is known about alteration patterns related to the Goldboro orebodies, however, Parent and Ethier (1989) reported an extensive zone of alteration centered on the apex and characterized by very dark to black shales, conspicuous quartz veins related to silicification, and strong arsenopyrite mineralization. Faribault *in* Malcolm (1976) gives a similar description for the Richardson shoots on the 400 foot level.

CONCLUSION & DISCUSSION

Until recently, three historic facts were guiding exploration programs on the property:

- Shaft sinking, on the apex of the Upper Seal Harbour anticline, had proven a succession of saddle veins following the planes of stratification and gold was mined essentially from narrow high grade veins.
- Operations were restricted to the apex or its vicinity as it was speculated that the geometry of the deposit was influence by the fold.
- The precise geological controls of the gold mineralization were unknown, however, early workers recognized that gold in the Richardson belt was concentrated in apparently east dipping shoots, one on each limb of the anticline, at about the same angle as the plunge of the anticline, but receding from the anticlinal axis with depth.

Our review of the literature and our recent work adds the following facts or hypothesis to our understanding of the property :

- The Upper Seal Harbour anticline might consist of an en echelon arrangement of minor anticlines (see Keppie, 1983).

- Most of the vein formation in Meguma gold deposits is attributable to structurally controlled wall replacement along strike parallel shear zones postdating upright folding, regional metamorphism, and syn or postdating granitoid plutonism. Mineralization would be quite late with respect to deformation, metamorphism and plutonic evolution (see Smith and Kontak, 1988c).
- Secondary geological controls of the mineralization may be related to a set of very subtle NW striking parallel structures with a 45° dip to the NE.

These factors enhance our belief that the potential for a deposit of significant size is excellent.

- There is good potential for an extension of the main orebody along strike toward the west, south of the West Goldbrook area.
- Significant gold mineralization is not restricted to the apex zone, it should occurs throughout the major strike parallel shear zone. The width of this shear zone is still unknown, but as to now, shearing and gold mineralization were encountered within the ramp workings and in drill core for at least 200 m across the Boston-Richardson Area.
- Confirmation of NW striking, NE dipping, shoot related structures could generate a powerful exploration tool capable of predicting preferential sites for high grade mineralization across the whole mine area, including areas underlain by East Goldbrook lithologies.

PART II: METALLURGICAL STUDIES

In order to verify unsatisfactory analytical results from mineralized sections of drill core, Orex initiated a serie of grade determination tests on auriferous material from the Goldboro property. Twelve tests were performed, ranging considerably in size of sample and scope, they went from a few kilos processed by a single analytical method to several tonnes submitted to a sophisticated metallurgical evaluation, as illustrated in Table 1. Overall, they demonstrate that most grade determination methods recuperate only a fraction of the gold contained in the sample, therefore emphasing the problems related to ore evaluation. For more detailed information concerning the various tests, the reader is referred to Appendix 1.

Table 1: Gold grade determination tests - summarized description

Test #	Location	Sample	AA g/t	FA g/t	Met g/t	DM g/t	Cya g/t	Flo g/t	GC g/t	Remarks
1	Lakefield	1		2.15					4.30	Report incomplete,
		2		3.92					4.99	database not included
		3		15.7					16.9	
2	CRM	Mucks	1.99	1.99	2.05				3.31	Mill load not properly evaluated
3	CRM	Head of 2		3.8					7.45	Head assay non-representative
4	McGill	Head of 2		3.4					6.07	Head assay non-representative
5	CRM	BR-62	3.29	1.48					28.1	Poor reliability of AA but
		BR-60	0.30	0.72					18.2	large database.
		BR-48	0.58	0.91					12.6	
		BR-35A	0.08	3.51					2.78	
6	CRM	BR-61	0.79	3.30					4.25	idem as #5
		BR-85		0.52					5.06	VG sections excluded
7	Lakefield	BR-65-1			0.18				0.23	Inefficient gravity
		BR-65-2A			4.54				1.46	extraction.
		BR-65-3			3.59				2.79	
		BR-65-4			0.07				0.17	
		BR-65-5			0.82				1.09	
		BR-52-3			1.32				1.35	
		BR-52-4			0.51				0.27	
		BR-52-5			0.65				0.54	
8	Lakefield	BR-55	1.88	1.87	2.94		1.96	2.01	9.21	Weak nugget effect only
		UG sample			204		75	74	113	Strong nugget effect
9	CRM	BR-55	1.88	1.87	4.91		3.10	4.05	8.80	Homogenized sample
		UG sample			151		103	302	166	Medium nugget effect
10	TUNS	Vein		1.59/2.20					3.88	Poor data on head grade
		Argillite		5.81/4.50					7.56	
11	Lakefield	125 level		2.01		3.00			2.55	Loss of gold in overflow
12	Lakefield	250 level		1.11		1.20			4.07	Refined methodology

Lakefield : Lakefield Research Laboratory,
CRM : Centre de Recherches Minérales du Québec,
McGill : McGill University in Montreal,
TUNS : Technical University of Nova Scotia,
AA : Standard procedure for preparation and measurement by Atomic Absorption,
FA : Standard procedure for preparation and measurement by Fire Assay,
Met : Standard procedure for preparation, split at -100, -170 or -200 mesh, for Metallics Sieve assay,
DM : "Double Metallics" special Metallics Sieve assay conceived to evaluate large samples,
Cya : Direct cyanidation of finely grounded (80% < -200 mesh at least) material,
Flo : Direct flotation of grounded material,
GC : Gravity extraction followed by cyanidation of the tails after grinding.

The evaluation problem is twofold. First, the Goldboro ore is characterized by a random distribution of gold grains generating a nugget effect. This nugget effect is well illustrated by log-normal distributions of analytical results where the majority of values correspond to the background gold content while only a few very high values stand out above the real grade. Second, different analytical and ore processing methods yielded different results on identical portions of the same samples.

- Direct cyanidation is impaired by a high arsenopyrite and carbonaceous material content in the ore, as well as by large gold flakes that will not dissolve.
- Flotation will not recover the large gold flakes.
- Metallics sieve yields lower results for yet unknown reasons (interference from carbon or arsenic ?).
- Atomic absorption has the same drawbacks as direct cyanidation, added to the fact that the assayed portion is very small, between 10 to 30 grams.
- Fire assaying has the metallics sieve problems, added to the fact that the assayed portion is much smaller, between 15 and 60 grams.
- Gravity separation of free gold followed by cyanidation of finely grounded gravity tails after or without flotation yielded consistently higher values than the other procedures.

This indicates that a bias is induced by some of the methods. It has to be measured to calculate a correction factor for each analytical or ore processing method. These factors will eliminate the bias when multiplied by the laboratories results.

This lead us to conclude that only large populations (to reduce the nugget effect, caused by the small size of the portions assayed, impact on the reliability of results) of A.A, F.A or even metallics results can be used to evaluate the gold content of a specific unit and this, only if a correction factor is applied (to reduce the bias caused by the assay method) to bring the average of those assays closer to the real gold content. Considering the reliability of the tests, the correction factor should be based on results from tests 12 and 9 mostly (representative head assays and best extraction methodology) with less consideration given to tests 8, 6 and 5. Any factors derived from the other tests would only be misleading, positively or negatively. Please refer to Table 2 for a list of the most reliable correction factors to consider.

