MICHAM EXPLORATION INC.

1988 SURFACE EXPLORATION PROGRAM

BANNOCKBURN PROJECT

MADOC, ONTARIO

OCTOBER 1988

By: David R. Bell,
David R. Bell Geological Services Inc.,
8 Church Street,
St. Catharines, Ontario

TABLE OF CONTENTS

SUMMARY

Modern exploration of the Bannockburn property began in 1981 and continued during the years 1984 to 1988. Much of the work comprised surface diamond drilling; the second last phase, in early 1988, consisted of a decline to the 75 ft. level and drifting on three veins.

During the summer of 1988 a surface exploration program was carried out on the property and surrounding area in the hopes of finding additional gold bearing quartz veins to further supplement the known geological reserves.

There were no new gold bearing quartz veins found in the surrounding area however in the northwest area of the northeast claim block on the Bannockburn property a northwest trending quartz vein returned values of 0.976 oz's Au./ton over a 2 foot width and 1.681 oz's Au./ton over a 1 foot width.

INTRODUCTION

The Bannockburn Project encompasses the old Bannockburn gold mine and other mineral rights holdings in Madoc Township, eastern Ontario. The modern period of exploration began in 1981 when Mono Gold Mines Inc. acquired these rights and initiated surface exploration. In 1984, they discovered new gold-bearing zones in the 'Northeast Block', where much of the subsequent work has been concentrated. Since that time Mono has carried out an extensive surface diamond drill program over the 'Northeast Block'.

In October 1987, Micham Exploration Inc. entered into a joint venture with Mono Gold Mines to further explore and develope the property. Micham Exploration, as operator of the joint venture, conducted the most recent work which comprised driving a decline, exploration drifting and underground diamond drilling in the north east block. To assist in accessing the potential of the area an airborne geophysical survey was also carried out in February 1988.

During the summer of 1988 a surface exploration program was carried out on the townships of, Tudor, Grimsthorpe, Elzevir and Madoc in which the Bannockburn property is located. The exploration program consisted of: ground examination of airborne geophysical anomalies; prospecting for gold bearing quartz veins and their subsequent trenching and sampling; surface stripping and diamond saw sampling of selected areas on the Bannockburn property.

BANNOCKBURN PROJECT

MADOC TOWNSHIP, HASTINGS COUNTY

GENERAL LOCATION MAP

Dkb.

SOUTHERN ONTARIO MINING DIVISION	FIGURE 1
NTS REF. 31 C/12	

LOCATION AND ACCESS

The Bannockburn property is located in southern Ontario approximately one kilometer northeast of the hamlet of Bannockburn, which is located on highway no. 62 about 18 kilometers north of the town of Madoc. Madoc is some 240 kilometers by road northeast of Toronto and lies at the junction of highways no. 7 and 62.

Access to the property is via highway no. 62 north to Bannockburn. Approximately one-quarter of a kilometer north of the Cooper Side Road on the east side of the highway is an all weather gravel road. This one kilometer long road leads directly to the northeast claim block, where the decline is located. With in other parts of the property and surrounding area access is by township and county gravel roads or private roads and trails.

PHYSIOGRAPHY

This part of Hastings County is divided into two different physiographic regions by the geological boundary between the precambrian rocks to the north and the Paleozoic rocks to the south. The northern three-quarters of the area is underlain by the Precambrian rocks and forms part of the Precambrian peneplain which slopes gently southward. All of the areas investigated lie within this peneplain.

Relief in the area is moderate being nowhere in excess of 200 feet. The area is drained by the Moira and Black rivers which both flow south emptying into the Bay of Quinte. Numerous low lying area's of swamp exist. Centered in Grimsthorpe township is a large shallow lake called Lingham lake.

HISTORY

The discovery of gold in 1866 at the site that was to become the Richardson Mine near Eldorado, which lies about 4 miles south of Bannockburn on highway no. 62 was the first such discovery in the province. Over the next 30 to 40 years several properties were discovered and on some of them sporadic mining operations were established.

The Bannockburn property is on land which was originally homesteaded by a man names Lloyd in the 1880's. It appears to have been one of the earlier mines discovered. The mine was operated sporadically until 1898.

Besides gold and minor amounts of base metals and iron ore, industrial minerals such as fluorspar and talc are important to the local economy. There is a 200 ton per day talc mine in operation just to the southeast of the town of Madoc.

PREVIOUS WORK

Modern exploration on the Bannockburn property began in 1981 and continued during the years 1984 to 1987 by Mono Gold Mines Ltd. This resulted in the discovery of gold bearing quartz veins in the northeast claim block. During this period Mono Gold carried out geophysical surveys, soil geochemical sampling, surface stripping and sampling, surface mapping and an extensive surface diamond drilling program for a total 41,000 feet.

In November of 1987, Mono Gold Mines entered into a Joint Venture Agreement with Micham Exploration Inc., whereby Micham could earn a 51% interest in the property for expenditures of 2 million dollars.

An underground exploration program was carried out by Micham Exploration Inc. The program consisted of driving a decline to the 100 foot level with drifting and cross-cutting of the quartz veins on the 75 foot level as well as 5,700 feet of underground diamond drilling.

The reserve estimates calculated by Orcan Mineral Associates Ltd. for Micham Exploration Inc. are 248,160 tons with a grade of 0.267 oz. Au./short ton uncut. These reserves are classified as "modified geological ".

REGIONAL GEOLOGY

Madoc Township, in which the Bannockburn property is situated lies astride the contact between the Grenville Province of the Precambrian Shield and the overlying Paleozoic formations. Paleozoic rocks, lying primarily in the southern one-third of the township, consist of Ordovcian limestone with minor conglomerate. Outliers are common in the northern two-thirds. Precambrian units comprise meta-volcanic and meta-sedimentary formations of the Hermon Group and younger felsic and intermediate intrusions, some of batholithic dimensions.

In the Hermon Group, Tudor Formation volcanics consisting of massive, dark green andesite's are the oldest rocks. The Madoc volcanics evidently younger, range in composition from andesite to rhyolite. Both volcanic formations comprise massive lavas, pillowed lavas, vesicular and amygdaloidal lavas, tuffs, and agglomerate. Meta-sedimentary rocks in the Group consist of argillite, conglomerate, quartzite, pelitic and psammitic schists, and marble.

PROPERTY GEOLOGY

The heart of the Bannockburn property is underlain by a northeasterly folded sequence of meta-sedimentary and meta-volcanic rocks of the Hermon Group. Southwest of the village of Bannockburn, a Granitic stock intrudes meta-sedimentary rocks. Mafic, intermediate, felsic, and lamprophyre dykes and sills are also present.

The meta-volcanic units consist of massive, mafic to intermediate greenstones of the Tudor Formation. The meta-sedimentary rocks are both pelitic and calcareous. The meta-pelites comprise pyritic schists, garnet-chlorite-biotite schist, and quartz-sericite schist. The calcareous meta-sediments are primarily impure marble. Thin beds of chloritic tuff occur in the meta-pelite sequences. Meta-volcanics, meta-pelites, and calcareous meta-sedimetary rocks occur extensively enough to be discrete mappable units, whereas schistose and tuffaceous horizons, because they tend to be more discontinuous and relatively thin are not.

All rocks are foliated to some degree, the meta-volcanics and calcareous units moderately and the meta-pelites more strongly. Broad northeasterly striking fold axes are modified locally; tighter folding is more common in the schistose rocks, evidently at or near the contact with the much more massive meta-volcanics. Major faulting in the area is not evident. Northerly striking, moderately to steeply dipping fracture and shear zones, some of which host quartz vein systems of economic interest, are present.

1988 SURFACE EXPLORATION PROGRAM

During the summer of 1988 a surface exploration program was carried out on the townships of Tudor, Grimsthorpe, Elzevir and Madoc. The Bannockburn property is located in the northeast section of Madoc township. The exploration program consisted of: ground examination of airborne geophysical anomalies; prospecting for gold bearing quartz veins and their subsequent trenching and sampling; surface stripping and diamond saw sampling of selected areas on the Bannockburn property.

Airborne Geophysical Anomalies

An airborne geophysical survey was flown in February of 1988 by Airodat LImited and covered the townships of Grimsthorpe, Elzevir and Madoc. The survey comprised of a magnetometer and VLF-EM four frequency electromagnetic survey.

Only the conductors that where deemed significant and where positioned on ground that was not staked at the time were investigated. This left only the conductors in and around Lingham Lake in the central portion of Grimsthorpe Township. Since Lingham Lake is large, shallow and swampy near its edge all the conductors lie underwater or in swampy sections adjacent to the lake. No surface outcrops were present to explain the conductors.

Prospecting

Surface prospecting of areas targeted as having a good potential to contain gold bearing quartz veins in area's of open ground were investigated in Tudor and Madoc Townships.

In Tudor Township an area containing several small (6" to 1' wide) quartz veins was found. These veins were blasted and several grab samples were taken. No gold values were found to exist. The quartz veins are located on the border between concessions six and seven in lots 7, 8, and 9 on the east side of the Moira River in Tudor Township.

In Madoc Township trenching and sampling was carried in the area known as the Lloyd patent on the west side of Hwy. No. 62 located at approximately 2+40 N and 6+70 E. Six grab samples were taken of what appeared to be the best mineralized material. All of the samples returned a value of 0.001 oz's Au. per ton. See figure no. 5. This area lies on claims encompassing the Bannockburn Property.

Stripping and Sampling - Bannockburn Property

Two area's were stripped with a powerful water pump and then mapped and samples taken with a diamond saw. These area's are known as the Northwest and Southeast and are located within the Northeast claim block on the Bannockburn property. See figure no. 4 for their location in respect to the underground workings.

In the northwest an area of approximately 100 feet by 150 was washed and mapped on a scale of 1"=20'. Thirteen diamond saw samples were taken in three locations marked as channel no's 1 to 3 on figure no. 2. Significant gold values were encountered in a northwest trending quartz vein approximately 1 to 2 feet wide. The values are 0.976 oz's Au./ton over a 2 foot width and 1.681 oz's Au./ton over a 1 foot width.

In the southeast an area of approximately 100 feet by 100 feet was washed and mapped on a scale of 1"=20'. Thirty-one diamond saw samples were taken in three channels marked no'4 to 6 in figure no. 3. No significant gold values were obtained in a northwest trending series of quartz veins varying in width from several inches up to 2 feet.

CONCLUSIONS AND RECOMMENDATIONS

The potential for additional reserves in the surrounding areas appears to be poor on any open ground in the area however on previously staked ground significant gold bearing zones do exist.

It is therefore recommended that an examination of the available data on the properties in the area be carried out and that those properties with known reserves and the potential to increase the known reserves at a minimal cost be carried out in the hopes that some agreement on the properties can be arranged. The Bannockburn property contains only marginal at best economic quantities of gold and any further increase in the reserves by acquisitions or joint venture agreements seems to be the logical coarse to take to further expand the reserves at a minimal risk and cost.

CERTIFICATE OF QUALIFICATIONS

I, David R. Bell, hereby certify:

1.) that I am a Consulting Geologist employed by David R. Bell Geological Services Inc., 8 Church Street, St. Catharines, Ontario.

2.) that I am a Graduate of Carlton University, Ottawa, Ontario, holding a Bachelor of Science Degree (B.Sc.) in geology, 1973.

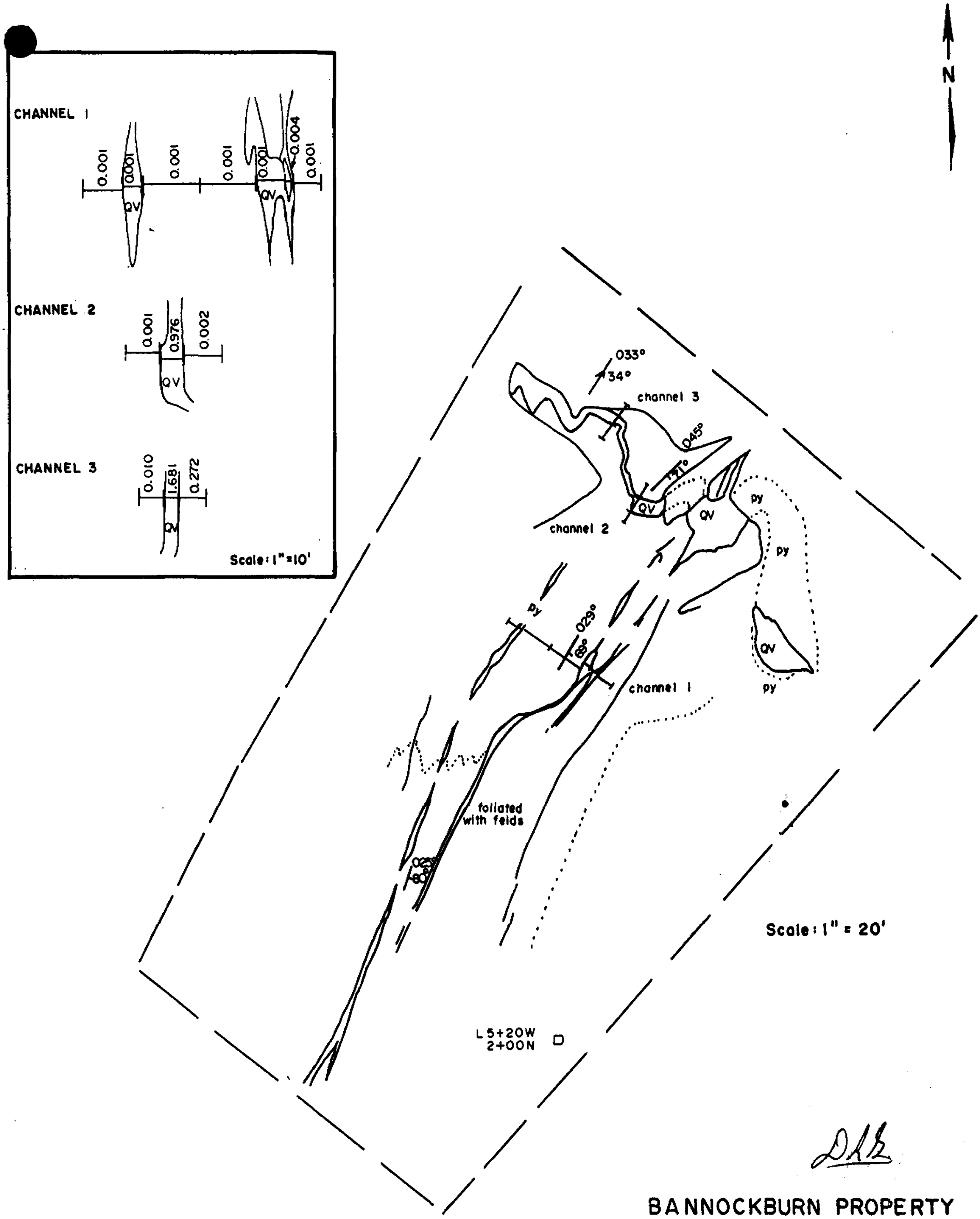
3.) that I have been practising my profession as a geologist continuously since 1973.

4.) that I am a Fellow of the Geological Association of Canada (1981), and a Member of the Canadian Institute of Mining and Metallurgy.

5.) that I am a Director and President of Micham Exploration Inc., and that I am a shareholder of the Company.

David R. Bell, B.Sc., F.G.A.C.