Table 2: Most reliable correction factors

Test	Metallics	Double Metallics	FA population	AA core assays
12		2.26	3.67	
9	1.79			
8			3.13 (1.54 to 5.37)	
5 & 6				26.21 (5.38 to 60.67)

However, to use these most reliable factors (Table 2) without thinking about the geology could lead to serious over or underestimation. The factors suggested by test #12 can be applied in a low background situation where large bull quartz veins with traces of nuggety gold and/or weakly mineralized arenites constitute the bulk of the ore while the factor given by test #9 should be applied to typical belt ore (high background sections) with fine grained gold associated to carbonaceous material and arsenopyrite. This segregation of numbers leads to the generation of two class of factors (see Table 3).

Table 3: Proposed correction factors

Geology	Metallics	Double Metallics	FA population	AA core assays
Low background	2.10	2.26	3.67	39.0
High background	1.79	1.93	3.13	7.0

Application of the proposed factors (Table 3) to grades based on assays results will have a major impact on the gold content estimate and therefore on the economics of the project. The conspicuousness of the analytical bias is, however, too strong to simply ignore the correction factors. Thus, we recommend a prudent approach implying reserve calculations using, first, the appropriate proposed factors to generate the most likely or "presumed" grades, and secondly, to calculate them again using the lowermost factor, 1.79, to provide a minimum estimate.

PART III: ORE RESERVES

MAIN OREBODY AND WEST GOLDBROOK AREA

A geology driven computer reserve estimation was carried out by an independent consulting firm, using SECTCAD software and the assay results (Parent, 1990). Ore reserve classification follows the USGS system. The mineralized area was divided in three portions reflecting variations in individual databases.

a) the Ramp Area (upper western portion of the Main orebody), generated class 2 measured resources (proven reserves) for an area with underground development and very high drilling and assaying density.
b) the Main orebody (exclusive of the Ramp Area) [2], generated class 1 and 2 indicated resources (probable and possible reserves) for an area systematically drilled for a large part, but poorly sampled and assayed.
c) the West Goldbrook Area, generated class 2 indicated resources (possible reserves) for an area systematically but loosely drilled and well assayed.

We recommend the use of correction factors to assayed grades in order to eliminate the analytical bias. According to rock type, one of the proposed factors from Table 3 should be used to obtain the presumed ore grade. However, the lowermost factor should be used if a bottom line approach is wished, thus using 1.79 for all types of assays. Extensive and mineralized altough unsampled portions of the Main orebody - exclusive of the Ramp area, were included in the ore reserves. Their grade was estimated using "low grade units" values from the Ramp Area since they represent the lowest expected grade in this environment. Original and reviewed reserve figures are presented in Table 4.

Table 4: Ore reserve estimates for the Goldboro project

	Tonnage tonnes	Assayed Grade g/t	Presumed Grade g/t	Minimum Grade g/t
MAIN OREBODY				
Ramp Area only - class 2 measured resources (proven reserves)				
Belts only	330,803	1.86	3.59	3.33
Low grade units	424,199	0.67	1.51	1.19
Exclusive of the Ramp Area - class 1 & 2 indicated resources (probable and possible reserves)				
Belts assayed	6,240,000	2.34	7.34	4.20
Belts not assayed	1,950,000		1.51	1.19
Low grade not assayed	11,310,000		1.51	1.19
WEST GOLDBROOK AREA - class 2 indicated resources (possible reserves)				
Hole BR-85	263,473	6.30	6.30	6.30
All other intercepts	8,351,344	0.35	0.67	0.62
TOTAL				
Ramp Area only	755,002	1.19	2,42	2.12
Main Orebody (less Ramp Area)	19,500,000	0.75	3.37	2.15
West Goldbrook Area	8,614,817	0.53	0.84	0.80

2 Refers to Boston-Richardson Area in ore reserve calculation of Parent (1990).

Reserve estimates for the Main Orebody - less the Ramp Area (see Table 4) are presented in function of the geology and availability of analytical results, instead of the 12,500,000 tonnes of class 1 and 7,000,000 tonnes of class 2 indicated resources presented by Parent (1990). The calculation of the class 1 indicated resources implied the use of analytical results composites from 47 holes for a total composite length of 6,468 meters. This was done to have an homogeneous database appropriate for the use of correction factors. However, valid tests results are available for five of those holes (BR-62, BR-60, BR-48, BR-61 and BR-55) for a total length of 282 meters, which, if they were integrated in the database, would increase the grade of those resources from 0.75 g/t to 1.46 g/t, a 95% increase in grade by updating only 4% of the database. This adds a lot of weight to the necessity of correcting the original assay database to compensate for the analytical method bias.

EAST GOLDBROOK PACKAGE

Areas underlain by the East Goldbrook lithologies, consisting of the former East Goldbrook mine and of the area bordering the Main Orebody to the south, were not included in the reserve estimates. However, limited exploratory work suggest a potential for significant quantities of ore. All the holes or underground workings show minimum thicknesses only, the upper (southern) boundary hasn't been outlined, and the only valid grade available is 2.8 g/t over a true thickness of 83 meters (mill test on hole BR-35A).

Table 5: Resources for the East Goldbrook package - south limb

Section	Hole	Thickness (m)	Length (m)	Tonnes to -100m
8650.0E	Ramp	105 meters	162.5 meters	4,606,875 tonnes
8775.0E	Ramp	105 meters		
8800.0E	BR-38	30 meters	43.75 meters	354,375 tonnes
8862.5E	BR-39	44 meters	68.75 meters	816,750 tonnes
8937.5E	BR-16	100 meters	43.75 meters	1,181,250 tonnes
8950.0E	BR-30	65 meters	37.50 meters	658,125 tonnes
9012.5E	BR-35A	110 meters	56.25 meters	1,670,625 tonnes
			TOTAL	9,288,000 tonnes

Also, hole BR-18 intersects the same package at minus 300 meters with some interesting assays. The potential of this East Goldbrook Area for yielding large tonnage of low grade material is excellent, as demonstrated by the above numbers giving nearly 10 millions tonnes for a limited portion of the south flank down to minus 100 meters only.

DISCUSSION

Major quantities of low grade material could be outlined South, North and East of the Main Orebody which is still open at depth and to the West (BR-87-03 intersected good grades 180 meters south of the apex, east of the area considered). The exact limits of the low grade mineralization still has to be defined, since the 220 holes drilled to date have been essentially restricted to a broadly defined auriferous halo straddling the apex zone .

The minimum amount of gold contained is considerable, between 1.6 and 2.4 million ounces, even excluding the East Goldbrook Area and the potential of adding to those reserves is excellent. However, a new accurate and practical assay method will have to be devised in order to raise the qualification of the reserves for mine planning, since correction factors can be used only on large population of numbers and not on individual assays.

OPEN PIT POSSIBILITIES

With current ore reserves of nearly 29 million tonnes and a good potential to increase these reserves, Orex management feels that this project is highly suitable for an open pit operation. To furthur emphasize this open pit potential, we estimated the quantities and ratios of different grades of ore for the central portion of the Main orebody, between 8712.5 E and 9012.5E. The reader is referred to Figures 4 and 5 as well as to the 1:1,000 geological map appended to this report.