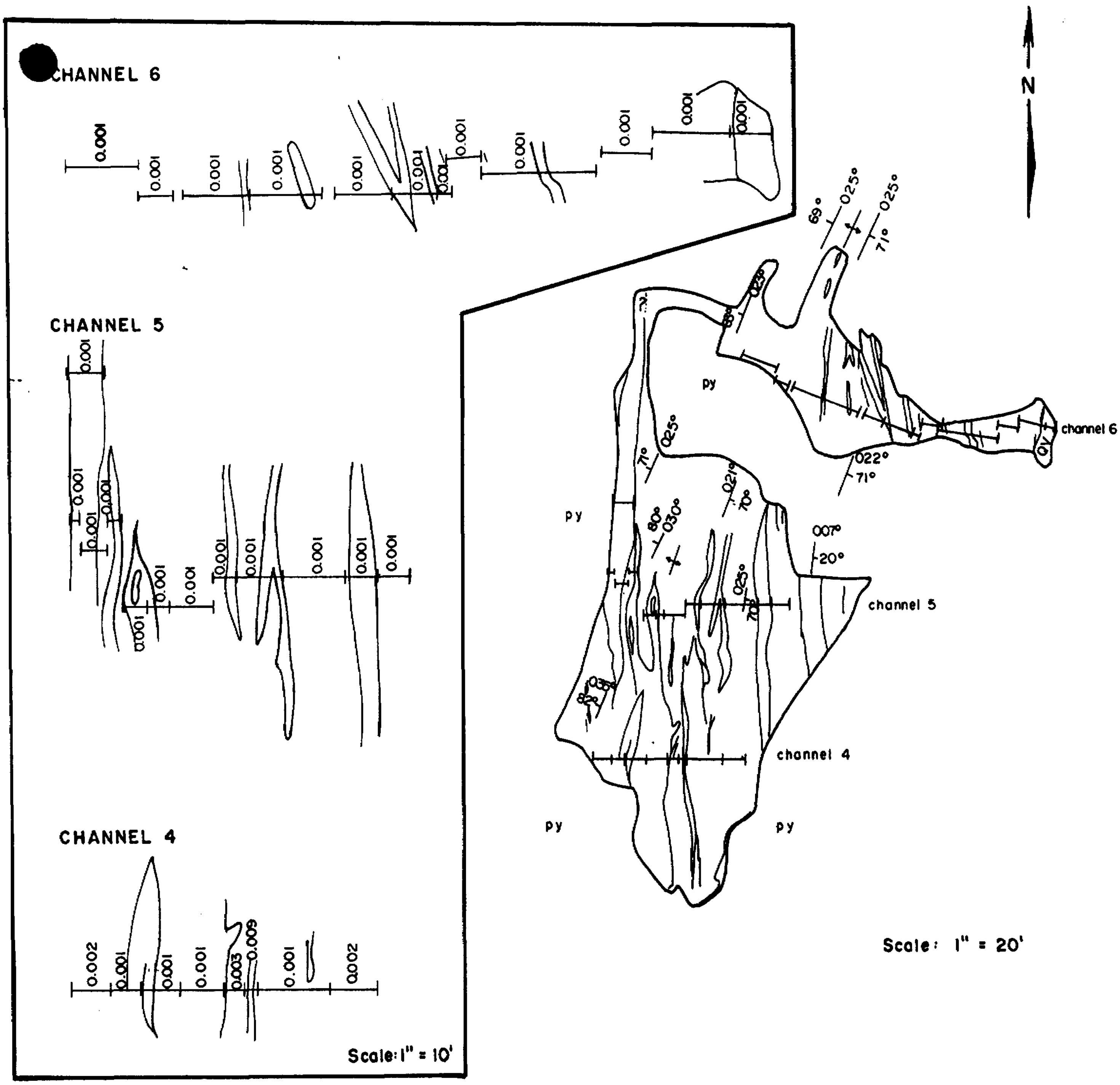
St. Catharines, Ontario
October, 1988



DAB

BANNOCKBURN PROPERTY
NORTHWEST TRENCH

FIGURE 2



DLB

BANNOCKBURN PROPERTY
SOUTHEAST TRENCH

FIGURE 3

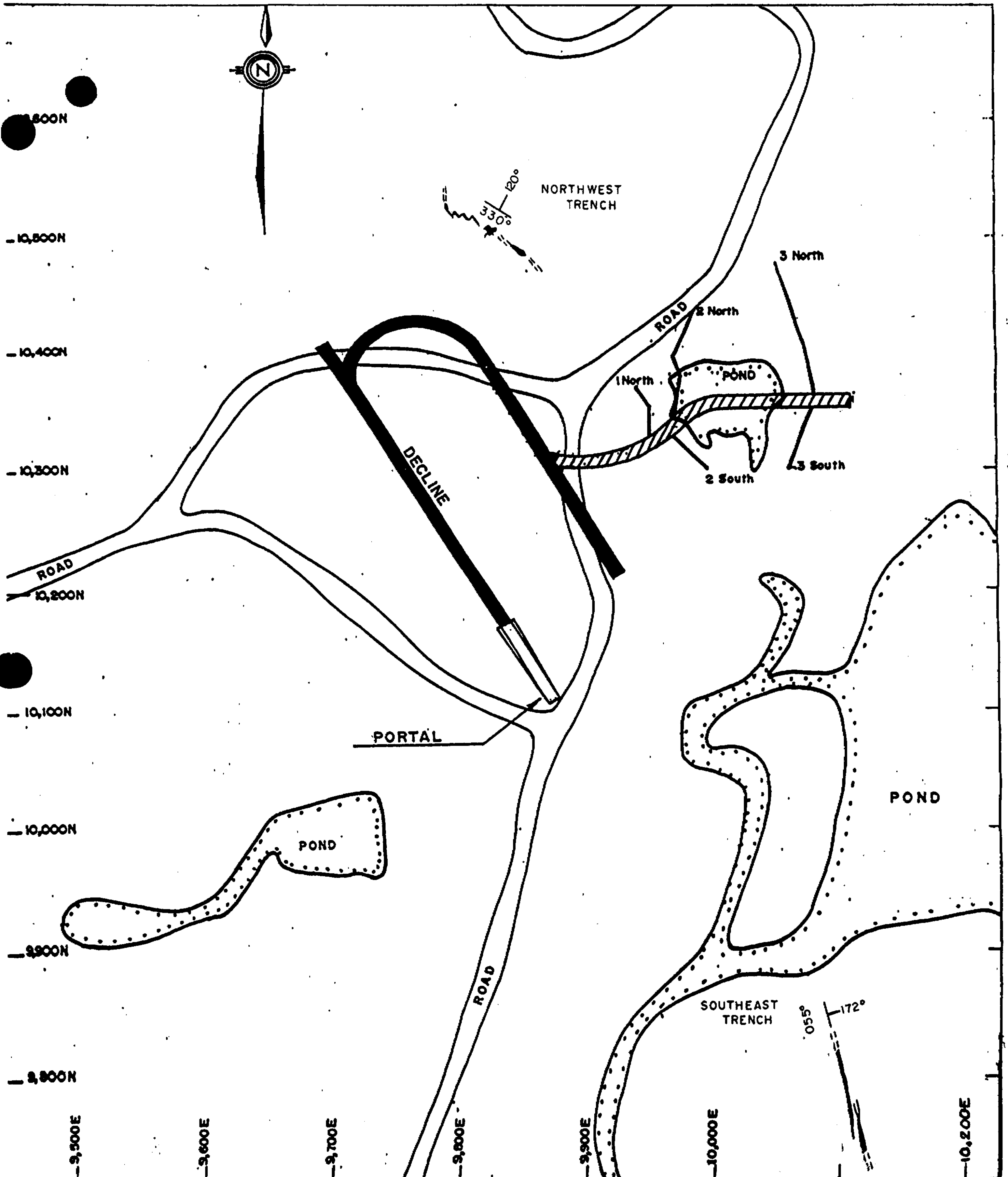
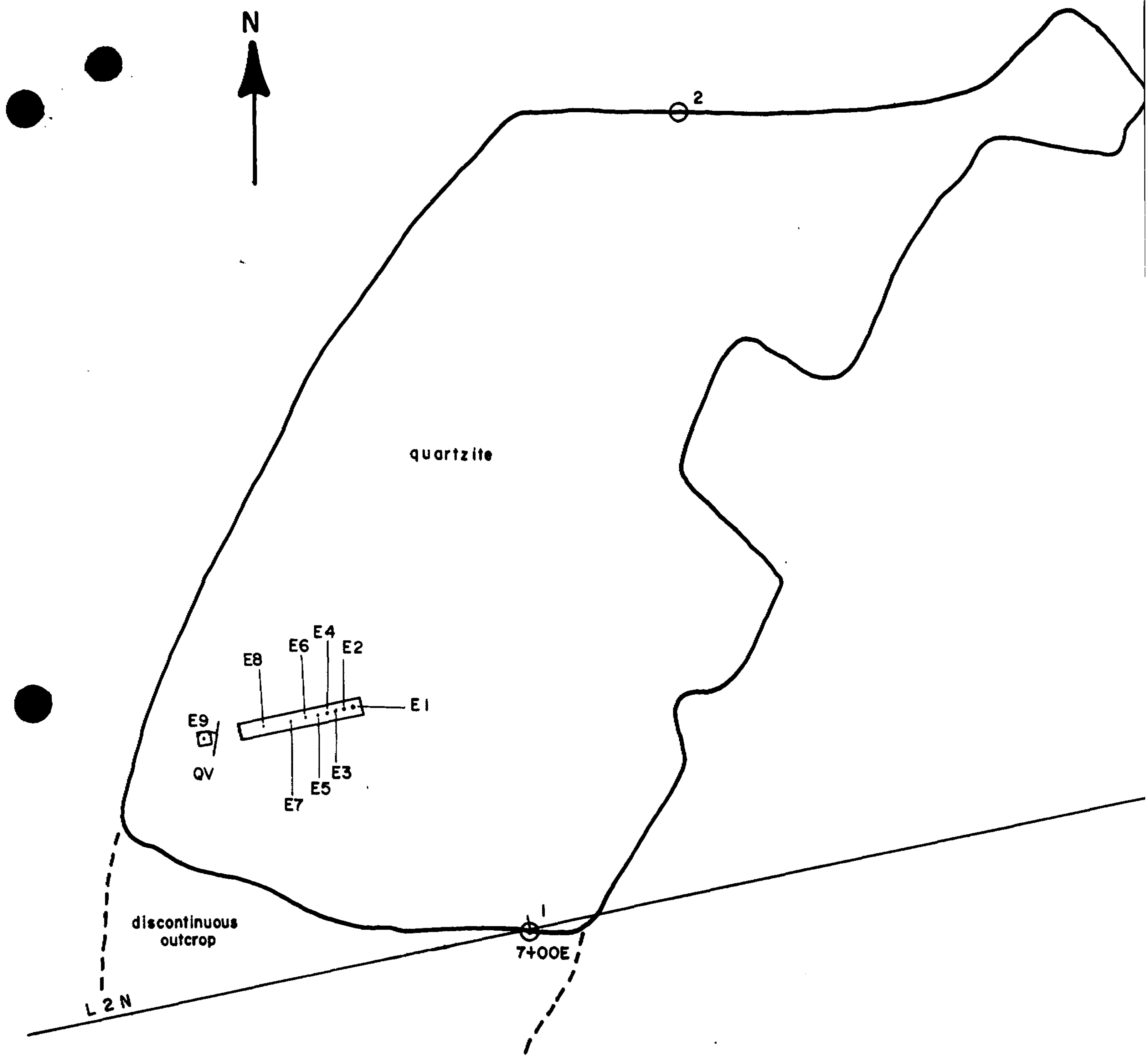


FIGURE 4

MICHAM EXPLORATION INC.
BANNOCKBURN PROPERTY

EXPLORATION PROGRAM
SCALE: 1" = 100' MARCH 1988

D.L.B.

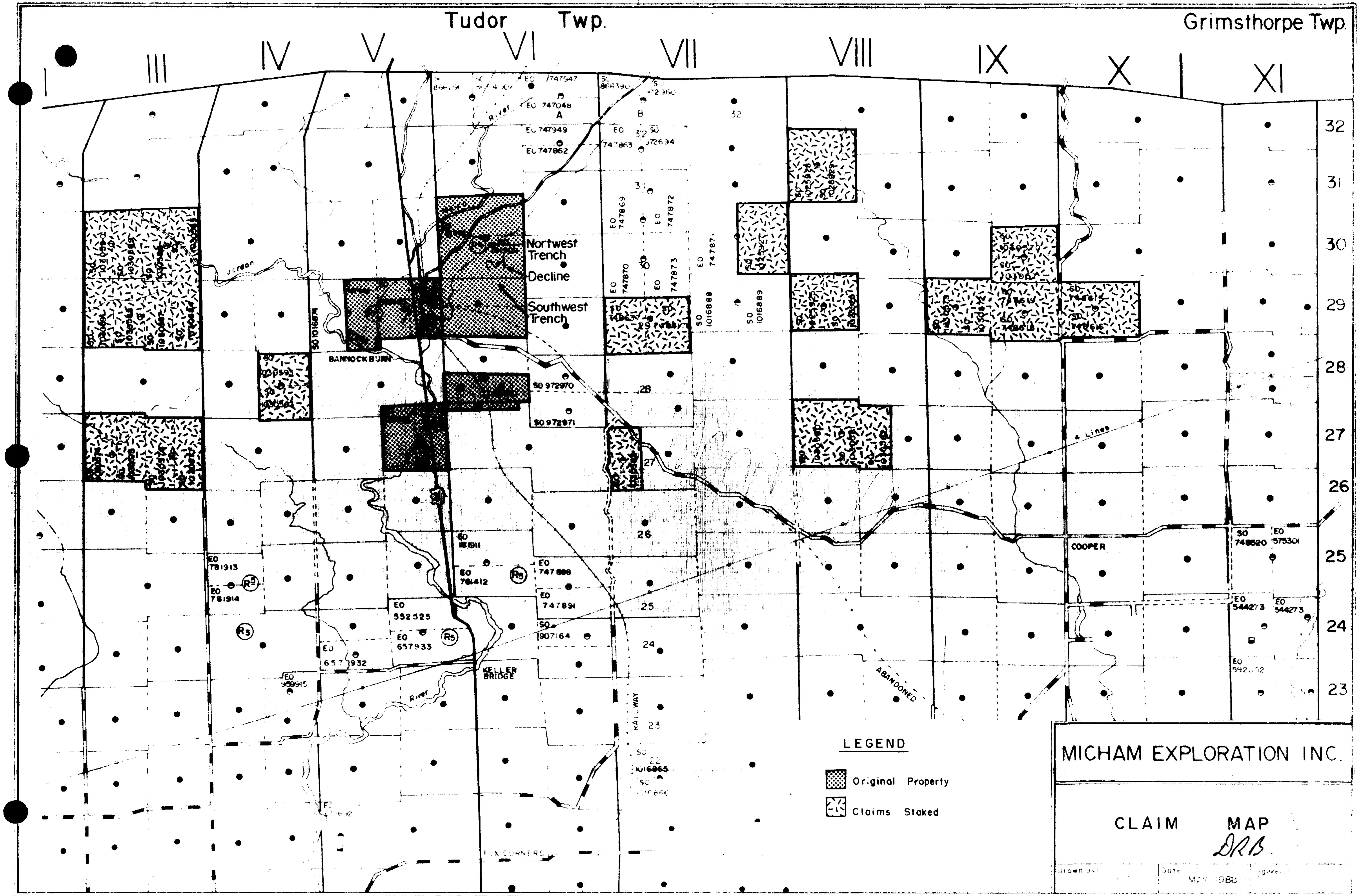


FIELD No.	TAG No.	ASSAY Au
E 1	4518	0.001
E 2	4519	0.001
E 3	4520	0.001
E 4	4521	0.001
E 5	4522	0.001
E 6	4523	0.001
E 7	4524	0.001
E 8	4525	0.001
E 9	4526	0.001

MICHAM EXPLORATION
BANNOCKBURN PROJECT
SAMPLE LOCATION MAP
E TRENCH
SCALE: 1" = 20'

FIGURE 5

DKB


Tudor Twp.
Grimsthorpe Twp.
I
III
IV
V
VI
VII
VIII
IX
X
XI
32
31
30
29
28
27
26
25
24
23
Nortwest Trench
Decline
Southwest Trench
BANNOCKBURN
KELLER BRIDGE
COOPER
RAILWAY
ABANDONED
River
4 Lines
LEGEND
Original Property
Claims Staked
MICHAM EXPLORATION INC.
CLAIM MAP

REPORT DATE 10/11/88 MICHAN EXPLORATION INC PA

MICHAN SAMPLE NUMBER	WDI SAMPLE NUMBER	TOTAL SAMPLE WEIGHT GRAMS	REJECT WEIGHT GRAMS	+150 MESH WEIGHT GRAMS	-150 MESH WEIGHT GRAMS	TOTAL ASSAY WEIGHT GRAMS	+150 MESH ASSAY G/T	-150 MESH ASSAY G/T	CORE ASSAY GRAMS/ TONNE	CORE ASSAY OZ/TON
4301		20718	20036	21.1	209.3	230.4	30.67	0.82	3.55	0.104
4302		19241	18888	23.9	207.6	231.5	53.32	3.50	8.64	0.252
4303		17711	17371	23.5	253.9	277.4	15.19	0.78	2.00	0.058
4304		18279	17869	21.0	208.6	229.6	2.31	0.80	0.94	0.027
4305		21378	20722	29.5	218.5	248.0	32.55	2.91	6.44	0.188
4306		218			199.5	199.5		6.38	6.38	0.186
4307		204			186.4	186.4		1.44	1.44	0.042
4308		84			72.9	72.9		0.18	0.18	0.005
4309		372	98		230.3	230.3		0.03	0.03	0.001

REPORT DATE 10/11/88 MICHAN EXPLORATION INC PA

MICHAN SAMPLE NUMBER	WDI SAMPLE NUMBER	TOTAL SAMPLE WEIGHT GRAMS	REJECT WEIGHT GRAMS	+150 MESH WEIGHT GRAMS	-150 MESH WEIGHT GRAMS	TOTAL ASSAY WEIGHT GRAMS	+150 MESH ASSAY G/T	-150 MESH ASSAY G/T	CORE ASSAY GRAMS/ TONNE	CORE ASSAY OZ/TON
4310		305	48		224.7	224.7		0.35	0.35	0.010

440

Report Date 08/18/88 MICHAM : Drill Programme

MICHAM SAMPLE NUMBER	TOTAL SAMPLE WEIGHT GRAMS	REJECT WEIGHT GRAMS	CORE ASSAY GRAMS/ TONNE	CORE ASSAY OZ/TON
4311	1300	875	0.03	0.001
4312	817	242	0.15	0.004
4313	2071	1663	0.03	0.001
4314	2737	2154	0.03	0.001
4315	5266	4365	0.03	0.001
4316	2079	1409	0.03	0.001
4317	2333	1998	0.03	0.001
4318	1257	865	0.03	0.001
4319	715	431	26.07	0.760
4320	1667	1301	0.08	0.002
4321	1096	688	0.34	0.010
4322	667	351	58.82	1.716
4323	950	473	10.72	0.313
4324	1031	515	0.08	0.002
4325	861	360	0.03	0.001
4326	1494	1072	0.03	0.001
4327	2618	2146	0.03	0.001
4329	1299	897	0.10	0.003
4330	520	196	0.30	0.009
4331	3118	2606	0.03	0.001
4332	3001	2501	0.06	0.002

Report Date 10/11/88 MICHAM : Drill Programme CERTIFICATE OF ANALYSIS

MICHAM SAMPLE NUMBER	TOTAL SAMPLE WEIGHT GRAMS	REJECT WEIGHT GRAMS	CORE ASSAY GRAMS/ TONNE	CORE ASSAY OZ/TON
4311	1300	875	0.03	0.001
4312	817	242	0.15	0.004
4313	2071	1663	0.03	0.001
4314	2737	2154	0.03	0.001
4315	5266	4365	0.03	0.001
4316	2079	1409	0.03	0.001
4317	2333	1998	0.03	0.001
4318	1257	865	0.03	0.001

REPORT DATE 10/11/88 MICHAM DRILL PROGRAMME : CERTIFICATE OF ANALYSIS

MICHAM SAMPLE NUMBER	TOTAL SAMPLE WEIGHT GRAMS	REJECT WEIGHT GRAMS	+150 MESH WEIGHT GRAMS	-150 MESH WEIGHT GRAMS	+150 MESH ASSAY G/T	-150 MESH ASSAY G/T	CORE ASSAY GRAMS/ TONNE	CORE ASSAY OZ/TON
4319	715	431	21.6	229.89	306.71	15.86	40.84	1.191
4322	667	351	30.55	221.25	257.77	28.65	56.45	1.646
4323	930	473	18.64	233.51	9.25	7.82	7.93	0.231

Report Date 10/11/88 MICHAM : Drill Programme CERTIFICATE OF ANALYSIS

MICHAM SAMPLE NUMBER	TOTAL SAMPLE WEIGHT GRAMS	REJECT WEIGHT GRAMS	CORE ASSAY GRAMS/ TONNE	CORE ASSAY OZ/TON
4324	1031	515	0.08	0.002
4325	861	360	0.03	0.001
4326	1494	1072	0.03	0.001
4327	2618	2146	0.03	0.001
4329	1299	897	0.10	0.003
4330	520	196	0.30	0.009
4331	3118	2606	0.03	0.001
4332	3001	2501	0.06	0.002
4333	512	235	0.03	0.001
4334	782	519	0.03	0.001
4335	354	76	0.03	0.001
4336	937	636	0.03	0.001
4337	628	310	0.03	0.001
4338	1242	920	0.03	0.001
4339	774	448	0.03	0.001
4340	1375	869	0.03	0.001
4341	2077	1647	0.03	0.001
4342	1342	1069	0.03	0.001
4343	1081	765	0.03	0.001
4344	509		0.03	0.001
4345	933	684	0.03	0.001
4346	1117	850	0.03	0.001
4347	1406	1122	0.03	0.001
4348	1348	1080	0.03	0.001
4349	2180	1912	0.03	0.001
4350	1955	1698	0.03	0.001
4351	520	217	0.03	0.001
4352	478	290	0.03	0.001
4353	2606	2301	0.03	0.001
4354	1404	1069	0.03	0.001
4355	1386	1162	0.03	0.001
4356	600	369	0.03	0.001

Report Date 10/11/88 MICHAM : Drill Programme CERTIFICATE OF ANALYSIS

MICHAM SAMPLE NUMBER	TOTAL SAMPLE WEIGHT GRAMS	REJECT WEIGHT GRAMS	CORE ASSAY GRAMS/ TONNE	CORE ASSAY OZ/TON
4320	1667	1301	0.08	0.002
4321	1096	688	0.34	0.010

REPORT DATE 10/11/88 MICHAM DRILL PROGRAMME :CERTIFICATE OF ANALYSIS

MICHAM SAMPLE NUMBER	TOTAL SAMPLE WEIGHT GRAMS	REJECT WEIGHT GRAMS	CORE ASSAY GRAMS\ TONNE	CORE ASSAY OZ\ TON
4301	20718	20036	3.55	0.104
4302	19241	18888	8.64	0.252
4303	17711	17371	2.00	0.058
4304	18279	17869	0.94	0.027
4305	21378	20722	6.44	0.188
4306	218		6.38	0.186
4307	204		1.44	0.042
4308	84		0.18	0.005
4309	372	98	0.03	0.001
4310	305	48	0.35	0.010
4311	1300	875	0.03	0.001
4312	817	242	0.15	0.004
4313	2071	1663	0.03	0.001
4314	2737	2154	0.03	0.001
4315	5266	4365	0.03	0.001
4316	2079	1409	0.03	0.001
4317	2333	1998	0.03	0.001
4318	1257	865	0.03	0.001
4319	715	431	40.84	1.191
4320	1667	1301	0.08	0.002
4321	1096	688	0.34	0.010
4322	667	351	56.45	1.646
4323	930	473	7.93	0.231
4324	1031	515	0.08	0.002
4325	861	360	0.03	0.001
4326	1494	1072	0.03	0.001
4327	2618	2146	0.03	0.001
4329	1299	897	0.10	0.003
4330	520	196	0.30	0.009
4331	3118	2606	0.03	0.001
4332	3001	2501	0.06	0.002
4333	512	235	0.03	0.001
4334	782	519	0.03	0.001
4335	354	76	0.03	0.001
4336	937	636	0.03	0.001
4337	628	310	0.03	0.001
4338	1242	920	0.03	0.001
4339	774	448	0.03	0.001
4340	1375	869	0.03	0.001
4341	2077	1647	0.03	0.001
4342	1342	1069	0.03	0.001
4343	1081	765	0.03	0.001
4344	509		0.03	0.001
4345	933	684	0.03	0.001
4346	1117	850	0.03	0.001
4347	1406	1122	0.03	0.001
4348	1348	1080	0.03	0.001
4349	2180	1912	0.03	0.001
4350	1955	1698	0.03	0.001
4351	520	217	0.03	0.001
4352	478	290	0.03	0.001

CERTIFIED BY F.VERSCHAEVE.MANAGER ANALYTICAL SERVICES

Report Date 08/05/88 MICHAM : Drill Programme

MICHAM SAMPLE NUMBER	TOTAL SAMPLE WEIGHT GRAMS	REJECT WEIGHT GRAMS	CORE ASSAY GRAMS/ TONNE	* CORE ASSAY OZ/TON
4501	1164	774	0.07	0.002
4502	964	605	0.03	0.001
4503	655	325	0.03	0.001
4504	907	574	0.03	0.001
4505	1134	765	0.03	0.001
4506	1164	839	0.03	0.001
4507	908	608	0.03	0.001

Report Date 08/10/88 MICHAM : Drill Programme

MICHAM SAMPLE NUMBER	TOTAL SAMPLE WEIGHT GRAMS	REJECT WEIGHT GRAMS	CORE ASSAY GRAMS/ TONNE	CORE ASSAY OZ/TON
4508	1591	1203	0.03	0.001
4509	1507	1109	0.03	0.001
4510	1246	893	0.03	0.001
4511	1847	1439	0.03	0.001
4512	2376	1975	0.03	0.001
4513	680	392	0.03	0.001
4514	1781	1076	0.03	0.001
4515	1016	660	0.03	0.001
4516	996	689	0.03	0.001
4517	1454	1083	0.03	0.001

WITTECK WDI DEVELOPMENT INC.
2840 South Sheridan Way, Mississauga, Ontario, Canada L5J 2M4
Telephone: (416) 823-7381 Telex: 06-982328

Report Date 08/11/88 MICHAM : Drill Programme

MICHAM SAMPLE NUMBER	TOTAL SAMPLE WEIGHT GRAMS	REJECT WEIGHT GRAMS	CORE ASSAY GRAMS/ TONNE	CORE ASSAY OZ/TON
4518	783	468	0.03	0.001
4519	740	441	0.03	0.001
4520	768	450	0.03	0.001
4521	658	346	0.03	0.001
4522	681	381	0.03	0.001
4523	777	450	0.03	0.001
4524	634	337	0.03	0.001
4525	944	582	0.03	0.001
4526	869	535	0.03	0.001

Certified by: Andrew Smith
Fernand Verschaeve
Manager, Analytical Services

ORCAN MINERAL ASSOCIATES LTD.
CONSULTING ENGINEERS

SUITE 1417 - 409 GRANVILLE STREET
VANCOUVER, CANADA V6C 1T2
TELEPHONE (604) 662-3722

Micham Exploration Inc.

Preliminary Feasibility Study of the BANNOCKBURN PROJECT Madoc, Ontario

July, 1988

Orcan Mineral Associates Ltd.

Vancouver, Canada

Table of Contents

Table of Contents (Cont'd.)

Table of Contents (Cont'd.)

I.

SUMMARY

I. SUMMARY

The Bannockburn property is located in southern Ontario 18 kilometres north of the town of Madoc and some 240 kilometres by road northeast of Toronto. It is situated in an old mining area which has produced iron, pyrite, some base metals, minor gold, talc, fluorspar, marble and slate.

Modern exploration of the Bannockburn property began in 1981 and continued during the years 1984 to 1988. Much of the work comprised surface diamond drilling; the last phase, in early 1988, consisted of a decline to the 75 ft. Level (1st Level) and drifting on three veins.

The Madoc area lies astride the contact between the Grenville Province of the Precambrian Shield and overlying Paleozoic formations. The property is underlain by a northeasterly fold sequence of meta-sedimentary and meta-volcanic rocks and, in the southwest, by a granitic stock. In an area termed the Northeast Block, a complex system of quartz veining occupies an area 1,200 feet north-south by 400-500 feet east-west. Much of the property exploration has been concentrated in this block, particularly the southern half (the Core Area). The primary vein system, the one which is gold-bearing, strikes northerly and dips 35 to 55 degrees easterly. It consists of four to ten narrow quartz veins (a few inches to rarely six feet) containing significant gold values. Underground work in the Core Area shows the veins to be very irregular but exhibiting reasonable persistence along strike and dip.

Reserves have been calculated based mainly on diamond drilling intercepts. Limited checking with chip and muck sampling of the underground workings strongly suggests that an upgrading of drill-calculated reserve grades will evolve, particularly at the lower grade values. As well, in the northern part of the Northeast Block (the North Area), assaying problems related to the nugget characteristic of the gold have resulted in a number of vein intersections apparently being severely undervalued. The calculated reserves, which contain no compensation for these anomalous conditions, and which are classified mostly as 'Modified Geological' are: 248,160 tons @ 0.186 oz gold per ton (cut), 0.267 oz gold per ton (uncut). The property potential is estimated to be a minimum of 500,000 tons.

A conceptual mining plan has been developed for the property based on a 200 tons per day operation. Development, at least down to the 5th Level (475 ft.), is by ramp with level intervals at 100 ft. vertical spacing. Track development and mining is to be used on the levels with main haulage by trucks up the ramp. Stoping will be by shrinkage and open stopes.

Metallurgical testwork indicates gold recovery in the order of 98 percent for a cyanide leach process. It is proposed that tailings be transported to an impoundment area in a 'dry' form.

No abnormal environmental problems are anticipated.

The estimated capital cost to place the property into production is $13,720,000, excluding working capital. Operating costs are calculated to be $90.40 per ton of ore milled. Sensitivity analyses indicate the positive effects of equity financing, increased grade, decreased capital and operating costs, and increased gold price.

It is concluded that, at current reserve and costs parameters, the Bannockburn property is not economically viable, but that it is an excellent exploration prospect. It is obvious from the overall study that it could be viable with some positive changes to some of the parameters. Some of these changes, particularly an increase in grade, are expected to materialize as further underground exploration is conducted. Consequently, it is recommended that a major exploration program be undertaken on the property. The program, to consist mostly of underground work (because, for the most part, further surface drilling will not enhance the known vein areas), is estimated to cost $4,500,000. It should provide most of the data necessary to proceed with a feasibility study.

2.

INTRODUCTION

2. INTRODUCTION

The Bannockburn Project encompasses the old Bannockburn gold mine and other mineral rights holdings in Madoc Township, eastern Ontario. The modern period of exploration began in 1981 when Mono Gold Mines Inc. acquired these rights and initiated surface exploration. Mono carried out work in 1981 and from 1984 to 1987. In 1984, they discovered new gold-bearing zones in the 'Northeast Block', where much of the subsequent work has been concentrated.

In October 1987, Micham Exploration Inc. entered into a joint venture with Mono Gold Mines to further explore and develop the property. Micham Exploration, as operator of the joint venture, conducted the most recent work which comprised driving a decline, exploration drifting and underground drilling in the Northeast Block.

2.1 SCOPE OF STUDY

The purpose of this study is to:

- Evaluate the Bannockburn gold property from current information.
- Provide conceptual designs for mining, concentrating and support facilities.
- Develop preliminary capital and operating costs, and potential profitabilities.
- Set out further surface and underground exploration.

The study includes the following:

- Review of geology and mineralization
- Estimation of reserves
- Review of metallurgy and preliminary design of flow sheet
- Conceptual mine design
- Determination of requirements for support services and auxiliary facilities
- Statement of environmental considerations.
- Preliminary capital and operating cost estimates
- Estimates of net present values for the project

The following parameters and units are used in this report:

- Imperial units, except for temperature and distances which are in metric units (degrees Celsius, kilometres)

- Canadian dollars except for metal prices which are quoted in United States dollars; conversion factor applied is U.S.$1.00 = Cdn.$1.20

- A date of June 1, 1988 for calculating purposes and data availability

- A plant operating capacity of 200 short tons per day (from Section 6.2)

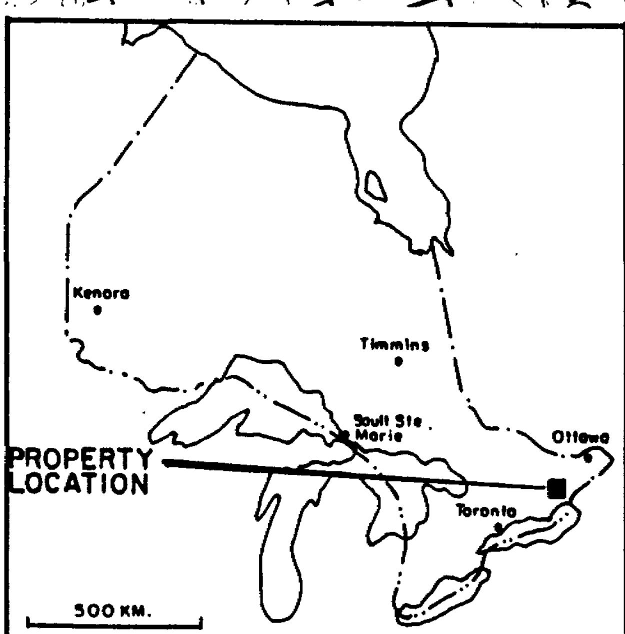
- Economic estimates that are 'before taxes and write-offs'

2.2 LOCATION AND ACCESS

The Bannockburn property is located in the northern part of Madoc Township, Hastings County, eastern Ontario (Figure 1). It is approximately 240 kilometres by road northeast of Toronto. The property is readily accessible from Provincial Highway 62, linking the towns of Madoc and Bancroft and connecting with Highway 7 (Trans Canada Highway) at Madoc. The property encompasses the small unincorporated settlement of Bannockburn, which is 18 kilometres north of Madoc.

Access to the office facilities and decline is by a short gravel road east from Highway 62 (Figure 2). The area to the northeast can be reached by means of an abandoned railroad grade and a road on the east side of Moira River. The portion of the property lying west of the Highway can be accessed by a private company road which crosses the Moira River by means of a wooden bridge. Bush trails and roads, which have been upgraded during recent periods of exploration, provide access by truck or foot to the area of the old Bannockburn mine and further west.

Considering the low relief, gentle slopes and presently available roads and trails, any part of the property can be gained with quite limited physical effort. The only problems of consequence are the Moira River and some ponds and swampy areas.



ORCAN MINERAL ASSOCIATES LTD. CONSULTANTS
VANCOUVER, CANADA

MICHAM EXPLORATION INC.

BANNOCKBURN PROPERTY

LOCATION MAP

MADOC TWP., ONTARIO

N.T.S. 31 C

SCALE 1:250,000	JUNE 1988	FIG. 1

CHONG

Services and supplies for mining operations are readily available from numerous towns and cities in eastern Ontario.

2.2.1 Physiography

Relief on the property is subdued, extremes in elevation being 850 feet and 900 feet. There is moderate rock outcrop exposure separated by many small swamps. Overburden is thin; it comprises well developed podzol soils in the better drained areas and organic rich soils in the poorly drained areas.

The vegetation consists of mixed hardwoods and softwoods, including poplar, maple, oak, white ash, ironwood, spruce, fir and hemlock. Open park-like land prevails in some areas where attempts at farming have long since been abandoned.

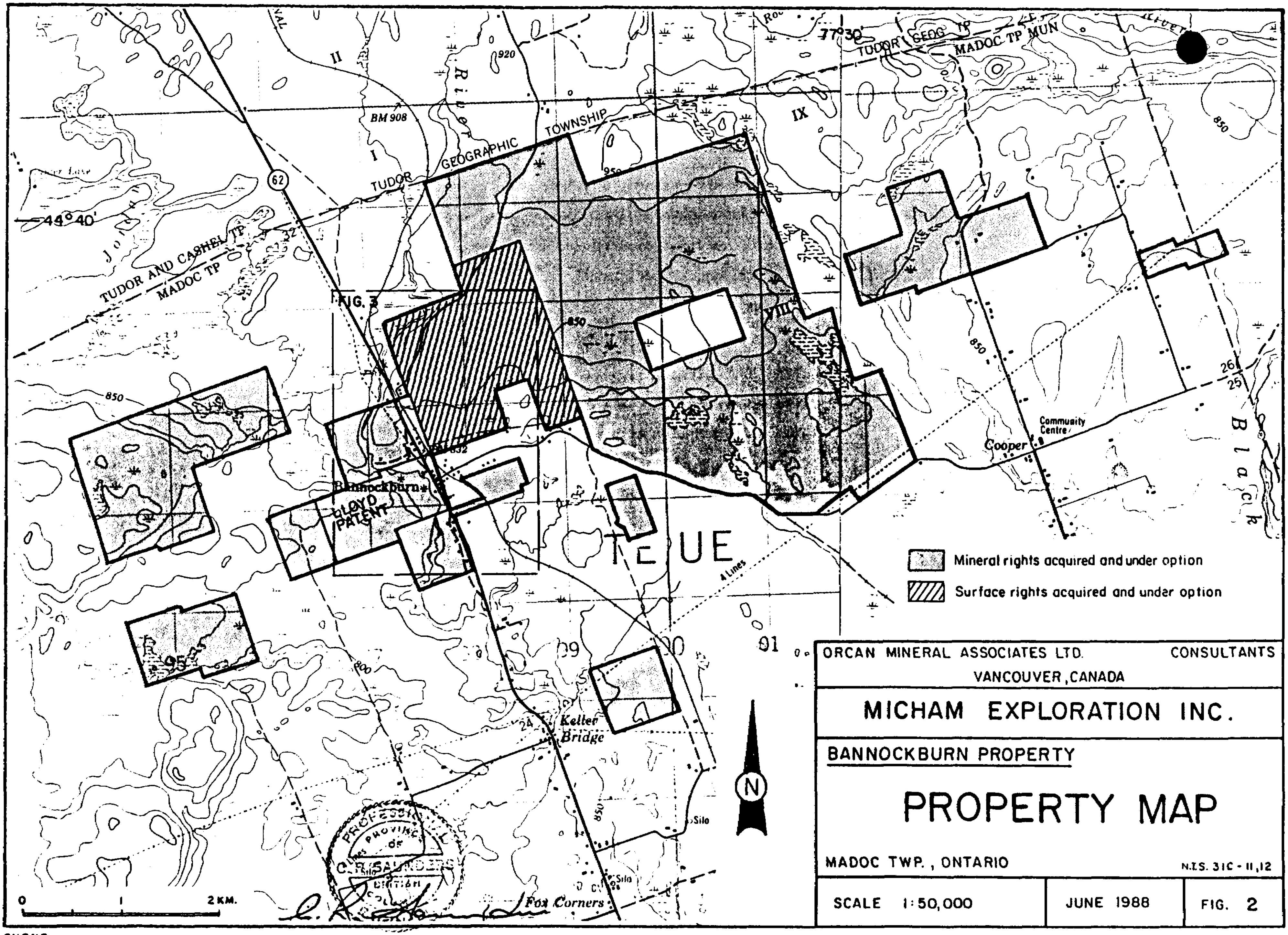
2.2.2 Climate

Climate in the Madoc area is typical of southern Ontario; summers tend to be warm and humid, and winters dry and cool. Temperatures range from highs of 25° to $35^{\circ}C$ in the summer and to lows of -10° to -15° C in winter. Rare extremes are to $+40^{\circ}$ and $-40^{\circ}C$.

Precipitation is in the order of 90 cm per year of which about one-quarter is snowfall. Thunderstorms are common in the summer months.

2.3 PROPERTY

The Bannockburn property comprises all of Lot 28 in Concession V (Lloyd Patent), 69 located mineral claims, and an option on certain other mineral rights under the Plantt Agreement. As shown on Figure 2, all claims are not contiguous. They are enumerated as follows:



TUDOR GEOG TP
MADOC TP MUN
77°30'
TUDOR GEOGRAPHIC TOWNSHIP
TUDOR AND CASHEL TP
MADOC TP
44°40'
BM 908
920
950
850
800
62
IX
VIII
II
I
FIG. 3
LLOYD PATENT
Bannockburn
Cooper
Community Centre
Keller Bridge
Fox Corners
Silo
Mineral rights acquired and under option
Surface rights acquired and under option
ORCAN MINERAL ASSOCIATES LTD.
CONSULTANTS
VANCOUVER, CANADA
MICHAM EXPLORATION INC.
BANNOCKBURN PROPERTY
PROPERTY MAP
MADOC TWP., ONTARIO
N.T.S. 31C - 11,12
SCALE 1:50,000
JUNE 1988
FIG. 2
0
1
2 KM.
N

CHONG

Claim Name	Lot Number	Concession
EO 572 483	27	V
EO 572 484	29	V
EO 572 485	29	V
EO 592 199	27	V
EO 652 301 (+ surface rts.)	29	VI
EO 652 302 (+ surface rts.)	29	VI
EO 740 470	28	VI
EO 740 472	27	VI
EO 747 862	31	VI
EO 747 863	31	VII
EO 747 869	31	VII
EO 747 870	30	VII
EO 747 871	30	VII
EO 747 872	31	VII
EO 747 873	30	VII
EO 747 941	29	V
EO 747 947	32	VI
EO 747 948	32	VI
EO 747 949	31	VI
EO 781 909 (+ surface rts.)	30	VI
EO 781 910 (+ sufrace rts.)	30	VI
SO 748 615	29	X
SO 748 616	29	X
SO 748 617	29	VII
SO 748 618	29	IX
SO 748 619	29	IX
SO 748 637	29	VII
SO 866 390	32	VII
SO 972 694	31	VII
SO 972 960	32	VII
SO 972 970	28	VI
SO 972 971	28	VI
SO 1025 925	29	VIII
SO 1025 926	29	VIII
SO 1025 927	30	VII
SO 1025 928	31	VIII
SO 1025 929	31	VIII
SO 1030 570	30	IX
SO 1030 571	30	IX
SO 1030 572	29	IX
SO 1030 573	29	IX
SO 1030 574	27	III
SO 1030 575	27	III
SO 1030 576	27	III
SO 1030 577	27	III
SO 1030 580	27	VIII
SO 1030 581	27	VIII

Claim Name	Lot Number	Concession
SO 1030 582	27	VIII
SO 1030 583	27	VII
SO 1030 584	29	III
SO 1030 585	30	III
SO 1030 586	30	III
SO 1030 587	29	III
SO 1030 588	29	III
SO 1030 589	30	III
SO 1030 590	30	III
SO 1030 591	29	III
SO 1030 592	28	IV
SO 1030 593	28	IV
SO 1072 246	30	VIII
SO 1072 247	30	VIII
SO 1072 248	28	XI
SO 1072 249	28	XI
SO 1072 256	24	VI
SO 1072 266	30	IV
SO 1072 267	30	IV
SO 1072 276	24	VI
SO 1072 277	30	IV
SO 1072 278	30	IV

The surface rights on Lots 29 and 30 in the west half of Concession VI have been acquired. An option to secure surface rights on Lots 29, 30 and 31 in the east half of Concession VI has been secured from 586108 Ontario Inc. (the Plantt Agreement).