As indicated in Table 6, some 9.6 million tonnes of high grade, 18.7 million tonnes of low grade and 24.5 million tonnes of unknown grade could be removed between sections 8712.5 and 9012.5 (strike length of 300 meters) for a ratio of approximately 1: 2: 2.5. Several factors could also improve these ratios, specially the extensions of the Boston-Richardson Area which really should be classified as high grade, a portion of the East Goldbrook belts that could improve from low to high grade and, evidently, the unknown grade sectors could yield high and low grade material.

Table 6: Distribution of ore grade types for typical cross-sections

Sections	AREA 8712.5E (m^2)	AREA 9012.5E (m^2)	TONNAGE / m. of strike length 8712.5E tonnes	9012.5E tonnes	8712.5E %	9012.5E %
High Grade						
Main Orebody	12,450	7,125	33,615	19,238	18%	11%
Ramp Area	4,150		11,205		6%	
Low Grade						
M-O North extension	9,300	1,500	25,110	4,050	14%	2%
M-O South extension	5,350	800	14,445	2,160	8%	1%
East Goldbrook South	10,300	19,000	27,810	51,300	12%	19%
Unknown grade						
East Goldbrook North		14,600		39,420		23%
Arenite	7,800	12,000	21,060	32,400	12%	19%
South Flank block	3,700	7,000	9,990	18,900	5%	11%
North Flank block	14,400	1,100	38,880	2,970	21%	2%
Total						
High grade	16,600	7,125	44,820	19,238	25%	11%
Low grade	24,950	21,300	67,365	57,510	37%	34%
Unknown grade	25,900	34,700	69,930	93,690	38%	55%

The areas covered by each geological unit on cross-section were calculated for 8712.5E and 9012.5E. These area values were then multiplied by their density in order to calculate the tonnages per meter of strike length. To get a mineable tonnage, the tonnes per meter values are to be multiplied by the length of the portion of the orebody under consideration. The ore was further divided in low grade and high grade to accommodate the different databases, and probable grades of different geological units, to allow rough estimations of global tonnage and stripping ratios.

This evaluation does not include either ends of the pit where high and low grade material would have to be removed to extract the apex portion or the West Goldbrook Area. It was done to prove the interest of open pit mining and delineate the areas that should be evaluated by the next definition phase. From the surface plans and sections, it is obvious that the unknown grade areas and the western extension of the main mineralized system will have a dramatic influence on the economics of the operation.

Nevertheless, the actual data more than justify the next step which should consist of identifying the first portion of the pit to be extracted and verifying the exact gold distribution in that sector. Priority should be given in the western partof the Main orebody where its extension should reach surface, south of the West Goldbrook mine area.

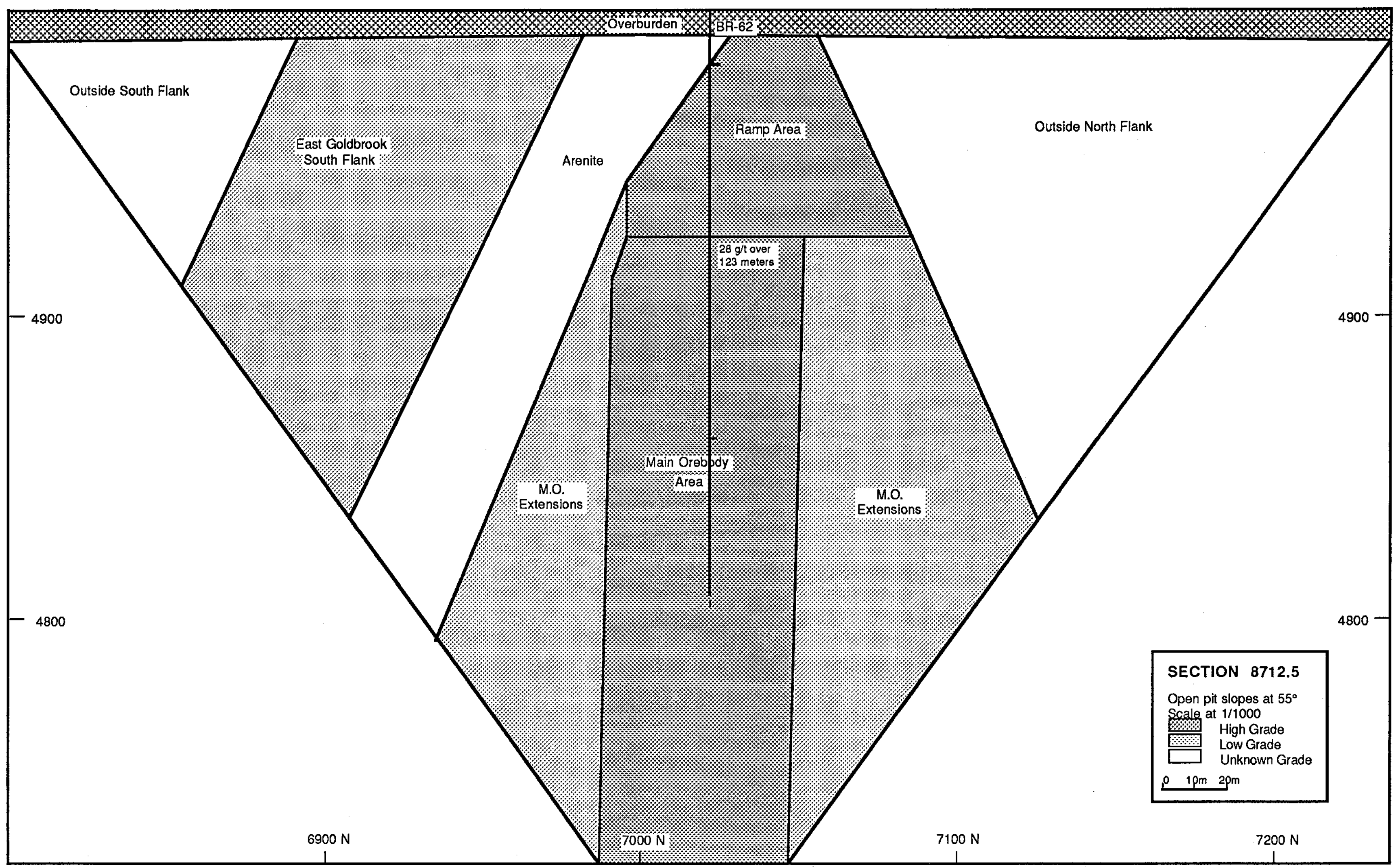

FIGURE 4: Section 8712.5E

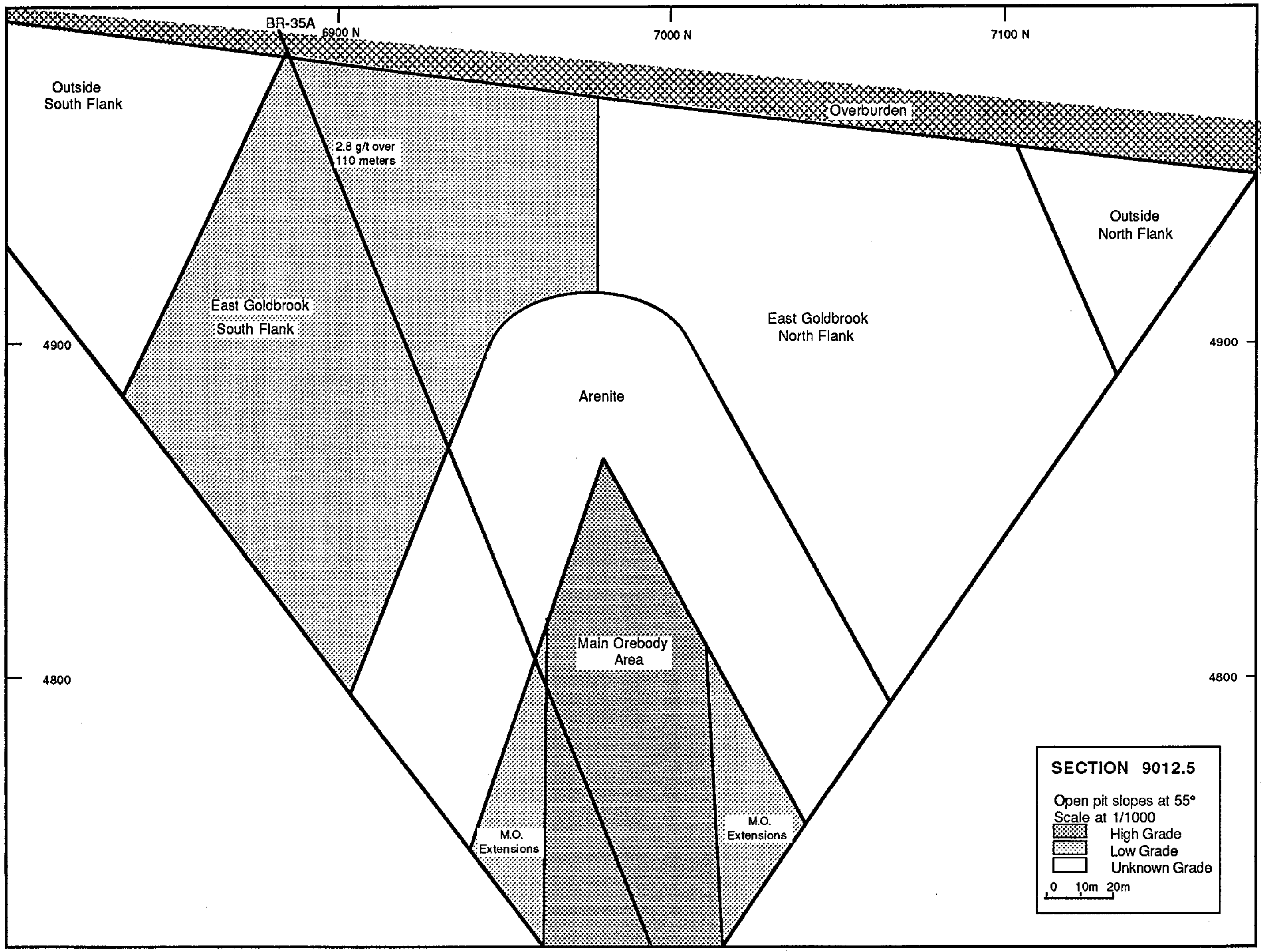

FIGURE 5: Section 9012.5E

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APPENDIX 1

DETAILED REVIEW OF METALLURGICAL STUDIES

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August 1990

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INTRODUCTION

Several tests have been conducted on ore material from the Goldboro property. They range considerably in size of sample and scope of the test from a few kilos testing one method to several tons with a complete metallurgical evaluation. Most of them yielded results showing considerable differences between head grades originating from direct assays of the material, and the calculated (real) grades. Also, most demonstrated that assay methods recuperated only a small portion of the gold contained in the sample. The assay methods used included Atomic Absorption (A.A.), Fire Assay (F.A.) and Metallics Seive Assay (Metallics or Double Metallics). The laboratories that conducted the tests were the Centre de Recherches Minérales du Québec (C.R.M.), Lakefield Research Laboratories (Lakefield), McGill University (McGill) and the Technical University of Nova Scotia, all institutions with the highest standards and untainted reputations.

TEST #1

The first metallurgical test done on ore from Goldboro was conducted at Lakefield, in May 1988, to verify gold recoveries using different methods. It suggested recoveries between 83 and 98.7% with 94% being the most common. Tentative comparison of mill grade to head grade showed a slight increase. Unfortunately, the head assays are based on only two normal F.A. giving a very low confidence level to the head assay and the tests are poorly documented in the report.

TEST #2

The second was more extensive and conducted at the C.R.M. in late 1988 with a report dated February 1989. It's objective was to compare the weighted average of assay results with the real gold content of muck samples from the 125 and 250 levels cross-cuts as well as give an idea of the probable mill grade if we mined the mineralized sections of those cross-cuts in bulk. The original samples (320) had been assayed by A.A. and, if results were between 1 and 3 g/t of gold, repeated by F.A. or by Metallics if the assay was equal or superior to 3 g/t. The mathematical average of those 320 assays was 2.23 g/t while the weighted average was 2.05 g/t.

The test was simple, it consisted of feeding the 320 crushed samples in a 76 X 91 cm ball mill, the mill discharge going to a -170 mesh screen which returned the +170 mesh material to the ball mill and sent the -170 mesh to a holding tank for cyanidation. The grinding circuit was operated until the discharge was clear and then cleaned, starting from the top of the circuit (screen), with all material washed dumped in the holding tank. Then the ball mill was opened and the mill load removed and kept separately for drying and assaying (F.A.). The holding tank material was then cyanided and the calculated assay came from the actual weights and direct assays of rejects and pregnant solution. Also, a head sample was taken by complete cuts of the feed every 15 minutes and assayed later by F.A.

The calculated head was 3.31 g/t and the direct assay was 3.4 g/t both superior to the original 2.23 or 2.05 g/t combined assay of the 320 samples. However, while the cyanidation balance can be considered as adequate, the material left in the ball mill (39.7 kg at 33.5 g/t) represented 52.8% of the gold and the assay is the average of four (4) F.A. being 10.9, 26.4, 27.6 and 69.2 g/t. Taking both extremes, it means that the real gold content should be between 2.13 and 5.18 g/t, also the four (4) F.A. done on the head sample varied between 2.0 and 4.3 g/t.

We concluded that:

- Even if the exact mill grade could not be pinpointed, it corresponded at least to the original assays and should be higher,
- More tests had to be done in order to verify the relation between assayed grades and real gold content,
- Complete gold extraction had to be done whenever gold nuggets were present, to reduce the uncertainty about grade to an acceptable level.

TESTS #3 and #4

A convenient sample had been generated by the second test, namely the head sample which represented fairly well the average material from the underground workings. It was used for the next tests at the C.R.M. and McGill. A two (2) kilos cut was taken, at the C.R.M., to check recuperations using a circuit with gravity separation up-front (Wilfley table) followed by flotation. This sample returned 7.45 g/t as total gold content, much higher than the head assays of 3.8 g/t (3 F.A.) or test #2 results of 3.31 g/t. But 80.5% of the gold came from the Wilfley table concentrate +200 mesh portion, so was probably coming directly from relatively coarse (+200 mesh) metallic gold, leaving 1.45 g/t as the probable very fine grained gold background.