3.

HISTORY

3. HISTORY

3.1 MADOC AREA

Mining has been carried out in the Madoc area for at least 150 years. Early activity was related to mining iron ore, both hematite and magnetite, with several mines operating from 1837 to about 1910.

The discovery of gold in 1866 on the Richardson farm near Eldorado, some three kilometres south of Bannockburn, sparked a gold rush in the area. In the years that followed, numerous prospects were explored and several were brought to production. However, it appears that none were commercially viable, probably due to combinations of grade, costs and the inferior recovery techniques that were then available. The Bannockburn Mine appears to have been one of the earlier operations established.

A number of other mines in the area were worked for base metals and industrial minerals as well as the mines operating for iron ore and precious metals. Pyrite mining was active until 1919, with fluorspar mining important from 1905 to 1920 and again from 1940 to 1951. Other industrial minerals of importance are talc, which was first discovered in the Madoc area in 1896, and which has been produced continuously to the present time since then, and marble and slate as well as other stones which have been quarried mainly for building purposes.

At the Bannockburn Mine, about 1894, four shallow shafts (maximum 70 feet deep) were sunk in addition to stripping and trenching. Drifting appears to have been minimal, 17 feet being mentioned for one of the shafts. A ten-stamp mill was in operation at the same time. In 1897, one of the shafts was deepened to 75 feet, and another 35-foot shaft was sunk. Although records are incomplete, approximately 3.5 ounces of gold were produced from an unknown tonnage of 'ore'.

There was little further work in the Bannockburn Mine area until 1965, when a Mr. Roland Belanger undertook exploration aimed at evaluating the old showings by diamond drilling and sampling. The work included 5,100 feet of core drilling in 12 holes, as well as the sampling of old trenches, excavating of new trenches, and dewatering and sampling of an old shaft. Although the results of this work are reported to have been encouraging, they cannot be confirmed because all reports, assays and drill logs were destroyed in a fire at Belanger's home.

3.2 BANNOCKBURN PROPERTY

In 1981, an initial exploration program was conducted for Mono Gold Mines Inc. which, in essence, covered only the main showing area around the old shaft and between that point and the Moira River. The work completed included a cut-line grid, geological mapping, electromagnetic and magnetic geophysical surveys, a limited amount of stripping and trenching, and a preliminary drill program involving 1,725 feet in 11 short drill holes. The main 75-foot shaft was partially dewatered and the mine structure was sampled.

The cut-line grid was extended in early 1984 to encompass the total area of the property as it was then constituted. The geophysical survey was run over the renovated and extended grid. The survey, consisting of a VLF-EM survey and a magnetometer survey, outlined several EM conductors and areas of anomalous magnetic readings. In September 1984, a program of geological mapping, prospecting and sampling of quartz vein outcrops was carried out, using the cut-line grid for control, to check out the geophysical anomalies. It was this work which resulted in the discovery of gold-bearing veins in the 'Northeast Block'. A specimen sample from the Discovery Vein assayed 0.966 oz gold per ton.

In February of 1985, 2,027 feet of diamond drilling were completed in eight holes on the Northeast Block. This drilling confirmed the presence of a gold-bearing vein system at shallow depths along a strike length of approximately 150 feet (the Discovery or 'D' Zone).

Another exploration program was partially completed by May 1985, being temporarily suspended to allow for geological mapping and the establishment of a detailed drilling grid during June 1985. The program resumed in June and was completed in July, 1985. This program outlined significant gold-bearing quartz veining over a strike length of more than 500 feet and to a vertical depth of approximately 240 feet. This drilling totalled 3,236 feet in 12 holes, and essentially extended the Discovery Zone northward.

From late August to late September, 1985, an additional 4,480 feet were drilled in 14 holes. At this point, an initial Inferred Reserve was calculated: 98,750 tons at 0.34 oz gold per ton.

In September, 1985, a geochemical survey of the entire Bannockburn property commenced; it was completed in December, 1985. A number of geochemical anomalies were identified.

Further diamond drilling was done during the period of late October to mid December, 1985. Fifteen holes totalling 7,038 feet were completed.

In 1986, exploration and testing of the quartz vein systems continued with three phases of drilling carried out in January, February, and July to October. A total of 18,589 feet was drilled in 52 holes, with several drill holes from previous programs deepened as well. The 1986 holes were designed to infill all the cross sections in the Discovery Zone and to test the quartz veins to the north (the 'H' Zone). From results of this relatively detailed drilling, preliminary reserve calculations were made over the known mineralized zones to a vertical depth of not more than 350 feet. Drill indicated and inferred reserves of 249,960 tons with a grade of 0.446 oz gold per ton over a minimum four-foot width were estimated for the 'D' Zone and the 'H' Zone. In this report, the 'D' or Discovery Zone and the 'H' Zone are subsequently referred to as the Core Area and the North Area respectively.

During February and March, 1987, a diamond drilling program comprising 10,102 feet in 32 holes was completed on the 'infill' area between the 'D' and 'H' zones. Drill Indicated and Inferred reserves were estimated to be 372,154 tons at a grade of 0.395 oz gold per ton.

In May and June 1987, a stripping program was carried out over the Discovery Vein and was extended some 400 feet south and 700 feet north. Some of the stripped area around the Discovery Zone was ground sluiced and the vein channel sampled

(Figure 6). The vein was drilled and blasted over 85 feet of strike length, and a bulk sample collected and sent for metallurgical testing.

Further stripping on the Northeast Block was done in September and October, 1987 resulting in the discovery of visible gold occurrences in several of the quartz veins.

In the first four months of 1988, a 11 ft x 15 ft decline was driven to the 75-foot or 1st Level, five drifts were driven into three veins, and an underground diamond drilling program was completed. The length of the decline is 880 feet plus an additional 100 feet equivalent for drill stations, etc. The drifts, at a nominal 7 ft x 6 ft cross-section, totalled 320 feet. The diamond drilling was done from three drill bays. It consisted of 5,700 feet in 35 holes.

4.

GEOLOGY

4. GEOLOGY

4.1 REGIONAL GEOLOGICAL SETTING

Madoc Township, in which the Bannockburn property is situated, lies astride the contact between the Grenville Province of the Precambrian Canadian Shield and overlying Paleozoic formations. Paleozoic rocks, lying primarily in the southern one-third of the township, consist of Ordovician limestone with minor conglomerate. Outliers are common in the northern two-thirds. Precambrian units comprise meta-volcanic and meta-sedimentary formations of the Hermon Group and younger felsic and intermediate intrusions, some of batholithic dimensions.

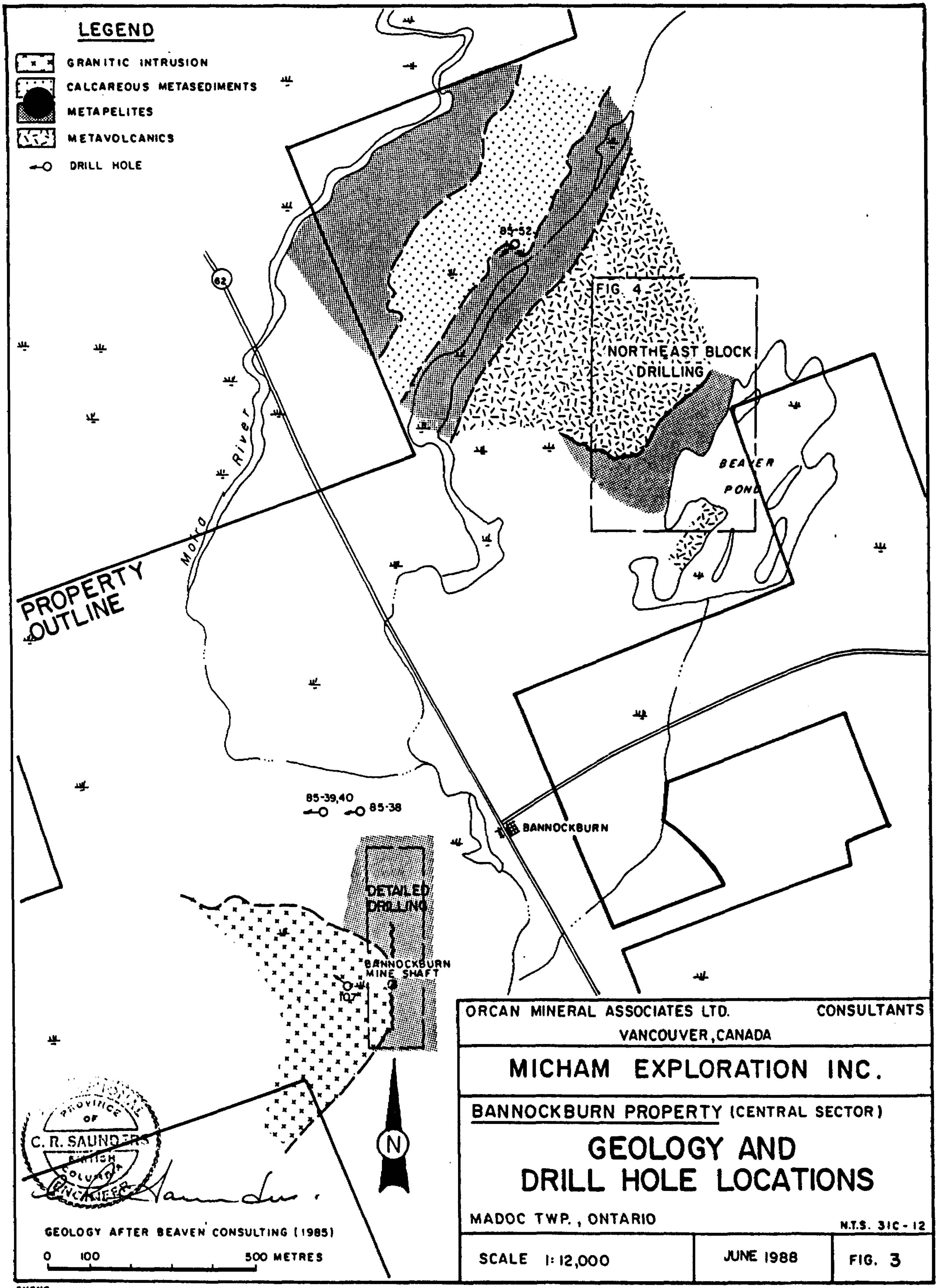
In the Hermon Group, Tudor Formation volcanics consisting of massive, dark green andesite's are the oldest rocks. The Madoc volcanics, evidently younger, range in composition from andesite to rhyolite. Both volcanic formations comprise massive lavas, pillowed lavas, vesicular and amygdaloidal lavas, tuff, and agglomerate. Meta-sedimentary rocks in the Group consist of argillite, conglomerate, quartzite, pelitic and psammitic schists, and marble.

4.2 PROPERTY GEOLOGY (CENTRAL SECTOR)

The heart of the Bannockburn property is underlain by a northeasterly folded sequence of meta-sedimentary and meta-volcanic rocks of the Hermon Group (Figure 3). Southwest of the village of Bannockburn, a granitic stock (the Cawley Creek Syenite) intrudes meta-sedimentary rocks. Mafic, intermediate, felsic, and lamprophyre dykes and sills are also present.

The meta-volcanic units consist of massive, mafic to intermediate greenstones of the Tudor Formation. The meta-sedimentary rocks are both pelitic and calcareous. The meta-pelites comprise pyritic schists, garnet-chlorite-biotite schist, and quartz-sericite schist. The calcareous meta-sediments are primarily impure marble. Thin beds of chloritic tuff occur in the meta-pelite sequences. Meta-volcanics, meta-pelites, and calcareous meta-sedimentary rocks occur extensively enough to be discrete mappable units, whereas schistose and tuffaceous horizons, because they tend to be more discontinuous and relatively thin, are not.

All rocks are foliated to some degree, the meta-volcanic and calcareous units moderately and the meta-pelites more strongly. Broad northeasterly striking fold axes are modified locally; tighter folding is more common in the schistose rocks, evidently at or near the contact with the much more massive meta-volcanics. Major faulting in the area is not evident. Northerly striking, moderately to steeply dipping fracture and shear zones, some of which host quartz vein systems of economic interest, are present.


LEGEND
GRANITIC INTRUSION
CALCAREOUS METASEDIMENTS
METAPELITES
METAVOLCANICS
DRILL HOLE
62
85-52
FIG. 4
NORTHEAST BLOCK DRILLING
BEAVER POND
Moira River
PROPERTY OUTLINE
85-39,40
85-38
BANNOCKBURN
DETAILED DRILLING
BANNOCKBURN MINE SHAFT
107
N
C. R. SAUNDERS
GEOLOGY AFTER BEAVEN CONSULTING (1985)
0 100 500 METRES
CHONG
ORCAN MINERAL ASSOCIATES LTD. CONSULTANTS
VANCOUVER, CANADA
MICHAM EXPLORATION INC.
BANNOCKBURN PROPERTY (CENTRAL SECTOR)
GEOLOGY AND DRILL HOLE LOCATIONS
MADOC TWP., ONTARIO
N.T.S. 31C-12
SCALE 1:12,000
JUNE 1988
FIG. 3

5.

MINERAL DEPOSITS

5. MINERAL DEPOSITS

Gold mineralization associated with quartz veins is presently known to occur at two localities on the Bannockburn property, in the old Bannockburn mine area and, one mile to the northeast, in the Northeast Block.

5.1 BANNOCKBURN MINE

At the Bannockburn mine, free gold associated with pyrite and carbonate occurs in a quartz vein zone comprising veins up to two feet in width, quartz stringers, and silicification. The zone occupies a northerly striking shear zone which dips vertically at the contact between syenite (on the west) and meta-sedimentary rocks (primarily schists).

Initially explored by sinking four shafts (prior to 1910) and trenching along the structure, the vein zone has now been traced along strike in excess of 650 feet during three periods of diamond drilling (1965, 1981, 1985). The most recent exploration, carried out in 1985, consisted of drilling twelve holes (5,000 feet) along the strike extension of the Bannockburn mine structure and beneath the lowest level of the old mine workings. No gold mineralization of economic interest was encountered. It is evident that further exploration of the Bannockburn mine structure is not warranted.

5.2 NORTHEAST BLOCK

In the Northeast Block, a complex system of quartz veining (simulating a stockwork) occupies an area 1,200 feet in a north-south direction and 400-500 feet in an east west direction (Figure 4). To understand the character and behaviour of the 'stockwork' and to assess its economic potential, in excess of 40,000 feet of surface diamond drilling, 1,300 feet of underground excavation, and 5,700 feet of underground drilling have been carried out since early 1985. Since it contains the underground workings, the south half of the block (the Core Area) has been subjected to the most intensive exploration to date. The North Area has been investigated by systematic drilling from surface.

The 'stockwork' comprises two principal vein systems. The strongest, a first order or primary system, strikes northerly and dips 35 to 55 degrees to the east; a second order, or secondary system, strikes northeasterly and dips moderately to steeply southeast. The primary system evidently occupies a weakly to moderately persistent fracture set; the secondary system apparently follows the foliation in meta-volcanic and meta-sedimentary rocks. Within the general structural framework, third and fourth order veinlets and stringers occur, but are not as common or generally persistent.

The primary vein system is the main, if not the only, one of potential economic importance. From four to ten parallel, narrow quartz veins containing significant gold values occur in the system. Veins range from a few inches up to six feet in width. The secondary system, best exposed in the decline, consists of numerous, narrow, very short lenses of quartz exposed across a width of 250 feet.

Sulphide mineralization in the quartz veins is relatively sparse. Where present, it consists primarily of pyrite and pyrrhotite. Chalcopyrite, galena, sphalerite, arsenopyrite, and bismuth minerals are rare; calcite is more common. Visible native gold is not uncommon; it may also occur as tellurides on occasion. Hydrothermal alteration, consisting almost exclusively of carbonate, tends to occur erratically in patches, and is not necessarily associated directly with quartz veins.

In the Core Area, these 'veins' have been traced for some distance along strike by drifting (Figure 6). Vein behaviour in the drifts tends to be very irregular; thickening, thinning, pinching, splitting, bending, knotting, and local folding are typical. However, the principal controlling fracture tends to persist and, most importantly, gold values of some significance continue.

5.2.1 Core Area

In the Core Area, the veins have been explored by surface and underground diamond drilling and by limited drifting on the 1st Level (75 ft. depth). Core intercepts range up to 15 feet, but the large majority are in the order of one and two feet. Grades are highly variable, the highest encountered to date being 58.8 oz gold per ton across one foot. Drift assays (chips and mucks) are less variable because the samples comprise larger portions of vein material. The veins have been intersected over a strike length of 700 feet and a vertical extent of 300 feet. A 'blind' Lower Zone, 'stratigraphically' below the Core Zone, has a vertical extent of about 400 feet.

The mineralized quartz veins strike generally north-south but short local variations up to 45 degrees from this norm are not uncommon (Figure 6). Dips are variable and not well tested as yet. They appear to average 35 to 55 degrees to the east. Individual veins and fractures do not exhibit shearing. Instead, the veins are fracture controlled in what appears to be a zone of fracturing related to shear structures. Minor vein branching occurs, but more commonly there are breaks and offsets along a 'vein'.