Then a ten (10) kilos portion of the same sample was sent to McGill to test a Knelson concentrator as a potential gravity separator for coarse (-10 mesh) heavy material and verify the gold content per size fraction. This test showed a calculated head assay of 6.07 g/t, consistent with the 7.45 g/t obtained at the C.R.M. and again much higher than the test # 2 results of 3.31 g/t or average head assay of 3.4 g/t (4 F.A.). Furthermore the gold was distributed in all the size fractions of the sample with the -400 mesh material grading 3.30 g/t, consistent with the head assay grades, and the screen liberation size is at least -50 mesh, explaining the poor recoveries (28.61%) by the Knelson concentrator which operated on much coarser material.

These results suggested strongly that:

- The head assay grades represent the background gold content only,
- Relatively fine grinding, at least -50 mesh, is needed to liberate the free gold,
- A gravity circuit is recommended to concentrate the free gold prior to any other gold extraction method,
- The gravity circuit should not be operated on material coarser than -58 mesh,
- The head sample from test #2 graded at least 6.07 g/t, therefore implying that the real gold content of test #2 was probably closer to the maximum obtained (5.18 g/t) rather than the average of 3.31 g/t,
- If the original material left in the mill can be found, the above could be proven by processing it through a gravity circuit followed cyanidation of the rejects to obtain the real grade of test #2.

TESTS #5 and #6

The next test was similar to test #2. It was conducted at the C.R.M. in April 1989 and the final report was available on June 29th, 1989. The objective was to verify the correspondence between the weighted average of assay results and the real gold content of core samples representing strategic sections from several surface diamond drill holes. In most instances, the sections started with the top of belt N1 and ended at the bottom of belt N5, as interpreted then, relatively close to the apex below the Boston-Richardson belt. Two samples corresponded to cross-sections of a package of thin belts above the Boston-Richardson belt.

Three holes (BR-62, BR-48 and BR-60) were chosen initially out of the eleven selected originally, because of their position in relation to the ramp area underground workings as well as the high ratio of belts to arenites, and one was taken (BR-35A) to represent the walls of a potential open pit operation since it included a package of East Goldbrook belts. They were to be processed in the following order, BR-62, BR-60, BR-48 and BR-35A, starting with the best looking hole and ending with the probable low grade because the mill was available for only a few days and we could not predict with certainty how many holes could be processed in the short span of spare time.

The tests were simple, the methodology being similar to test #2 with the following exceptions:

1- The mill load was passed on a Wilfley table to collect a Wilfley concentrate that was entirely assayed and middlings and fines from which samples were taken and cyanided,
2- Samples were collected at the head and below the -170 mesh screen by taking complete cuts for five seconds every five minutes.

These two modifications were done in order to reduce the problems related to a strong nugget effect. However, it was assumed that cyanidation would be highly efficient even for small nuggets and that the Wilfley table could isolate nearly all nuggets in the concentrate to leave relatively homogeneous background material in the middlings and fines.

Two sets of data were available to give an idea of the probable head grade, the first consisted of A.A. and F.A. on a portion of the sections treated and the second of A.A. assays (F.A. if greater than 15 g/t) on core samples representing lengths between 0.5 and 1.5 meters. Evidently the first gives an incomplete idea of reality but still suggests something since all the "good looking" core was assayed while the second represent the total length with a relatively standard procedure.

The results are presented below in Table 1, in a summary of available data that gives a fair idea of the variability and unreliability of normal assaying even when repeated under close to similar conditions. This is specially evident when comparing the core assay to the partial and head assay for hole BR-62, the partial assay to the head or core assays for hole BR-48 and the head assay to the core assay for hole BR-35A. Also, the correlation factor has been calculated between the partial assays and their equivalent assay from the core assay population, the result (correlation factor of 0.08) shows the futility of predicting grades from those since they cannot even be repeated.

Table 1: Summarized results of test 5

Hole	Section Meters	Weight Kilos	Grade g/t	Head Assay g/t	Core Assay g/t	Partial Assay g/t / Meters
BR-62	123.4	326.5	28.07	1.48	3.29	1.61 / 59.4
BR-60	112.6	254.0	18.20	0.72	0.30	0.54 / 25.9
BR-48	102.1	249.6	12.59	0.91	0.58	10.38 / 26.2
BR-35A	102.7	432.5	2.78	3.51	0.08	----------------

When looking at those results, it is obvious that the grade does not correspond to the assays with two exceptions out of eleven trials. The core assays, representing the largest population of assays for each sample (between 121 assays for BR-62 and 95 assays for BR-60) and therefore the most accurate from a statistical point of vue, are between 8.53 (BR-62) and 60.67 (BR-60) times lower than the real gold content.

These extreme discrepancies led us to verify the possibility of contamination of our samples by material treated in the same circuit previously. The first step was in trying to identify the where, when and how of such an hypothetical contamination, when knowing that the complete circuit had been cleaned and cyanided prior to starting the processing of BR-62 and was cleaned completely between each hole afterward.

As for the where, the answer was simple because for the first three holes (BR-62, BR-60 and BR-48) the head assays show low values only and the gold content of the mill load is higher. Consequently, if contamination explained the discrepancies, it had to be between the sampler and the mill, either in the mill or in the four inches vertical pipe feeding it. The pipe was cleaned by circulating water until it ran clear of particles in suspension and the mill front end is removed for cleaning, leaving the interior easily accessible and visible with all lifters joints cemented with epoxy, the covering being a continuous stainless steel liner. The only portion of the mill not readily accessible was the feeding scoop which is the opposite of a gravity trap and should not contain any particles, specially after having run with only a cyanide solution for several hours prior to starting our tests and only water at the end of each test. We thus reached the conclusion that only minute quantities of material could have been left from the previous test and contamination was highly unlikely.

The next step was twofold, first reassay of cyanide solutions to verify the accuracy of the laboratory since most of the gold came from them and second a mineralurgical examination of gold particles from several portions of our tests as well as from the previous one. The reassay only confirmed that the assays used to calculate the metallurgical balances were accurate, they were done on twelve samples consisting of pregnant solutions from the cyanidation of the middlings and fines from the Wilfley table for all the holes. The small variations obtained could not even begin to explain the differences between grade and assays.

For the mineralurgical examination, eight samples were selected and, by concentration, fifty gold grains were collected from each giving four hundred gold grains to observe and describe. The samples selected were a head and cyanidation tails samples from the previous test, the cyanidation tails of BR-35A, BR-48, BR-60 and BR-62 and one sample each from the screen underflow and mill load of BR-62. It was assumed that if there was any contamination, the best place to find it would be in the tails, at the end of the circuit.

The mineralurgical examination showed that the gold grains from the previous test were thicker and covered by a thin film of rust that could be identified visually and in the microbe specters by a strong "Fe" peak. Out of four hundred gold grains, the one hundred coming from the previous test could be identified because they were thicker and covered with rust while none of the three hundred coming from the Goldboro samples had those characteristics.

The overall conclusion was that it was impossible that contamination had occurred and the reassay of selected solutions showed that the original assays, used to calculate grades, were accurate. Of all the gold grains observed, it was easy to identify those that came from Goldboro and no grains from the previous test was observed in the three hundred grains found in the Goldboro samples. If the differences between head grade and final grade had to be explained by contamination, sixteen grams of gold would be needed for a total of 5,361,257 gold grains of a size similar to those observed (125 x 125 x 10 microns) and they would have represented 93% of the gold grains present at all stages, the possibility that the three hundred gold grains found all came from the remaining 7% portion of the total population is nil for all practical purposes.

Nevertheless, it was decided to process two more holes (test #6), in August 1989, in a close to identical manner, the exceptions being that the clean-up material would be kept with the mill load and treated with it on the Wilfley table and the screen underflow would be assayed by partial cyanidation. Before starting the process, all piping was changed, a new -170 mesh screen installed, the mill scoop was removed, sandblasted and covered with white epoxy, all other parts of the circuit either changed or cleaned thoroughly if feasible and the balls in the ball mill were changed to a new set that had been used previously only to grind iron ores. Having thus insured that there was no possibility whatsoever of contamination, the holes were processed. The first, BR-61, came from an environment similar to BR-62, BR-60 and BR-48 including the interpreted N1 to N5 belts in a section that seemed low grade with minimal alteration, quartz veining and mineralization compared to the previous holes. The second, BR-85, was an angular hole transecting the anticlinal structure 400 meters west of the ramp area. It was considered an excellent hole with six sections containing visible gold including one spectacular zone. The results are presented in Table 2.

Table 2: Summarized results of test 6

Hole	Section Meters	Weight Kilos	Grade g/t	Head Assay g/t	Core Assay g/t	Partial Assay g/t / Meters
BR-61	73.2	194.0	4.25	3.30	0.79	1.71 / 27.1
BR-85	99.7	264	5.06	0.52	-------	----------------
V.G. (5)	5.2	13.7	32.4			
V.G. (1)	0.9	2.6	473.8			
BR-85 tot	105.8	280.3	10.63			

Again, large discrepancies can be observed between the assays and the grade even if BR-61 should have been a low grade hole with only a minimum of free gold and BR-85 best problematic sections, those containing visible gold (V.G.), had been removed in an attempt to reduce the nugget effect. This was considered the final proof needed to say that no contamination had occurred in the first holes and the explanation of the discrepancies between assays and grade had to come from another source, probably the nugget effect. However, such high variations implied an extremely strong nugget effect in an environment were the gold grains, in general, were relatively small. This can only be explained by a strong segregation component to the nugget effect, suggesting that the small gold grains could be found close to one another giving the overall effect of a very large grain. That is unless the assay method generates a bias for an unknown reason.

The general conclusions of test #5 and #6 were:

- Large discrepancies were generally observed between assays and grade even if the assays consisted of a relatively large population,
- A few large gold flakes were observed, up to 0.63 grams in the high grade sample from hole BR-85,
- After grinding, most of the gold grains were small, .125 mm in diameter, very thin (less than .01 mm) flakes that could behave strangely in a liquid environment,
- Cyanide tails would be very difficult to evaluate (a 90 g/t sample had been collected from the tails of BR-60 when the operator failed to flush the valve content prior to sampling. Upon visual observation with a binocular, the gold consisted of extremely thin flakes that seemed unaltered even if they had been in a cyanide solution for a 48 hours period. Evidently, that assay has not been included in the final metallurgical balance.),

- Even very fine grained material, from which the coarse gold had been removed, showed high variations in assays,
- Visual estimation based on alteration strength and mineralization pinpointed exactly the best holes while assays were random,
- The central portion of the anticlinal were extremely high grade (BR-62, BR-60 and BR-48), with the grade diminishing on the flanks (BR-61) for the same belts,
- The whole cross-section of the anticlinal could yield economic values (BR-85),
- The package of thin belts(BR-35A), corresponding to the East Goldbrook belts, could yield very interesting grades completely outside of the main area of interest,
- The next steps would be to get similar results from other holes and laboratories and verify if the assay methods were flawed somehow.

TEST #7

This test was the continuation of the precedents, it was done in July and August 1989 at Lakefield and the report was completed on October 19th, 1989. The samples corresponded to interpreted geological units, belts N1 to N5 or arenites separating them, from two diamond drill holes. The first, BR-65, came from a position close to the apex between BR-62 and BR-60 which had been tested before, and the second, BR-52, was a vertical hole drilled 32 meters south of the apex. The samples represented shorter sections of the drill holes in order to better identify the gold distribution within the N1 to N5 belts package.

Each sample was first crushed to -20 mesh before being fed to a Falcon concentrator to produce a gravity concentrate, that was treated later on Wilfley and Mozley tables, and gravity tails that were batch grounded in a ball mill to 90% -200 mesh. The grounded material was then removed and cyanided for 48 hours, with activated carbon being added in the last hour to collect the gold from the pregnant solution. The results are given in Table 3

Table 3: Summarized results of test 7

Sample	Length Meters	Weight Kilos	Grade g/t	Metallics g/t	Partial Assay g/t / Meters	Cyan. g/t	Cya head g/t
BR-65-1	20.3	61.1	0.23	0.18	0.00 / 0.0	0.18	0.24
BR-65-2A	24.2	58.8	1.46	4.54*	2.43 / 19.7	0.69	0.78
BR-65-2B	25.8	54.6	2.79	3.59	3.94 / 16.4	2.48	3.75*
BR-65-3	29.5	75.5	0.17	0.07	0.00 / 4.3	0.13	0.12
BR-65-4	25.8	51.6	0.82	1.09	0.71 / 17.0	0.35	2.43
BR-52-2	26.8	51.2	0.46	------	0.24 / 2.7	0.37	0.71
BR-52-3	13.0	23.6	1,35	1.32	1.76 / 6.9	0.58	1.02
BR-52-4	28.5	74.2	0.27	0.51	0.39 / 2.44	0.22	0.19
BR-52-5	20.6	47.8	0.54	0.65	0.46 / 14.5	0.41	0.42

* More assays were taken than usual but only those corresponding to the usual sampling pattern are indicated.

The most obvious suggestions of the above numbers are that we deal with very low grade material with no nugget effect. In general, the head assays are higher or very close to the metallurgical balance (in theory the real gold content) even for the fine grained material that was cyanided. This is contradictory in principle, in the sense that the higher than real head assay have to be explained by a nugget effect but, if this nugget effect exists, the head assays will normally be lower than the real grade. An attempt was made at solving this problem by passing the tails of sample BR-65-2B on a Wilfley table at the C.R.M. This showed a grade of 0.35 g/t for the residue instead of the previous 0.03 g/t which was more in line with the fine grained portion from the Wilfley table (0.05 g/t).

These unexpected results showed that questions still had to be answered before a definite method could be used for gold determination of the Goldboro ore. Knowing the material sent, it seemed obvious that the methodology used missed some of the gold. The most likely explanation is that the Falcon concentrator could not remove the free gold because a significant portion had not been liberated yet and, consequently, most of the gold flakes ended in the cyanidation tank where they were not dissolved properly. This explanation is based on the observations of the test #4 done at McGill for the liberation size and the mineralurgical observation of gold grains, after grinding, done at the C.R.M. for test #5 (we received this data only after starting test #7).

In order to prove this, new tests were needed to demonstrate that direct cyanidation was not efficient and dissolved only a small portion of the gold and a large sample with an excellent database of assays had to be processed to show that assays, however sophisticated, yielded significantly lower grades than milling.

TEST #8 and #9

Following all the above, we felt that a test should be made specifically to compare metallurgical and assay methods. All the previous data, with the exception of test #7, on ores from Goldboro and elsewhere in Nova Scotia showed a strong nugget effect due to the presence of large gold nuggets and strong segregation of the gold grains, so the test would have to take that into account.