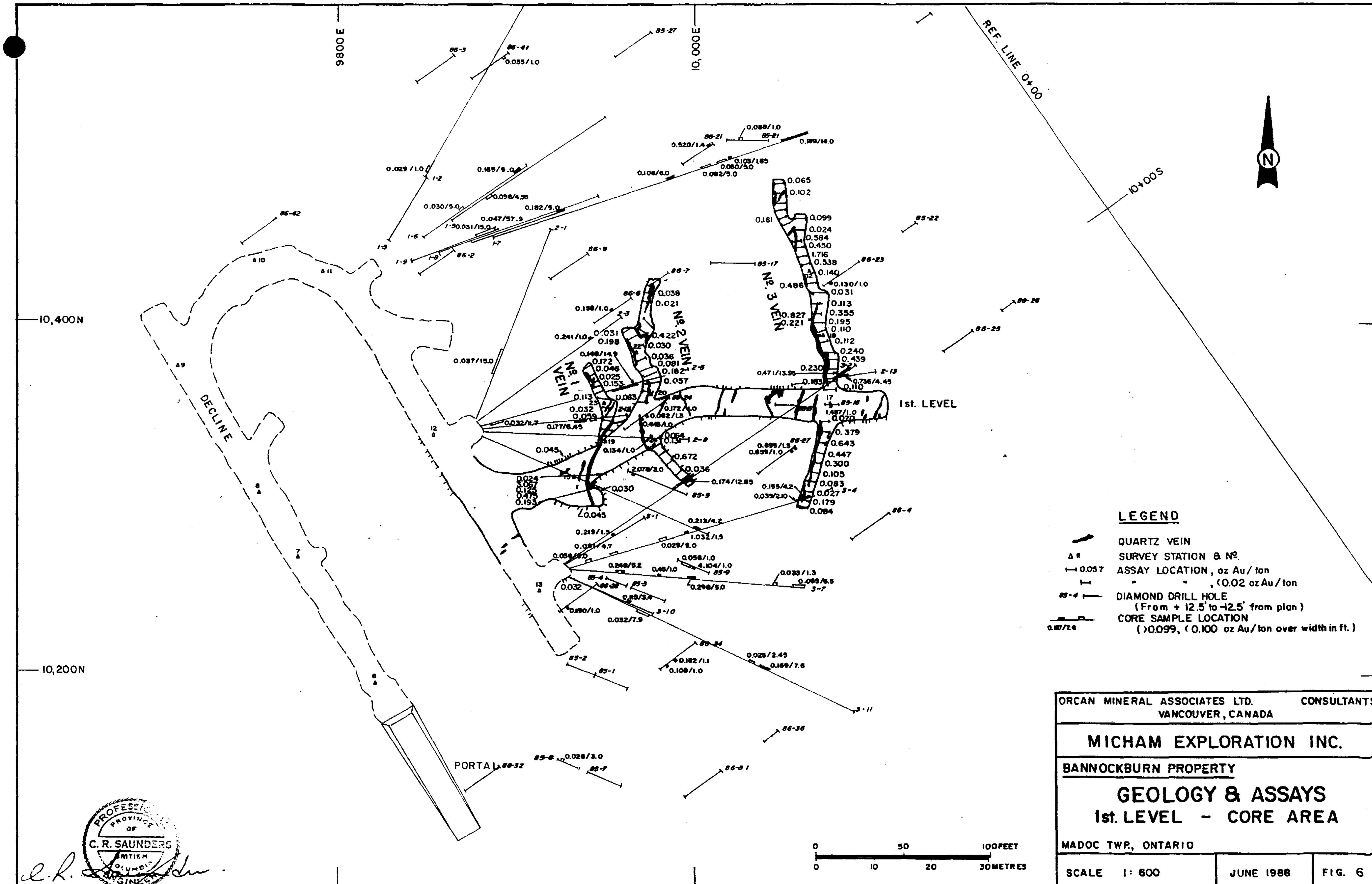
One vein, No. 3, exhibits the best continuity. It can be traced along strike for at least 400 feet, and along dip for some 200 feet. Elsewhere, there may be enough branching and small subsidiary veins to form local, mineable stockworks. Such an area could be present between No. 1 and No. 2 veins north of the 1st Level crosscut. If stockwork mineralization can be mined, ie. it is of mineable grade, then there will be possibilities for increasing the daily tonnage and/or lowering the overall mining costs through better stoping productivity.

Continuity of gold mineralization and of quartz veins is difficult to discern from the drilling results alone. Drill holes can easily pass through breaks in a vein or through small barren areas within a vein. The nugget effect of gold (Section 6) compounds this problem. A dense drilling pattern can help obviate the problem, but the only sure means of exploring a vein is by drifting and raising.

5.2.2 North Area

Mineralized quartz veins in the North Area have been explored by diamond drilling only. At least ten parallel veins are present within a zone about 500 feet in width and some 600 feet in length. Most intersections are less than 1.5 feet in thickness. The veins seem to strike north-south (correlation of individual veins is uncertain) and dip about 45 degrees to the east. Continuity is difficult to assess on the basis of drilling results alone.

A problem related to assaying procedures arose during the 1987 exploration of this area (see Section 6.2). Consequently, a number of intersections in which visible gold was noted returned low value assays. It is believed that most of these intersections are of ore grade, although present records indicate otherwise. They can be checked most effectively from underground by drifting and raising. Additional diamond drilling will be of limited help because of the erratic nature of the gold mineralization (nugget effect), but could be of use for correlation purposes. Obtaining more representative grades for these intersections (and for others where visible gold may have been present but not noted) should have a marked positive effect on the reserve grade and tonnage in the North Area.


9800 E
10,000 E
10,400 N
10,200 N
REF. LINE 0+00
10+00 S
N
No. 1 VEIN
No. 2 VEIN
No. 3 VEIN
DECLINE
1st. LEVEL
PORTAL
LEGEND
QUARTZ VEIN
SURVEY STATION & No.
0.057 ASSAY LOCATION, oz Au / ton
" " , (0.02 oz Au / ton
DIAMOND DRILL HOLE
(From + 12.5' to -12.5' from plan)
CORE SAMPLE LOCATION
(>0.099, (0.100 oz Au/ton over width in ft.)
0 50 100 FEET
0 10 20 30 METRES
ORCAN MINERAL ASSOCIATES LTD.
VANCOUVER, CANADA
CONSULTANTS
MICHAM EXPLORATION INC.
BANNOCKBURN PROPERTY
GEOLOGY & ASSAYS
1st. LEVEL - CORE AREA
MADOC TWP., ONTARIO
SCALE 1 : 600
JUNE 1988
FIG. 6
PROFESSIONAL
PROVINCE OF
C. R. SAUNDERS
BRITISH COLUMBIA
ENGINEER
CHONG

6.

RESERVES

6. RESERVES

6.1 SAMPLING PROCEDURES

6.1.1 Surface Sampling

Two types of sampling have been utilized on surface on the Bannockburn property: preliminary grab samples followed by detailed channel-type samples. Assaying was done on both types. The channel samples were cut at regular intervals (five-foot for the most part) with a diamond saw. Normally, the sawn cut extended across the full width of the vein and to a depth of approximately one inch.

6.1.2 Underground Sampling

In the underground phase of the operation, two conventional types of sampling procedures have been followed, namely chip-channel sampling of the faces and walls of the drifts and cross-cuts, and sampling of the broken muck from each successive round. In both cases the samples were taken by Micham employees.

Chip-channel procedures involved the cutting (chipping) of three samples across the full width of the drift face, normally a top, centre and lower cut, the assays of which were averaged arithmetically and assigned as a single value for that particular round. In addition, the faces of each development round were photographed as were any significant geological features on the walls of the drift. The sampling results, geological mapping and photographs of the drift headings make a complete and complementary record of the underground exploration.

Muck sampling procedures involved the transporting of the muck from each development heading to surface where it was dumped in an orderly fashion to form a surface stock pile for each round A representative sample was selected from various portions of each individual muck pile on a grid system. Individual muck samples were approximately 50 pounds each and were later averaged and weighted to the tonnage produced from each round.

6.2 ASSAYING PROCEDURES

All samples were sent for fire assay, with those containing visible gold being screened for metallics at 150 mesh. The +150 mesh fraction was weighed and the total fraction was fired to produce a dore bead. The -150 mesh fraction was treated normally; a one assay ton aliquot was taken for fire assay. The amount of gold was measured, in each case, by atomic absorption after digestion of the dore beads. The results of the two fire assays were averaged by weighting, with the figure obtained representing the assay for the sample. The procedure is standard for most laboratories when asked to 'take metallics'.

During the course of early exploration programs on the Bannockburn property, the exploration personnel became familiar with assay values in relation to visible gold content. It was generally accepted that an assay value of about 1.0 oz gold per ton was returned for each piece of visible gold noted. Results from the 1987 drilling program, comprising 10,102 feet in 32 holes, did not always follow this rule (see Figure 8). Investigation found that insufficient digestion time had been allowed for dissolving the gold in the higher grade dore beads. This resulted in reducing the grade of samples containing visible gold. Some comparatively high grade samples assayed lower than expected; other samples containing visible gold returned extremely low grade assays. For example, a sample with a piece of visible gold returned an assay of 0.060 oz gold per ton.

The 1987 drilling was done on sections at 50-foot intervals from just north of the Core Area to about the middle of the North Area. However, the same problem may have occurred during earlier drilling programs and for other high grade samples for which visible gold was not noted during core logging. The overall effect is to decrease some core intercept grades which, in turn, decreases the ore reserve grade. Resampling of the core, where possible, may help overcome this problem.

6.3 COMPARISON OF SAMPLING RESULTS

6.3.1 Chip and Muck Samples

The following table shows the relationship of the chip sampling results to the muck sampling results from the underground program on the 1st Level.

Vein	Drift Heading	Length (feet)	Tonnage (tons)	Chip Ave. (oz/t)	Muck Ave. (oz/t)
#1	1 Dr. North	45	157	0.073	0.057
#2	2 Dr. North	91	412	0.103	0.115
#2	2 Dr. South	35	103	0.178	0.027
#3	3 Dr. North	130	584	0.248	0.233
#3	3 Dr. South	51	219	0.233	0.244
Totals and Averages		352	1,475	0.182	0.169

For the most part the correlation is quite good considering the vagaries inherent in both chip and muck sampling of gold deposits. Furthermore, if the No. 2 Drift South samples are removed from the averages (because they show the greatest discrepancy between chip and muck sample results), the results are even closer: 0.182 for the chips and 0.179 for the mucks.

6.3.2 Chip/Muck and Diamond Drill Core Samples

Correlation between drill core assays and chip/muck assays is more difficult to establish. There are only a few drill core assays available for comparison even when core assays outside the drifts (within 15 feet along strike or dip) are included in the calculations. The averaged assays are shown in the following table.

Veins	Drift Heading	Average Grades (oz Au/ton) Chips	Mucks	Drill Core
#1	1 Dr. North	0.073	0.057	0.022
#2	2 Dr. North	0.103	0.115	0.076
#2	2 Dr. South	0.178	0.027	0.097
#3	3 Dr. North	0.248	0.233	0.247
#3	3 Dr. South	0.233	0.244	0.179
Weighted Averages		0.182	0.169	0.155

The drift results have up-graded the drill core results, particularly for the lower grade assays. Average increase in grade is 13.2 percent (9.0 for mucks, 17.4 for chips). This is a significant feature in terms of exploration and reserve calculations. In the latter case, the implication is that the cut-off grade used for reserve calculations made from drill core assays can be lower than the break-even grade required for mine operations. This apparent up-grading of core assays is an extremely important feature of the Bannockburn deposit; it must be thoroughly addressed in future exploration programs.

6.4 CUTTING HIGH GRADE ASSAYS

Most gold mines cut individual 'high grade' assays in order to produce more realistic reserve and production grades. The cutting is done on the basis of statistical studies and experience. It is necessary because of the 'nugget effect' of gold on sampling and assaying. However, 'cutting' is a process which can have a profound effect on the potential economics of a new, unmined mineral deposit and, thus, cannot be treated lightly.

A common approach to reducing the effect of high assays is to cut to 1.0 oz. This is an arbitrary system which may serve in many cases and which may even be an industry average, but it may also be misleading. For the Bannockburn deposit, a study was made of all drill core assays of 0.01 oz gold per ton or greater, a total of 819 assays. It was found that only three assays, all greater than 10.0 oz gold per ton, are truly anomalous. An additional 11 assays ranged from 3.0 to 7.0 oz gold per ton; the total number above 3.0 oz gold per ton is 1.7%. (The total above 1.0 oz gold per ton is 6.5%.) Consequently, it was decided to cut high grade assays to 3.0 oz gold per ton, still a slightly arbitrary figure, but one that, when used in reserve calculation, should more truly reflect the actual grade of the reserve blocks.

6.5 MINING RESERVES

Because of the present exploration status of the Bannockburn property, there are very little in the way of mining reserves that can be calculated. However, the category definitions and calculation parameters are set out in the following sections as background for their limited use in the current calculations and for use in future calculations.

6.5.1 Definitions

Mining Reserves - Mineralization that, for the most part, is potentially mineable at a profit and that occurs, usually with other mineable reserves, in quantities sufficient to support the capital costs normally associated with a mining project.

Proven Reserves - Reserves that have been exposed at least on one side along strike or dip by underground openings. Maximum projections are 25 feet along strike or dip.

Probable Reserves - Ore projections to a maximum of 50 feet beyond proven ore, where warranted by drill intersections or known geology. Maximum projection beyond proven ore, without support of drill intersections is 25 feet.

Possible Reserves - (a) Ore projections beyond proven or probable reserves to a maximum of 50 feet if based only on geology; (b) Ore grade reserves in zones clearly defined by three or more drill holes spaced not more than 50 feet apart.

6.5.2 Parameters

The following parameters have been used for determining potentially mineable ore in the Bannockburn deposit.

Cutting of High Grade Assays - Discussed in Section 6.4; all assays are cut to 3.0 oz gold per ton. (Results for uncut grades are shown as well.)

Minimum Mining Width - Because of vein dips ranging down to at least 35°, a common minimum true width (perpendicular to the vein) of six feet has been used.

Dilution - In most cases the minimum mining width is greater than the width of the ore grade intersections and no further dilution, other than to the minimum mining width, is necessary. In a few cases, where the intersection is more than five feet, one foot of dilution is added.

Dilution Grade - Any assays adjacent to ore grade intersections are used; if none are available, the dilution is included at zero grade. Where drift results are available, and the drift is comparable in width to a stope, the drift assays, (chips) are used.

Tonnage Factor - 12.0 cubic feet per ton.

Cut-Off Grade - The 'base-case' cut-off grade has been established as 0.05 oz gold per ton, a relatively low figure, but one which, at the current state of knowledge about the Bannockburn mineralization, appears to be justified. It represents an 'in-stope' cut-off rather than a zone cut-off. At current metal prices, 0.05 oz gold is worth about $20, an amount that will pay for at least half the estimated milling costs. (Low grade in a stope, that has to be mined and sent for waste, incurs virtually the same operating costs as ore, except for milling.) As noted in Section 6.3.2, there are good indications that low grade, as indicated by diamond drilling, returns significantly higher grades when mined - in the range of 50 to 150 percent higher. Thus, if on average, the cut-off value is doubled when mined, the value of the ore is sufficient to pay for milling costs.

In addition, the assaying problem noted in Section 6.2 will have caused a slight decrease in overall grade that will be expressed mainly as low grade assays.

It is expected, that on average, the lower grade reserve blocks will return appreciably higher grades when mined.

6.6 MODIFIED GEOLOGICAL RESERVES

Most of the Bannockburn reserves are in this category because little in the way of underground exploration (drifts, raises) has yet been done.

Geological Reserves are somewhat synonymous with 'Mineral Inventory'. They commonly include all reserves down to a low grade and very small width; they contain few mining parameters. Such reserves are based almost completely on drilling results.

In order to include some mining realism in these reserves, and thereby some potential economics, modifications to the more simple 'Geological Reserves' calculations have been used. Hence the term 'Modified Geological Reserves'.

6.6.1 Definitions

Indicated Reserves - Reserves defined by diamond drilling results; may be found in only one drill hole. Maximum projections on dip of 50 feet; horizontal projections from sections of 25 feet to either side (primarily dictated by the 50-foot spacing of the sections).

Inferred Reserves - Projections beyond Indicated Reserves where no further drilling results are available. No limits on project distance but commonly limited to 25 or 50 feet by geological considerations.

6.6.2 Parameters

The same as for Mining Reserves (Section 6.5.2.).

6.7 BANNOCKBURN RESERVES - 30 June, 1988

Most of the Bannockburn reserves (94%) have been calculated from information plotted on cross-sections (Figures 7 and 8 are examples). A small amount has been calculated on a longitudinal section of a portion of the No. 3 Vein. All of the cross-section reserves are classified as Modified Geological; those on the longitudinal are Mining Reserves.

All of the reserves are within the northeast part of the property, in the Core and North areas. For illustration purposes, the reserve blocks are shown on a plan and a longitudinal projection (Figure 9).

6.7.1 Reserve Summary

	Tonnage (short tons)	Grade (oz Au/ton)	
		Cut	Uncut
A. Mining Reserves	14,650	0.210	0.210
Geological Reserves	233,510	0.185	0.270
Total Reserves	**248,160**	**0.186**	**0.267**
B. Probable Reserves	14,650	0.210	0.267
Indicated Reserves	171,970	0.183	0.277
Probable and Indicated	186,620	0.186	0.272
Inferred Reserves	61,540	0.188	0.252
Total All Categories	**248,160**	**0.186**	**0.267**

6.7.2 Reserves by Levels

Reserves by Levels is somewhat of a misnomer in that only one level, the 1st Level at 75 feet below the surface, has been established at this time. However, the current plan is to drive new levels at 100-foot intervals and, thus, subsequent 'Levels' are defined by elevation at this time. Reserves for a level are those up to the next level.

Level	Tonnage (short tons)	Grade (oz Au/ton)	
		Cut	Uncut
1st (75 ft.)	74,120	0.199	0.391
2nd (175 ft.)	106,470	0.171	0.174
3rd (275 ft.)	41,090	0.194	0.214
4th (375 ft.)	15,740	0.221	0.413
5th (475 ft.)	10,740	0.166	0.312
Total All Levels	248,160	0.186	0.267

6.7.3 Reserves by Veins

Correlation of veins between sections is difficult and uncertain at the present level of understanding of the Bannockburn deposit. The only area where correlation is possible is in the vicinity of the explored part of the 1st Level, and then only for some veins.

Vein	Tonnage (short tons)	Grade (oz Au/ton)	
		Cut	Uncut
1	20,350	0.182	0.182
2	49,170	0.180	0.187
3	42,220	0.194	0.194
Unclassified	136,420	0.187	0.331
Total Reserves	248,160	0.186	0.267

6.7.4 Reserves by Cut-Off Grades

The base-case reserves are calculated at a reserve block cut-off grade of 0.05 oz gold per ton. The effect on tonnage and grade of raising the cut-off by 0.01 increments is shown in the following table.