It was decided that in order to compare adequately and reduce the randomness of the nugget effect influence, two laboratories would be used treating splits from the same original sample. A 27 kilos sample, assay rejects of eight samples representing an 11.4 meters section from diamond drill hole BR-55, was sent to Lakefield for preparation. Those rejects were first mixed and then crushed to 100% -20 mesh. Also a 10 kilos sample was sent from Goldboro, it was a cocktail mix of high grade material including several blocks with V.G. and others with high carbon and sulphides content coming from the ramp area. This last was sent to compare variations in methods between very high grade and normal ore. It was also crushed to 100% -20 mesh.

Table 4: Summarized results of tests 8 and 9

	SAMPLE 1		SAMPLE 2	
Nature of the test	Lakefield g/t	C.R.M. g/t	Lakefield g/t	C.R.M. g/t
Original A.A. (Assayers To.)	1.88	1.88		
Original F.A. (Assayers To.)	1.87	1.87		
1 Kilo direct Cyanidation	1.96	3.10	74.9	103.1
1 Kilo direct Flotation	2.01	4.05	74.0	302.1*
1 Kilo F.A or Metallics	2.44	4.91	204.0	150.5
1 Kilo F.A.	3.44			
1 Kilo Gravity / Cyanidation	13.10	8.80	113.0	165.7
1 Kilo Gravity / Cyanidation	5.31			

* Two large grains were removed from the tailings directly, their loss would have reduced the grade to approximately 100 g/t.

Then a 5 kilos sample was riffled from both and sent to the C.R.M. to duplicate the tests being done at Lakefield. Afterward, at Lakefield they riffled out several one kilos portions from each sample and proceeded to do the tests on those and at the C.R.M. they homogenized the samples in a special unit prior to riffling in one kilo portions. The tests per se were the following for each sample:

- Direct cyanidation after grinding to approximately 80% (Lakefield) or 90%(C.R.M.) -200 mesh,
- Direct flotation after grinding to approximately 80% (Lakefield) or 90%(C.R.M.) -200 mesh,
- Total fire assay by repeating 30 grams F.A. at Lakefield or metallics on a +200 mesh/ -200 mesh separation at the C.R.M.,
- Gravity extraction with a Wilfley table, concentrate upgraded with a Mozley table at Lakefield, with the tails being grounded to approximately 80% (Lakefield) or 90%(C.R.M.) -200 mesh before cyanidation.

Also available were the original A.A. and F.A., from Assayers Ltd in Toronto, for the original core samples. The results are presented in Table 4.

Up to a certain point, those results were the most significative given by any metallurgical test on ores from the Goldboro property. For the first time, relatively large samples (1 kilo) of identical material were assayed by different methods to see if the assaying methods induced a bias. Sample 1 yielded results quite similar to past tests, suggesting an important bias related to the analytical method, totally independent of the nugget effect. If the differences were due only to the nugget effect and probability distribution, there would be 1/5,040 chance of getting them in the order given for Lakefield and 1/360 chance for the C.R.M. for a combined probability of 1/1,814,400. For sample 2, the grade and associated nugget effect is too large to yield any strong conclusions. However, the trend is identical even if partially obscured by the nugget effect and a zealous operator who picked free gold from the flotation tails.

Since we know that the C.R.M. used a more efficient homogenization method than Lakefield and that sample 1 has a much smaller nugget effect, we assume that the results, at the C.R.M., reflect the assaying bias freed of the probability distribution of gold grains. Therefore direct cyanidation seems to recover 35% of the gold, direct flotation 46% and metallics sieve assay 56%, if we used Lakefield's results, those recoveries would be even lower. These numbers can be used to generate a correction factor to eliminate, or at least reduce considerably, the bias associated with certain assay methods. In the case of the original assays, since they were done on smaller non-homogenized samples, it is impossible to draw conclusions except that the correction factor should be applied on large populations of numbers only and it would be more conservative to use a factor identical to the metallics sieve assay since it is the lowest.

TEST #10

This test was conducted by the Minerals Engineering Centre for the Technical University of Nova Scotia and consisted of crushing and grinding two samples (Vein and Argillite) prior to tabling and cyaniding the table's tails. It was only a simple test to verify the differences between assays and grade for two types of material, as a follow-up by the Nova Scotia Mines Department geologists on Orex past observations. The results are given in Table 5.

Table 5: Summarized results of test 10

Sample	Weight Kilos	Grade g/t	Fire Assay g/t	Neutron Activation g/t
Vein material	7.2	3.88	1.59	2.20
Argillite	5.2	7.56	5.81	4.50

These results are in line with the rest but the limited information on the source material precludes from using them to generate correction factors for large populations.

TESTS #11 and #12

These tests were the most extensive in terms of size and information gathered. Two samples were collected underground by slashing the walls of the 125 and 250 cross-cuts in an area containing typical vein material to give close to seven tonnes of rock for each. Each muck pile was sampled thoroughly twice, the first sample was collected by taking pieces from all over the muck pile to make a composite sample that was assayed by double metallics and the second sample was taken on a regular grid pattern by collecting two kilos samples around each grid point (20), then each of those two kilos was crushed and splitted in two, each half being then assayed three to five times by standard F.A. Also, chip samples were taken on the walls across the width slashed and were assayed following the same procedure of multiple F.A.

It is not the objective of this report to relate the wealth of information given by those tests but the salient points concerning the assays bias will be explained. First and foremost, we had a good database on the sources with a very large, more than 40 kilos, sample that was assayed by a modified metallics sieve assay and good populations (133 and 153 assays) of F.A. from samples collected on the grid. The chip samples also gave interesting results but are considered less representative of the whole because of the two dimensional nature of such a sample compared to a "three" dimensional muck sample. The results are given in Table 6.

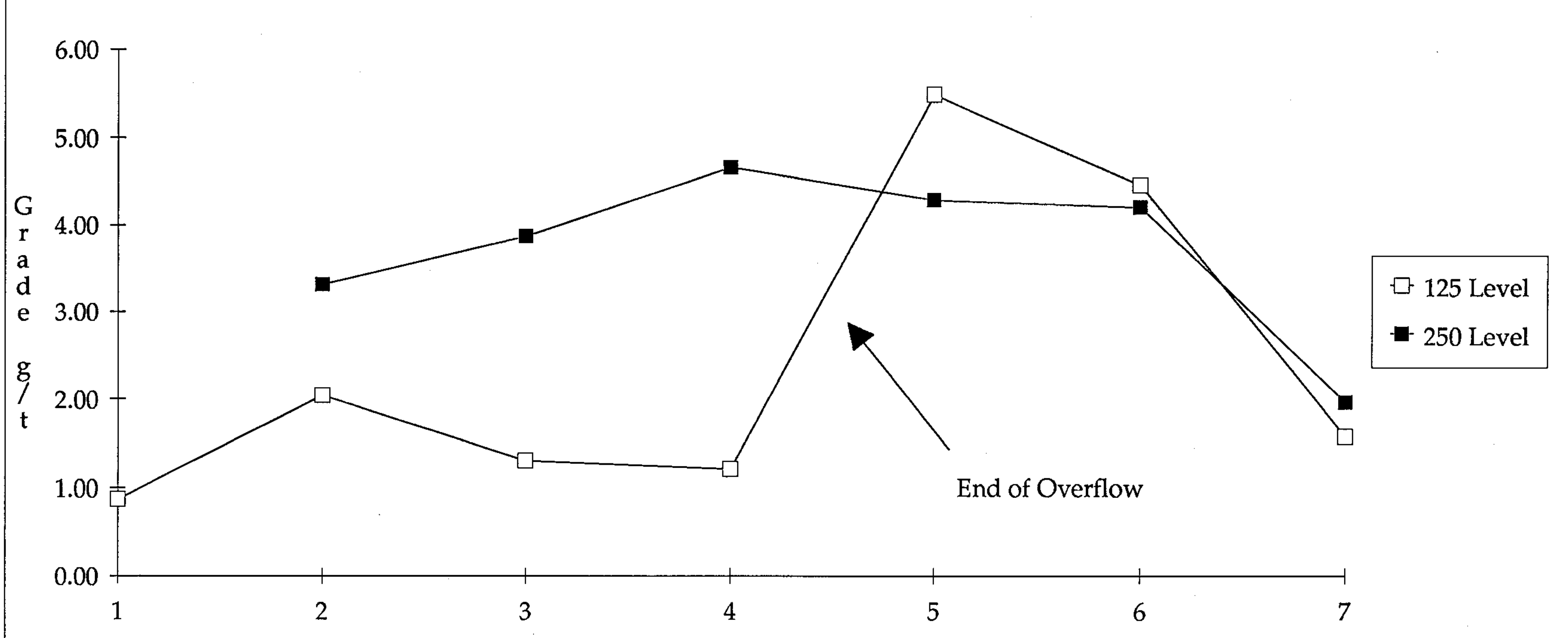
Table 6: Summarized results of tests 11 and 12

Sample	Weight Kilos	Grade g/t	Metallics g/t / Kg	Mucks F.A. g/t / nb F.A.	Chips F.A. g/t / nb F.A.
125 level	6,728	2.55	3.33 / 41.9	2.01 / 153	3.22 / 25
250 level	6,828	~~4.07~~ 4.5	1.80 / 44.7	1.11 / 133	2.09 / 44

Voir Michel Roy, Drex

Again it can be seen that the assays do not reflect reality. This is specially evident for the 250 foot level sample which contained a greater proportion of it's gold as free grains in quartz veins, the real gold content was more than three times the grade suggested by the assays. For the 125 level sample, the situation is much more complex, first a greater proportion of the gold was fine grains associated to black argillites or arsenopyrite and second a large proportion of the material was lost due to thickener overflow for most of the test and the grade changed dramatically after this was stopped. No firm conclusion can be based on this but the evidence is strong that gold was lost and not measured in the overflow, as shown by the "Grade Variations per Shift" graph were the change in grade is obvious for the 125 level, with or without overflow, compared to a relatively steady grade for the 250 level which should have been the more erratic because of the higher proportion of free gold.

FIGURE 1: GRADE VARIATIONS PER SHIFT



Other strange results of these tests include the steady reduction in total gold content of the cyanidation circuit starting at the 24 hours mark and the reprecipitation of gold in the leach solution after 24 hours also.

CONCLUSION

When considering those results, it is evident that a grade evaluation problem exists. The problem is twofold, first the ore from Goldboro is subject to a random distribution of gold grains generating a nugget effect. This "nugget effect" is evidenced by log-normal distributions of assay results giving a large number of assays corresponding to the background gold content and a few very high values above the real grade.

Second, different grade determination methods yielded different results on identical samples. Thus a bias exists, caused by the assay method, and has to be measured to generate correction factors for each method that will eliminate the assay method bias when multiplied by the assay results. Direct cyanidation is impaired by the high arsenopyrite and carbonaceous material content and the large gold flakes that will not dissolve; flotation will not recover the large gold flakes; metallics sieve yields lower results for yet unknown reasons (interference from carbon or arsenic?). The procedure that yielded consistently higher values incorporated efficient gravity separation of free gold followed by cyanidation of finely grounded gravity tails after or without flotation.

This lead us to conclude that only large populations (to reduce the nugget effect impact on the reliability of results) of A.A, F.A or even metallics results can be used to evaluate the gold content of a specific unit and this, only if a correction factor is applied (to reduce the bias caused by the assay method) to bring the average of those assays closer to the real gold content. In an attempt to determine this factor, let us consider the following:

Table 7: Validity evaluation

Test	Source data	Method	Overall	Comments
1	Very poor	Good	Poor	Report incomplete, database not included
2	Excellent	Good	Medium	Would be excellent overall if the mill load had been properly evaluated
3	Medium	Excellent	Medium	Source assays too limited for firm conclusion
4	Medium	Excellent	Medium	Source assays too limited for firm conclusion
5	Good	Excellent	Very good	Poor reliability of original assays but large database
6	Good/NA	Excellent	Very good	see #5 except no assays for BR-85
7	Poor	Poor	Poor	Method flawed: inefficient gravity extraction
8	NA	Very good	Very good	Comparison of assay methods, some nugget effect obscuring the results partially
9	NA	Excellent	Excellent	Comparison of assay methods, very limited nugget effect, in sample 2 only.
10	Poor	Excellent	Medium	Interesting as confirmation only
11	Very Good	Medium	Medium	Flawed method due to loss in overflow
12	Very Good	Excellent	Excellent	Refined methodology based on the results of the previous 11 tests

Based on the above, any correction factor should be based on results from tests 12 and 9 mostly with less consideration given to tests 8, 6 and 5. This would suggests the following factors in function of the original assay method:

Table 8: Correction factors obtained

Test	Metallics	Double Metallics	FA population	AA core assays
12		2.26	3.67	
9	1.79			
8			3.13 (1.54 to 5.37)	
5 & 6				26.21 (5.38 to 60.67)

However, to use the above factors without thinking about the geology could lead to serious over or underestimation. The factors suggested by test #12 can be applied in a low background situation where most of the gold consists of nuggets (bull quartz veining mostly) while the factor given by test #9 should be applied to argillites (high background sections) with fine grained gold associated to carbonaceous material and arsenopyrite. This segregation of numbers leads to the following:

Table 9: Correction factors suggested

Geology	Metallics	Double Metallics	FA population	AA core assays
Low background	2.10	2.26	3.67	39.0
High background	1.79	1.93	3.13	7.0

The low background factors could be applied to low grade sections containing some quartz veins and the high background factors can be applied to the belts. These factors, needless to say, can be applied only to large populations of assays and it would be prudent not to use those relating to AA core assays unless special circumstances warrant it. Also, in cases of doubt as to which factor to use, the smaller number should be taken. It is evident that applying those factors to ore reserves based on assays will have a major impact on the gold content and therefore on the economics of the project. However, the evidence of their existence is too strong to simply ignore it.

Legend

Trace of belt on surface

Outline of reserve block

Definite

Extensions possible

LOCATION MAP

PROPERTY BOUNDARY

AREA COVERED BY MAP

SCALE: 2 inches to 1 Mile - 1/31,680

NOTE: U.T.M. coordinates correspond to the grid coordinates of the Goldboro Property

ex: 4 488 700 E, 5 007 100 N (U.T.M.)
8700 E, 7 100N

AR 90-174

EXPLORATION OREX INC.
GOLDBORO Project

GEOLOGICAL OUTLINE OF THE BOSTON RICHARDSON MINE AREA

SCALE 1:1000