Cut-Off Grade	Tonnage (short tons)	Grade (oz Au/ton)	
		Cut	Uncut
0.05	248,160	0.186	0.267
0.06	227,980	0.198	0.285
0.07	227,980	0.198	0.285
0.08	211,630	0.208	0.302
0.09	187,300	0.224	0.330
0.10	187,300	0.224	0.330
0.11	171,060	0.235	0.352
0.12	148,080	0.254	0.388
0.13	148,080	0.254	0.388
0.14	134,710	0.266	0.414
0.15	132,840	0.268	0.417

6.8 PROPERTY POTENTIAL

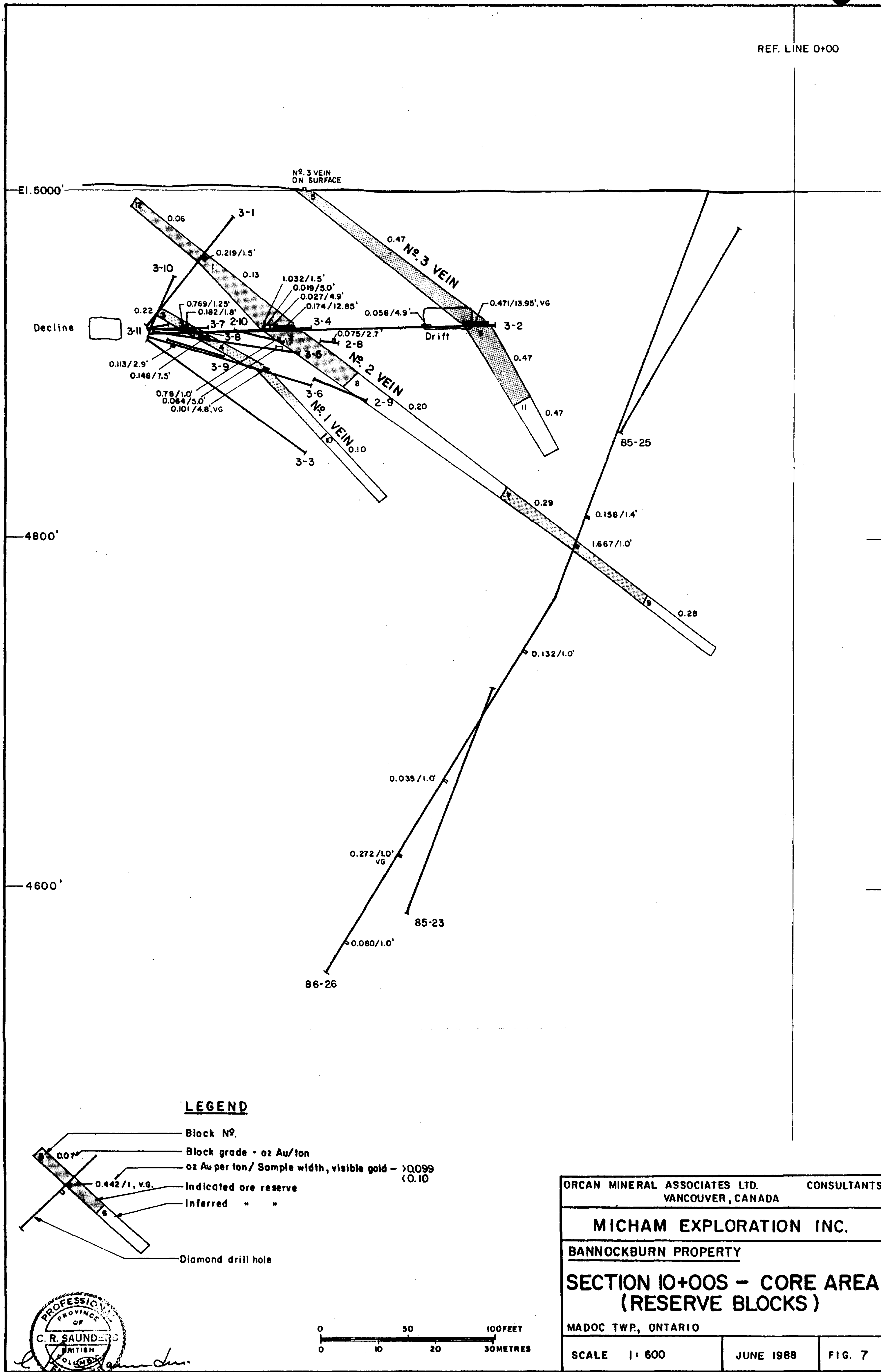
An estimate of the reserve potential of the Bannockburn property is required in order to provide some conceptual economics for further planning and financing.

The over-all reserve potential of the property cannot readily be estimated from present information. The depth continuity for mineralization has barely been tested; areas to the north of the North Area, south of the Core Area, and elsewhere on the property have received mostly surface examination; mineralization controls are not clearly understood; current reserves are based mainly on results of diamond drilling with very limited confirmation by underground work.

There is little in the geological setting to suggest that more mineralization of a similar nature will not be found elsewhere on the Bannockburn property. Further prospecting, geological mapping and, as necessary, diamond drilling are required.

The greatest potential, at this time, is in and about the area of the known deposit (Core and North areas). The depth potential is at least as much as the current reserves and could be several times larger. A zone 'stratigraphically' below the present Core/North Zone is indicated by several intersections in a few drill holes. This 'Lower Zone' is more nebulous than the Core/Noth Zone at this time, but it represents virtually new and untested mineral potential. It appears to be a 'blind' zone (does not reach surface), in which case, it suggests possibilities for other such blind zones nearby or elsewhere on the property. As well, extensions to the north of the North Area and to the south of the Core Area have not yet been thoroughly tested.

Until proven otherwise by future exploration, it is assumed and expected that significant additional reserves will be located on the Bannockburn property. For purposes of conceptual economics, and until a better estimate can be made, the property potential is considered to be a minimum of 500,000 tons.



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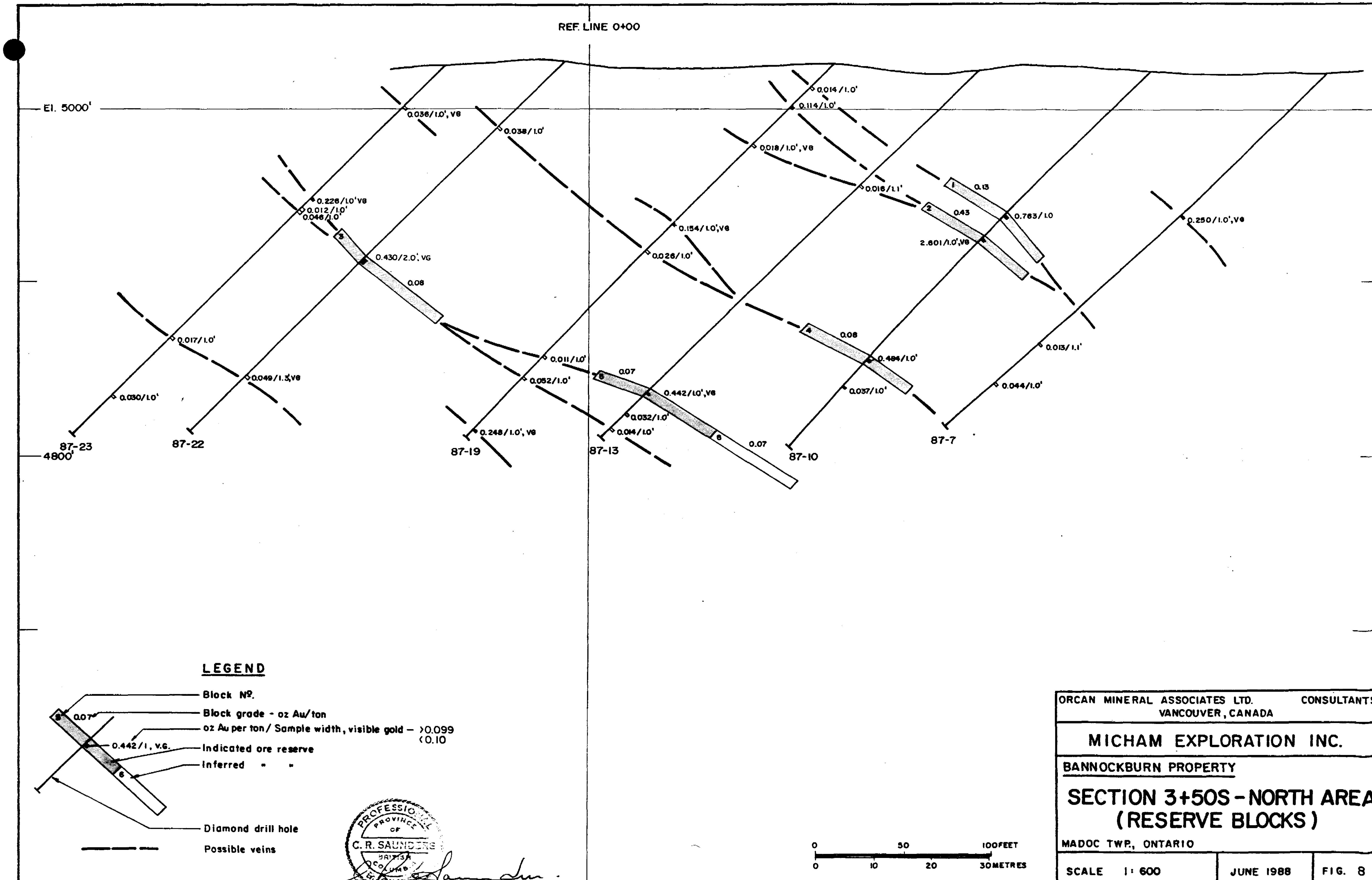
MICHAM EXPLORATION INC.

BANNOCKBURN PROPERTY

SECTION 10+00S – CORE AREA (RESERVE BLOCKS)

MADOC TWP., ONTARIO

SCALE 1 : 600	JUNE 1988	FIG. 7



REF. LINE 0+00
El. 5000'
4800'
0.036/1.0', VG
0.038/1.0'
0.014/1.0'
0.114/1.0'
0.018/1.0', VG
0.016/1.1'
0.13
0.763/1.0
0.43
2.601/1.0', VG
0.250/1.0', VG
0.226/1.0' VG
0.012/1.0'
0.046/1.0'
0.430/2.0', VG
0.08
0.154/1.0', VG
0.026/1.0'
0.08
0.484/1.0'
0.013/1.1'
0.017/1.0'
0.049/1.3', VG
0.030/1.0'
0.011/1.0'
0.052/1.0'
0.07
0.442/1.0', VG
0.032/1.0'
0.014/1.0'
0.248/1.0', VG
0.07
0.037/1.0'
0.044/1.0'
87-23
87-22
87-19
87-13
87-10
87-7
LEGEND
Block No.
Block grade - oz Au/ton
oz Au per ton / Sample width, visible gold — >0.099 <0.10
Indicated ore reserve
Inferred " "
Diamond drill hole
Possible veins
0.07
0.442/1, V.G.
PROFESSIONAL
PROVINCE OF
C. R. SAUNDERS
BRITISH COLUMBIA
0 50 100 FEET
0 10 20 30 METRES
ORCAN MINERAL ASSOCIATES LTD. CONSULTANTS
VANCOUVER, CANADA
MICHAM EXPLORATION INC.
BANNOCKBURN PROPERTY
SECTION 3+50S - NORTH AREA
(RESERVE BLOCKS)
MADOC TWP., ONTARIO
SCALE 1 : 600
JUNE 1988
FIG. 8

CHONG

7.

MINING

7. MINING

7.1 PRESENT STATUS

The property is presently equipped as an exploration site. The only existing underground opening to the gold-bearing veins is a decline at a slope of 15 percent (11 ft x 15 ft cross-section), collared at 10,170 North and 9,835 East, extending down to a depth of 90 feet. A total of 352 feet of drifting has been done on three veins from the decline on the 1st Level (Figure 6). The decline has been allowed to flood and the portal has been temporarily sealed with mine muck.

Ancillary services and facilities are minimal, and for this study are considered to be non-existent.

7.2 DESIGN CRITERIA

1) On the basis of current reserves, only a small operation is justifiable; 'Taylor's Rule' for determining production rate states:

$$\text{Production Rate (tons per day)} = \frac{5\ (\text{Expected Mineable Tons})^{0.75}}{\text{Working days per year}}$$

For 250,000 tons	=	160 tpd
For 500,000 tons	=	270 tpd

Considering the current reserves and potential of the Bannockburn property, an appropriate production rate for use in this study is considered to be 200 tons per day

2) The nature of the mineralization as it is presently understood, dictates that production will be from small narrow stopes; consequently, several stopes will have to be in operation simultaneously and development of new stopes will be an on-going process.

3) Current reserves are near-surface (within 500 feet); access will be by ramp from surface to obviate the larger capital costs of a shaft and related sophisticated equipment.

4) Ground conditions are good with respect to drilling, blasting and support.

5) The distribution of gold in the quartz veins precludes, in most cases, the use of diamond drilling results for detailed stope planning, although the results will be useful for general mine planning. Consequently, for definition of ore shoots, it will be necessary to drift and raise on the veins.

6) Stope development and stoping methods are predicated on dips ranging from 35 to 60 degrees.

7) Minimum mining width is six feet; most stoping will be done at minimum mining width.

8) Water inflows are unlikely to be a serious problem.

7.3 CONCEPTUAL MINE DESIGN

Current information regarding reserves and ore zone configurations is insufficient for comprehensive mine design although it does provide a framework for conceptual design. Consequently, the description and related drawings in the subsections which follow may be somewhat idealized because they do not always relate to specific ore zones or veins.

7.3.1 Daily Mine Production Rate

At a continuous milling rate of 200 tons per day for a seven day week, mine production will have to be $200 \div 5 \times 7 = 280$ tons per day for a five day week. Surge capacity in the order of 900 to 1,000 tons, i.e. sufficient for a three day weekend, will be required on site.

7.3.2 Main Access

The present 11 ft x 15 ft decline, driven at -15 percent, will serve as the main access and haulageway. It will be continued down to a depth sufficient to develop the current reserves, approximately 500 feet. If further exploration, in the form of deep diamond drill holes, indicates a continuation of the ore zones to considerably greater depth, it will become necessary to evaluate the economics of a shaft versus further deepening of the decline.

7.3.3 Secondary Access

Secondary access for ventilation and escape will be provided by one or more 7 ft x 7 ft raises. Depending on the position of the decline and of the developed levels, they will be connected with the decline at minimum 150 ft vertical intervals or, where possible, with each level.

7.3.4 Level Development

It is proposed that levels be developed from the decline every 100 feet vertically from surface. The interval of 100 feet, rather than 150 feet which is common in many mines, has been selected because of the variability of grade and continuity of the mineralization and the apparent variability of vein dips. The chief disadvantage of a close level spacing is the increased cost of development. In this instance, at the initial stage of the project, this disadvantage is offset by better definition of the ore zones for development and mining purposes. Where the ore rolls or rakes between levels, a closer level interval allows greater flexibility in the application of stoping methods.

The option exists either to drive level development on a comparatively large format, eg. 10 ft x 10 ft, for use with trackless diesel equipment, or on a smaller format, eg. 7 ft x 7 ft, for use with trackbound battery-electric and compressed air equipment.

For this project, level development will be both a significant cost and a significant source of ore. It is therefore important to minimize both the cost of development and the dilution of the development ore by waste rock.

Although very small trackless diesel haulage equipment exists, its economical range is very restricted (about 100 feet for a 1 cu yd diesel scooptram) and its ventilation requirements are substantially greater than those of compressed air or battery-electric machinery. If greater distances are to be driven, larger diesel equipment is needed and thus more ventilation and larger drift cross-sections.

For these reasons it is proposed that the levels be driven, following the veins, on a 7 ft x 7 ft format with 24-inch track and 40-pound rail. These level drifts will serve for ore definition and for haulage; after drifting on a vein is completed, some slashing of the sharper bends can be done to facilitate production tramming. The ore will be hauled in two-ton 'rocker-dump' or equivalent muck cars pulled by small battery locomotives. It will be dumped into a box-hole or ore pass and will then be loaded through chutes into diesel trucks in the ramp for transport out of the mine.

7.3.5 Raises

Ore will be defined between levels, and its dip ascertained, by means of 7 ft x 5 ft raises driven to follow the gold-bearing veins. Spacing of the raises will be dependent on the combined results of geological mapping, sampling of the levels, and diamond drilling information; ideally, they will be at approximately 100-ft intervals. The raises will later serve for stope access and ventilation and as secondary escapeways between levels. They may also serve as temporary ore and waste passes should the need arise.

7.3.6 Stoping

From the geometry of the mineral zones, two primary stoping methods are envisioned: shrinkage and open stoping. Raises driven between levels, eventually to be used for stope development, will serve to indicate the continuity and dip of the ore shoots and, thereby, be of prime importance in determining which stoping method is most appropriate to the ore zone being developed.

Shrinkage Stoping

Shrinkage stoping will be applicable to those zones where the footwall dips at 50 degrees or steeper. There is the option of drawing the broken ore through chutes direct into muck cars or through drawpoints by means of a rocker-shovel loading into muck cars. The first method is the most productive but has several disadvantages, particularly where comparatively small tonnages are involved:

- large, expensive timbers are required
- skilled timbermen are required
- chutes hang-up more easily than drawpoints and are more vulnerable to the effects of blasting than are drawpoints
- when a stope has been pulled empty a chute must be strongly bulkheaded if the drift below is to be kept in use.

For these reasons, it is proposed to use drawpoints for muck extraction. Mining will be by uppers drilling with handheld drill machines (stopers). Access will be via raises and timbered manways. An approximate 10 ft crown pillar will be left at the top of the stope in order to maintain use of the level above. The pillar will be blasted out when the upper level is no longer needed.

Open Stoping

Where an ore zone is flatter than 50 degrees broken muck may not run freely enough to permit drawing down a shrinkage stope, either in the course of the mining or when it is being drawn empty. In this case it will be necessary to mine the ore in panels up-dip using pillars of waste or low grade rock for support. It may be necessary to install stulls as a working platform, and to slush ore down the footwall and into drawpoints or boxholes.

It is proposed to develop a stope from short cross-cuts driven into the footwall from the vein drift; a boxhole will be angled back to intersect the vein about 20 feet above the drift. A chute will be installed in the cross-cut. The sill pillar will be recovered from the stope below when the level can be abandoned.

Other Stoping Methods

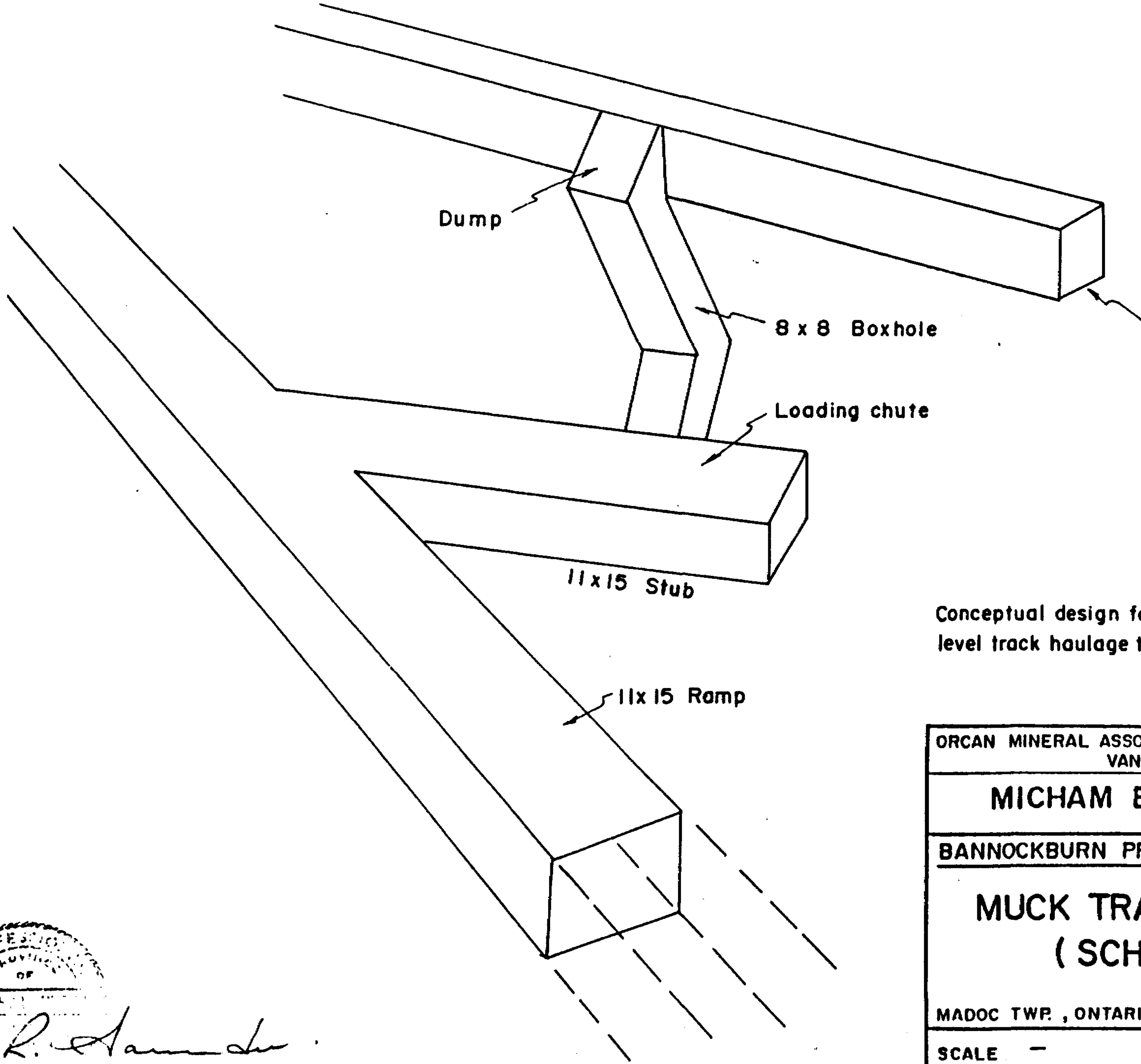
It is recognized that unique ore zone configurations may require special development and stoping solutions. In such cases, costs of development and mining versus total recoverable value will have to be estimated for each case.

7.3.7 Ground Support

Ground conditions already encountered in the decline and drifts indicate that support problems will be minimal and, for the most part, can be solved by the use of expansion-shell rockbolts. Additional support of backs in open stopes will be provided by rock pillars and stulls.

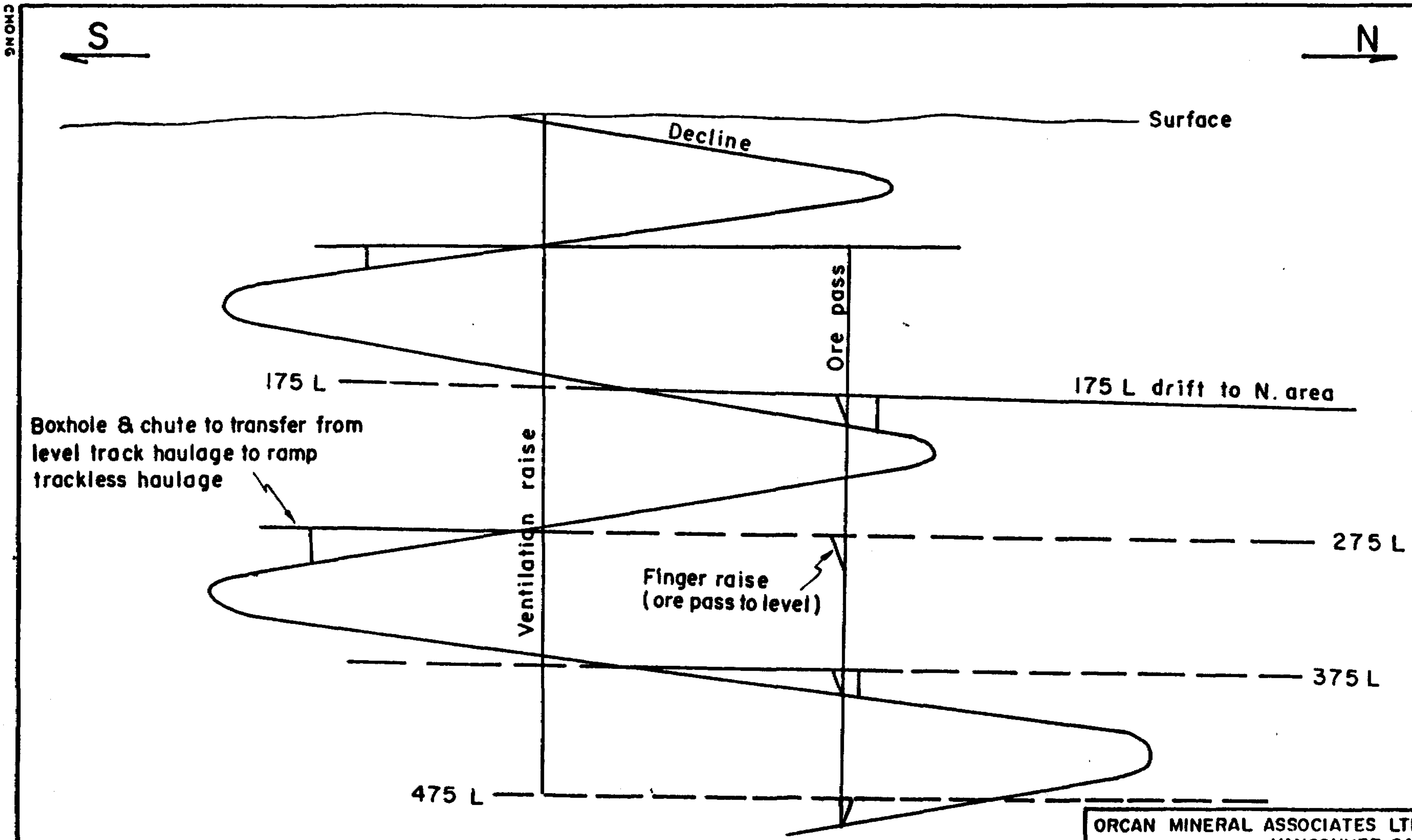
7.3.8 Ore and Waste Handling

The transport of ore and waste on the levels is discussed in section 7.3.4: small battery-powered locomotives will haul two-ton rocker-dump muck cars.



Conceptual design for muck transfer system from level track haulage to ramp trackless.

ORCAN MINERAL ASSOCIATES LTD. VANCOUVER, CANADA — CONSULTANTS		
MICHAM EXPLORATION INC.		
BANNOCKBURN PROPERTY MUCK TRANSFER SYSTEM (SCHEMATIC) MADOC TWP. , ONTARIO		
SCALE —	JUNE 1988	FIG. 10



ORCAN MINERAL ASSOCIATES LTD. VANCOUVER, CANADA	CONSULTANTS

MICHAM EXPLORATION INC.

BANNOCKBURN PROPERTY

ORE PASS SYSTEM (SCHEMATIC)

MADOC TWP., ONTARIO

SCALE -	JULY 1988	FIG. 11

SCHEMATIC OF BANNOCKBURN MINE LOOKING EAST

CONCEPTUAL DESIGN FOR ORE PASS SYSTEM

PROFESSIONAL ENGINEER PROVINCE OF BRITISH COLUMBIA C. R. SAUNDERS

C. R. Saunders

It is essential that there be surge capacity underground both between the level and ramp transport systems and, on a larger scale, between the mine output and the mill intake. For these reasons, it is planned to develop a local box-hole system for each level and an ore pass for the mine as a whole. Such a system is also necessary for the separate handling of ore and development waste muck. The boxholes will be driven to the levels from points on the main ramp some 20 to 30 feet below (Figure 10). Diesel-powered trucks will be loaded from the boxholes through air-operated chutes. The muck will then be hauled up the ramp to the mill or to a waste disposal area.

The ore pass system will run from the top level to the bottom of the mine with a chute at the bottom (Figure 11). The ore pass will be fed by finger raises from each level, separate from the boxholes. The installation of control chains to hold the muck above a level (and thereby permit dumping from a lower level) is an option that is considered for only two levels at this time (of a possible four). The operational flexibility provided by control chains is partially provided by the boxhole system at each level.

Production haulage requirements can be met by one truck with a capacity in the order of 25 tons.

7.3.9 Ventilation

The main ventilation of the mine will be accomplished by means of a large intake fan on the collar of the ventilation raise with the ramp used for exhausting. Some auxiliary exhausting may be done through stope raises and completed stopes that form airways from level to level and to surface. Ventilation control within the mine will be by means of doors and brattices. Dead ends will be ventilated by booster fans and collapsible plastic ducting, 42 inch on the ramp and 16 inch on the levels. Raises will be ventilated while driving by 3 inch venturis.

A heater will be installed at surface to heat the intake air during periods of cold weather.

7.3.10 Drainage

Minor water inflows have been encountered in the decline but none of a serious nature. To handle this water, and water from mining and diamond drilling operations, sumps will be developed from the main ramp at each level. Pumping will be from level to level, at least for the initial years of operations; each sump will be equipped with an electric pump on an automatic switch.

The levels will be driven slightly upgrade (0.5%) so that water will drain back to the sump at the ramp.

7.4 DEVELOPMENT PLAN

Development planning is somewhat speculative at present, being based on limited information about zone configurations and characteristics, on mostly unconfirmed reserves, and on conceptual mine design. However, reasonable generalizations can be made on which to base capital and operating cost estimates suitable for the present level of study. Alterations and refinements can be made as more information becomes available, in particular, after the proposed feasibility work has been completed.

It is assumed that when a production decision is made, the main access ramp (decline) will be down to just below the 2nd Level; veins will have been completely drifted on the 1st Level in the Core Area and partially (half) drifted on the 2nd Level in the Core and North areas, and several raises will have been driven between levels and to surface.

7.4.1 Quantities

Access - Driving to each level totals about 900 feet including diamond drill bays and the main cross-cut on the level.

Drifts - For the present, it is estimated that 1,000 feet of vein drifting will be required in the Core Area on each new level, 1,000 feet per level in the

North Area, 500 feet between the Core and North areas, and 400 feet of cross-cut and drill bays in the North Area. In addition, other drifting for deep hole diamond drilling set-ups, exploration outside the Core and North areas, and for general mine operations (charging stations, boxhole cross-cuts, etc.) is estimated to be 200 feet per level. At present, it is not planned to drift between the Core and North areas on the 1st Level. Ore definition above the 2nd Level of the North Area will be by raises and, if required, by sub-levels.

Level	Drift Footage
1	0
2	1,000
3	2,500
4	2,500
5	2,500
Other	800
	9,300 feet

Raises - Exploration and development raise footage on veins is estimated to be about 1.4 times the vein drift footage, i.e., about 1,400 feet per 100-foot vertical interval for each of the Core and North areas. In addition, a ventilation raise from 5th Level to surface and an ore pass raise from 5th Level to 1st Level, will be required.

Level	Raise Footage
1	1,000
2	1,700
3	2,800
4	2,800
5	2,800
Ventilation	600
Ore Pass (with fingers)	600
	12,300 feet

Diamond Drilling - A nominal footage of 15,000 feet per level is included at this time which, excluding any further drilling on the 1st Level, totals 60,000 feet.

7.4.2 Proposed Rates of Advance

The amount of development required to establish and sustain the proposed mine production (approximately 6,000 tons per month) is uncertain and difficult to estimate from present data. It has to be assumed that reserves calculated for the 1st and 2nd levels, where the most complete information is available, will be repeated on the lower levels, i.e., about 1,000 tons per vertical foot, and thus the amount of development on each level will be about the same. This is an idealized situation but it will have to suffice until more data are available.

Access - Experience indicates an advance rate of approximately 600 feet per month.

Drifts - On the basis of present reserves and a very approximate calculation of drifting requirements on the 2nd Level, it is estimated that, on average, about 30 tons of ore will be developed per foot of advance. This equates to approximately 200 feet of drifting per month. During the initial few years of operation, it will be necessary to exceed this average in order to provide more mining flexibility, to provide more lead time for reserve definition and mine planning, to explore potential but less obvious mineralized zones, and to obviate possibly disappointing periods of underground exploration. It is proposed that four drift crews (i.e. 500 feet per month) be employed during the first year of operation, and two crews (250 feet per month) thereafter. Both of these figures should be under periodic review as more data become available.

Raising - Productivity should be similar to that for drifting, particularly when it is considered that all or portions of some raises will require slushing because of low angle dips. Again, as with drifting, the amount of raising will have to be greater during the first year or so of operations in order to establish stopes, define reserves, and provide operating flexibility. Additional footage is required for the ventilation and ore pass raises as well. Calculations based on 1st Level data indicate that about one foot of raise will be required for each 25 tons of ore to be mined. This is approximately 250 feet per month (the equivalent of two raise crews). Consequently, it is proposed that four raise crews (500 feet per month) be employed for the first year and two raise crews thereafter.

Diamond Drilling - Diamond drilling is not a limiting factor on development, other than it should be done as early as possible on a new level. One machine operated on two shifts can drill 5,000 feet per month.

7.4.3 Timing

Some of the development is sequential and, therefore, is on the critical path for mine development and production. Other development which follows, while essential for stope preparation, can be conducted simultaneously. The critical path sequence is: (1) drive decline; (2) drive ore transfer drift and boxhole; (3) drive on vein towards North Area; (4) drift and cross-cut into North Area. Other vein drifting in the Core Area can get underway after (2); raising in the Core Area can follow shortly thereafter. A somewhat idealized schedule is shown in Figure 12; it does not take into account work that may be going on elsewhere in the mine (levels above; eventually, levels below). However, it does show that, at best, from initial development until stoping is underway in the Core Area about six months will elapse; in the North Area, about one year. During development, some of the drift muck and most of the raise muck should be ore. Estimated ore production and sources are shown on Figure 12 as well. The figures are for illustrative purposes and thus subject to considerable alteration as planning becomes more detailed.

7.5 PRODUCTION MINING

On average, ore production will be from the following sources:

Micham Explorations Inc. - Bannockburn Project

FIGURE 12: CONCEPTUAL DEVELOPMENT SCHEDULE

Months	1	2	3	4	5	6	7	8	9	10	11	12	Total
GENERAL MINE DEVELOPMENT													
Decline	2 - 3		3 - 4		4 - 5								2,100'
Level Preparation													
Main Cross-cut		3		4	5								600'
Transfer Drift		3		4		5							600'
Boxhole & Chute		3		4	5								90'
Other Drifting													250'
Raising													
Ventilation Raise													600'
Ore Pass													600'
LEVEL DEVELOPMENT AND ORE PRODUCTION (3rd LEVEL)													
Core Area													
Vein Drifting													1,000'
Raising													1,400'
Stope Develop.													-
Stoping													-
North Area													
Access Dr. & X-C													900'
Vein Drifting													1,000'
Raising													900'
Stope Develop.													-
Stoping													-
Diamond Drilling													
Core Area													5,000'
North Area													5,000'
Other													5,000'
Est. Production*													
Driftng		100	300	700	800	700	800	600	800	600	800	200	6,400
Raising				300	700	700	700	700	1,200	1,400	1,400	700	7,800
Stope Develop.					500	500	500	500	600	1,000	1,000	4,600	
Stoping							300	500	1,000	1,500	2,000	3,000	8,300
Total	0	100	300	1,000	1,500	1,900	2,300	2,300	3,500	4,100	5,200	4,900	27,100

* Approximate tonnages (for illustrative purposes)

2 - 3 2nd Level to 3rd Level

Critical Path

Maximum Advance (non-crash program)

Indefinite Timing

Development (drifts, raises)	12-15%
Stope Development	5-10%
Stoping	75-80%
(stoping includes 20-25% pillar recovery)	

However, depending on the amount of development being done at any particular time, as much as 50 percent of ore production could come from drifting, raising and stope development.

Approximately 40 percent of stope production is expected to be derived from shrinkage stopes and 60 percent from open stopes. (Note: From the present sections (Figures 7 and 8), it appears that most of the ore zones do not dip steeply enough to be mined by shrinkage stoping. The sections show 'apparent dip' because they are not at right angles to the strike of most of the veins; the true dip is steeper.) Daily production per two-man crew should be approximately 100 tons drilled and blasted for a shrinkage stope and 40 tons for an open stope. The rate of muck extraction will be about 30 tons per shift during shrinkage stope mining and 40 tons per shift for open stope mining. The muck remaining in the shrinkage stope will be pulled as required.

When development, stope development and stoping are stabilized, monthly ore production will be approximately as follows:

	Tonnage
Development	700
Stope Development	500
Open Stopes	3,000
Shrinkage Stopes (mining)	600
Shrinkage Stopes (pulling)	1,200
	6,000

This production will require the following number of single-shift working places; double-shifting will require fewer active working places.

Stopes being developed	2
Open stopes mining	4
Shrinkage stopes mining	1
Shrinkage stopes pulling	1

Extraction of sill and crown pillars is included as stope production.

8.

MILLING

8. MILLING

8.1 REVIEW OF METALLURGICAL TESTWORK

Metallurgical testwork has been done on one sample of Bannockburn 'ore'. The work was done by Witteck Development Inc. of Mississauga, Ontario during the latter half of 1987. The sample tested comprised 1,500 pounds of material blasted from No. 3 Vein on surface. The average assay, from seven samples, was 0.295 oz gold per short ton. The rather high standard deviation of the assays, 0.080 oz gold per ton, has been attributed to the nugget effect of gold. Testwork included mineralogical studies, grindability (Bond Work Index), gravity concentration by tabling, flotation, cyanidation of flotation concentrates, direct cyanidation of the ore, and leach residue filterability.

The sample should be representative of the general character and nature of the Bannockburn ore although, on the basis of current reserve estimates, it is somewhat higher grade than 'run-of mine ore'. The effects of surface oxidation are probably minimal, but further testing should be done on samples collected from underground. The samples should include all ore and waste types in order to be absolutely sure of the physical characteristics of the material entering the mill. More testwork is in order to confirm the present results and to determine results for different grades, particularly those closer to the current reserve grade of 0.186 oz gold per ton.

Witteck's testwork is thorough and appears to have been well done. The selection of a conventional cyanide mill circuit appears to be the right decison based on the Witteck report. Gold recovery by this process should be in the order of 98 percent.

8.2 MINERALOGICAL STUDIES

Mineralogical studies were carried out on heavy concentrates obtained from the metallurgical test sample. Four polished mounts were made of this material. The concentrate contained 86.3% of the gold; the remaining 13.7% was in the tails. The important minerals found were sulphides, several bismuth minerals, several iron oxides and native gold. Pyrite accounted for about 80% of the concentrate, marcasite for less than 10%, pyrrhotite for 3-5%, arsenopyrite for 3-4%, and the remainder 2-3%. Gold sizes range from 1 micron to 100 by 130 microns (elongate grains to 170 microns).

To summarize, the auriferous mineralization occurs as fully liberated, fine to coarse grained particles of silver-poor gold, and also as numerous middling exposed particles commonly associated with arsenopyrite, chalcopyrite, several types of bismuth minerals, galena and gangue silicates. The ore should be highly amenable to gravity separation of the coarse free milling gold. Cyanidation of the gravity tails or direct ore feed should provide excellent gold extractions. In the coarse sizes (48 mesh), all of the gold is not free, but it is expected that it will be free or exposed at finer grinds (minus 200 mesh) to which material is usually milled prior to cyanidation in agitated tanks.

8.3 RECOVERY PROCESS

The following notes on the Recovery Process are based on the Preliminary Flow Sheet (Figure 13).

8.3.1 Crushing

Crush to -3/8 inch on an operating basis of ten hours per day for six days per week at a rate of approximately 25 tons per hour.

8.3.2 Fine Ore Storage

Require a live capacity of 350 to 400 tons.

8.3.3 Grinding

Use an 8 ft x 8 ft Ball Mill with a 250 HP motor. This is slightly larger (by about 20%) than is necessary as indicated by the initial testwork. If the grinding circuit is inadequate it presents a serious bottleneck at the front-end of the process which is not easily resolved. Further testwork should be done to confirm the work index of the ore.

8.3.4 Gravity Concentration

A jig should be installed ahead of the cyanide circuit (after the Ball Mill) to recover coarse gold from the ore. Without suitable gravity recovery, some of the coarse gold would otherwise be lost.

8.3.5 Preaeration

Testwork indicated a preaeration time of eight hours. Additional capacity has been added to provide volume which may be required if the thickener has to be pumped out, or if leaching has to be curtailed for maintenance. It also makes provisions for some surge capacity within the circuit.

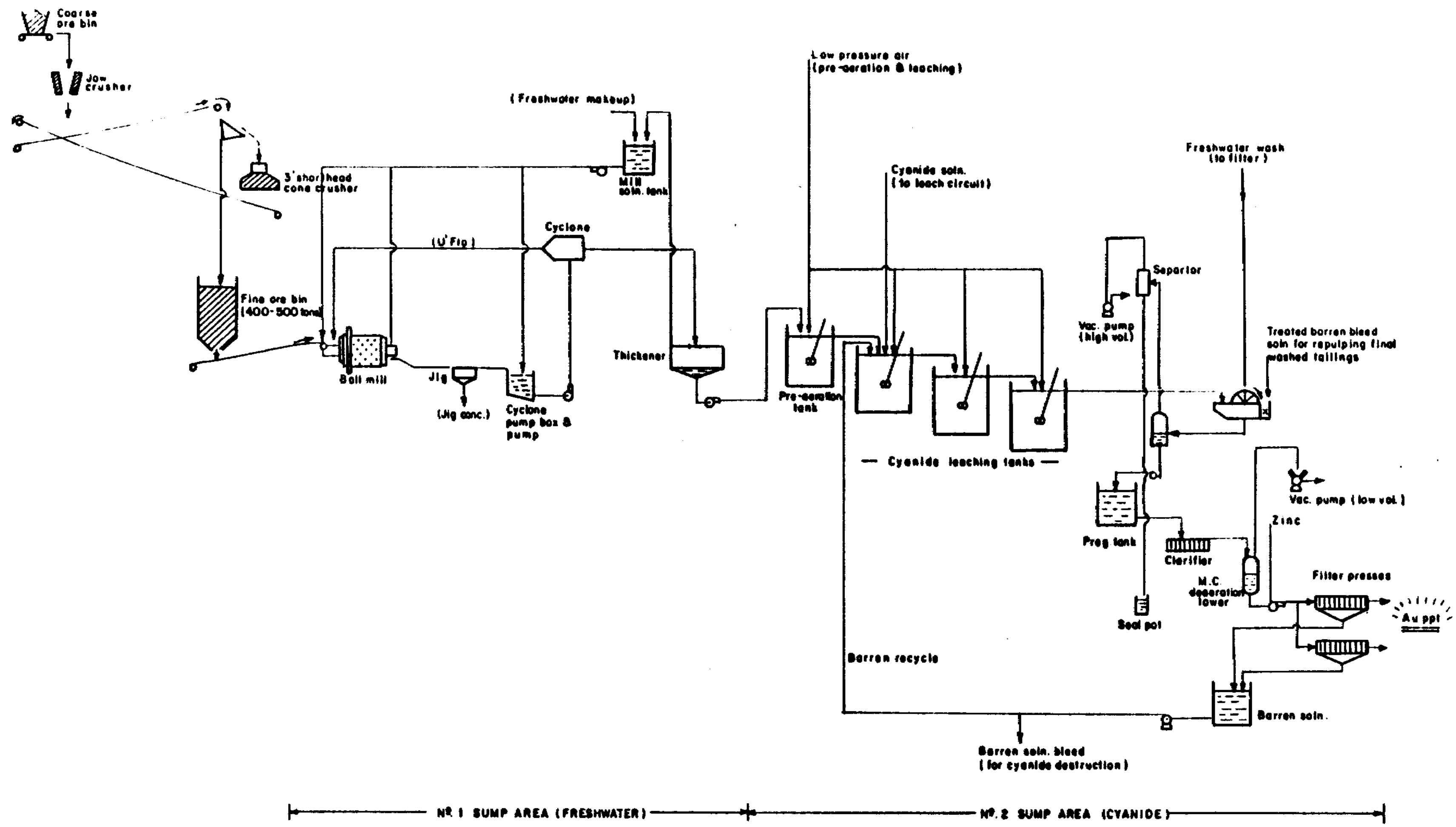
8.3.6 Leaching

A 48-50 hour leaching period is required according to results of the testwork. This circuit has three stages, each able to provide a 20-hour residence time. The additional time will be necessary when routine maintenance on the filter has to be done (changing filter cloths, etc.).

8.3.7 Filtration

The leach pulp discharges directly to a filter unit. A single stage filtration will be adequate if effective washing can be maintained at worst operating conditions, i.e. when cloth life is nearly finished. The filter unit should be 300 sq ft to provide sufficient washing capacity to maintain minimum soluble loss at all times.

CHONG



ORCAN MINERAL ASSOCIATES LTD. VANCOUVER, CANADA — CONSULTANTS

MICHAM EXPLORATION INC.

BANNOCKBURN PROPERTY

PRELIMINARY FLOW SHEET

MADOC TWP., ONTARIO

SCALE —	JUNE 1988	FIG. 13

8.3.8 Precipitation

The Merrill-Crowe system is ideally suited to the direct precipitation of gold and silver from relatively clean and easy-to-filter pulps. A single deaeration tower will be adequate. The final zinc precipitate is expected to filter easily in a standard plate and frame filter press.

8.4 TAILINGS DISPOSAL

Without firm information with respect to the disposal of tailings, the process outlined in Figure 12 does not show how wastes leave the mill. Tailings can be slurried and pumped to a containment area or they may be 'conveyed' essentially as 'dry' waste. Beyond the filtration stage either method may be used depending on the circumstances or constraints which apply to this specific situation. Barren solution will contain cyanide and it is usually possible to recycle a major portion of this stream to cut down on the amount of cyanide required overall. This always has to be balanced against the effect of build-up of unwanted ions in solution (poisons). The circuit shown has provision for reuse of 30% of the barren effluent. The remaining 70% may be treated for cyanide destruction prior to disposal either as a separate non-toxic effluent or as the liquid required to form a pumpable final tailings slurry to a conventional tailings pond area. Note that it may be possible to recycle a higher proportion of the barren solution if overall quality is reasonable.

To minimize the use of freshwater, consideration should be given to putting out tailings in 'dry' form. The conventional tailings pond concept requires either the retrieval of supernatant liquid or controlled overflow for cyanide destruction prior to discharge beyond the limits of the mine/mill complex. Experience shows it is a much easier and more preferable task to recycle barren solution directly in the mill and to treat within the mill the portions which have to be 'environmentally acceptable' for discharge.

Freshwater into the circuit should be fed as washwater to the filters. Any additional freshwater requirements will be on an 'as required' demand basis introduced directly to the mill solution feed tank (make-up water for grinding).

9.

ANCILLARY SERVICES AND FACILITIES

9. ANCILLARY SERVICES AND FACILITIES

9.1 ELECTRICAL POWER

It is almost certainly possible in this instance to have the mine and mill connected to an Ontario Hydro power supply. The nearest available connection is to a 44 kv line along Highway 7 nearly ten miles from the mine site. Whether it will be more economical to connect to the hydro grid or to use diesel-generated power will depend on the arrangement that can be made with Ontario Hydro regarding installation costs.

In any event, diesel-generated standby power to supply 50% of the total load will be required. The chief requirement will be to keep the mill circulating load moving. On the other hand, the provision of heat and light will be an important safety feature, especially in winter.

Power consumption is expected to be:

Mill	380 kW
Plant	850 kW
Mine	200 kW

9.2 WATER SUPPLY

9.2.1 Underground

Water for drilling and dust suppression can be obtained from the nearby ponds and creeks. This source was entirely satisfactory during the exploration phase in the early months of 1988.

9.2.2 Process

It seems likely that the nearby ponds and creeks will supply water for the mill in sufficient quantity and quality. The need for pH control during some parts of the year should not be ruled out. As far as possible, process water should be recycled in order to protect the environment, especially with regard to the residences nearby which draw their water from shallow wells.

9.2.3 Potable

A well was drilled for the exploration phase of early 1988 which yielded potable water in sufficient quantity and quality. A larger operation could entail the drilling of other wells.

9.3 COMPRESSED AIR

If a decision is taken to connect the mine to the Ontario Hydro grid then the most efficient course of action will be to install stationary electric compressors. If the mine is to generate its own power, however, self-contained diesel compressors are recommended. Total requirements for the mine will be in the order of 3,250 cfm at 100 psi.

Compressed air is to be supplied by two out of three compressors so that servicing can be carried out regularly and production can be sustained in the event of a serious breakdown.

Compressed air could be conveyed underground through the 6-inch pipe used for driving the decline but consideration should be given to substituting an 8-inch main line to reduce line losses. Distribution lines on the levels will be of 4-inch pipe.

Receiver tanks will be installed at the portal and at the entrance to each level to act as water traps.

9.4 MAINTENANCE FACILITIES

The following maintenance and storage facilities will be required:

(a) 40-man dry and lamproom

(b) 40 ft x 60 ft heavy equipment workshop

(c) 8 ft x 10 ft rockdrill and small equipment workshop

(d) 8 ft x 10 ft electrical shop

(e) 40 ft x 20 ft compressor house

(f) 15 ft x 15 ft mine rescue, first aid, safety room

(g) 40 ft x 60 ft warehouse

Separate from main complex and from each other:

(h) Fuel and lubricant storage

(i) Powder magazine

(j) Cap magazine

The existing 40 ft x 60 ft workshop building may be adequate for items (b), (c) and (d). A group of trailers or modular buildings will suffice for (a) and (f). A purpose-built building would be constructed on site for (e), or it, along with the warehouse (g), could be included in the mill complex.

Storage areas and buildings will have to conform to Ontario Health & Safety Act standards.

9.5 OFFICE FACILITIES

Office space, in the form of trailers or modular buildings, will be required for the operations manager, mine superintendent, mine foremen, surveyors, samplers, geologist, accountant, bookkeeper and secretarial staff.

Offices for mill supervisory and technical personnel will be included in the mill building.

10.

ENVIRONMENTAL IMPACT

10. ENVIRONMENTAL IMPACT

In the on-going process of assessing the Bannockburn property with the view to developing an operating mine, it is necessary, as part of this process, to address the potential environmental impact of such an operation. The following sections set out some of the environmental requirements and indicate areas of greatest concern.

10.1 ENVIRONMENTAL CONSIDERATIONS

Only obvious subjects for consideration are discussed at this time.

10.1.1 Land Use

The area has been used in the past for mining projects and there are small operating mines in the general Madoc region. The type of mining envisioned herein should not be incompatible with current recreational and residential uses.

Existing land uses and zoning practices in the Bannockburn area should be documented. If required, rezoning applications will have to be made to transfer appropriate lands to industrial use. Such applications should be made to the Madoc Township Council.

10.1.2 Anticipated Operating Conditions

Air Quality - Changes to air quality should be minimal, the only discharges being from ventilation of the mine and mill, and from mobile equipment. Diesel equipment used underground will be equipped with exhaust scrubbers.

Water Quality - A water quality monitoring program will be required in which upstream quality can be compared to downstream quality to determine any effluent impact on the Moira River. Plant and tailings effluents will be required to meet specified chemical criteria; regular sampling will be necessary.

Noise - Because the project is close to Bannockburn, the main focus of environmental restrictions probably will be on noise. Baseline studies and monitoring likely will be required in relation to machinery operation, underground blasting (at the nearest critical receptor - nearest house), and increased highway traffic. Mufflers on mobile equipment and noise barriers around stationary equipment will be required, both of which are normal practices at any industrial operation.

10.2 SOCIO-ECONOMIC IMPACT

A socio-economic impact study should not be necessary, but indications of payroll, skilled and unskilled labour requirements, and potential for use of local services and suppliers should be useful information for local authorities.

There will be no site accommodation for the workforce.

10.3 REQUIRED PERMITTING

A list of permits that may be required, classified by the Ministry responsible for each, is given below. Approval time for each one increases with the complexity of the aspects being approved and can range from three to nine months. Careful scheduling will be required.

A. Ministry of Environment

(i) Liquid Industrial Waste Permit - Tailings Pond
(ii) Sewage Works
(iii) Water Works Permit (if above 50,000 litres per day)
(iv) Air Emissions

(v) Solid Waste Disposal Site - Landfill

(vi) Water Take Permit - for surface and underground water requirements

B. Ministry of Natural Resources

(i) Powerline

(ii) Work Permit - District Cutting Licence

(iii) Tailings Dam - location approval, structural approval

(iv) Solid Waste Disposal - Land use permit

C. Ministry of Labour

(i) Notice of Project

(ii) Health and Safety Matters

11.

COST ESTIMATES

11. COST ESTIMATES

11.1 CAPITAL COSTS

11.1.1 Basis of Estimates

Estimates are based on conceptual mine design, preliminary flowsheet, and services and facilities appropriate to the operation. This applies to equipment sizing and quantities as well.

Prices are current (1988). Verbal quotations were solicited from suppliers for most items where prices were not available from other recent experience. Some used equipment has been considered, particularly where several similar pieces of equipment are required. Mine development costs are a combination of those experienced during recent contracted work at the Bannockburn property and estimates for a continuous operating mine.

Installation costs for the mill are based on experience with other such projects and have been calculated as 1.2 times the mill equipment costs. For a modular design, this multiplier could be lowered to a minimum of 0.9.

Because the estimating has not been done in great detail, provision has been made for spare parts and miscellaneous items, within each major cost area, by adding 10-20 percent, as deemed appropriate, to cover these costs.

Engineering, procurement and construction management appear low because, for the most part, they are included in the mill cost estimate.

11.1.2 Comments on Major Cost Items

Mine

(1) Heavy Equipment - Includes a haulage truck (26-ton), all major underground equipment, a scooptram (2-yd) and spare parts. The scooptram will be required for clean-up under chutes in the decline, for moving large pieces of equipment from surface and between levels, and for moving muck on surface (in the latter case, in lieu of a front-end loader).

(2) Small Powered Equipment - Pumps, fans, rock drills, slushers, powered tools, etc.

(3) Linear Goods - Ventilation ducting, air and water pipes and hoses, rail, switches. Most of these items are costed under operating costs for development and mining; the amounts included here are for start-up and to establish initial stocks.

(4) Small Tools - Hand tools, mechanics tools, first-aid equipment and other small items.

(5) Tanks and Receivers - For the mine only.

(6) Miscellaneous Items - Chute hardware, ore pass control chains, explosives and cap magazines, initial supplies of drilling, blasting and ground support materials.

(7) Capital Development - Includes driving the Decline from the 2nd to the 5th Level, main cross-cuts on the 3rd, 4th and 5th Levels, Ventilation Raise from the 5th Level to surface, and the Ore Pass from 5th to 2nd Level.

(8) Preproduction Development - Preparing the mine for sustained production when mill operation commences. Includes limited raising, developing five stopes, and test mining of two stopes.

Mill

(1) Milling - Crushing, grinding, screening, duplex jig. Basis of crusher operation is 10 hours per day for six days per week. The ball mill is about 20 percent overcapacity to allow for harder grinding or increased tonnage.

(2) Leaching - Thickeners, preaeration, leaching tanks and mechanisms.

(3) Filtration and Precipitation - Pumps, tanks, rotary drum vacuum filter, Merrill-Crowe system.

(4) Laboratory and Refinery - All necessary equipment and controls.

(5) Installation Cost - Engineering, procurement, construction.

Ancillary Services and Facilities

(1) Electrical - 44 kV line terminal, substation, distribution, standby diesel generator, miscellaneous items.

(2) Vehicles and Surface Equipment - Three pick-up trucks, flat-deck with crane, main ventilation fan and heater for mine, welder, etc.

(3) Buildings - Office, maintenance, lamp room, core storage, dry facilities.

(4) Tailings - Tailings dam (from mine waste), ancillary items. (It is proposed to transport the tailings in a dry, rather than slurry, state.)

(5) Miscellaneous Items - Fuel tanks, initial fuel and fabrication supplies, water and sewage systems.

Administration

(1) Environmental Studies - A recognition that some studies will be required.

(2) Equipment and Supplies - Office and engineering.

(3) Management - Project management, EPC other than mill.

(4) Miscellaneous - Provincial sales tax and freight which are significant costs.

11.1.3 Summary of Capital Costs

Only major areas of cost are shown here because the present state of the project does not warrant detailed costs.

Mine		
Heavy Equipment	$ 602,000	
Small Powered Equipment	369,000	
Linear Goods	128,000	
Small Tools	105,000	
Tanks and Receivers	14,000	
Miscellaneous Items	285,000	
Capital Development	2,167,000	
Preproduction Development	293,000	$ 3,963,000
Mill		
Milling	662,000	
Leaching	490,000	
Filtration and Precipitation	585,000	
Laboratory and Refinery	650,000	
Installation	2,865,000	5,252,000
Ancillary Services and Facilities		
Electrical	656,000	
Vehicles and Surface Equipment	490,000	
Buildings (other than mill)	125,000	
Tailings	550,000	
Miscellaneous Items	176,000	1,997,000

Administration		
Environment	50,000	
Equipment and Supplies	105,000	
Management	460,000	
Miscellaneous	646,000	1,261,000
Contingency (Approx. 10%)		1,247,000
TOTAL CAPITAL COST - BANNOCKBURN		$13,720,000

11.2 OPERATING COSTS

11.2.1 Basis of Estimate

At this stage of the Bannockburn project, operating costs cannot be estimated with great accuracy. They are related to conceptual or preliminary designs and uncertainties regarding mining and milling conditions.

Equipment operating costs have been reasonably defined. Consumables for the mine and mill (reagents, power, blasting agents, steel, etc.) have been estimated mostly from experience with other mining operations. Labour rates used are those for mines in southern Ontario; payroll load is 35%.

Productivity in the mine is the area of greatest uncertainty. Again, factors used are those common to similar mines.

The general parameters employed are:

Operating days per year		350
Work schedule:	Mill - continuous; 7 days/week	
	Mine - 2 shifts, 5 days/week	
Daily throughput:	Mill	200 short tons
	Mine	280 short tons
Annual Production		70,000 short tons

Because the cost of connection to the 44 kV line is uncertain at this time, no capital cost has been included for this item. Instead, a small surcharge has been applied to the power operating cost.

11.2.2 Summary of Operating Costs

Cost Area	Cost Per Ton	
Exploration		
Surface	$ 0.50	
Underground	3.00	$ 3.50
Mining		
Supervision and Staff	4.00	
Development	12.00	
Stoping	18.50	
Main Haulage	1.00	
Level Haulage	6.00	
Power	1.50	
General Mine Expense	2.00	45.00
Milling		
Supervision and Labour	24.30	
Reagents and Supplies	4.60	
Cyanide Destruction	2.50	
Power	2.50	33.90
Plant		
Access and Site Maintenance	0.50	
Surface Plant and Services	3.00	
Other	0.50	4.00
Administration		
Head Office	1.00	
Site Office	3.00	4.00
TOTAL OPERATING COST PER SHORT TON		$ 90.40

12.

ECONOMIC ANALYSIS

12. ECONOMIC ANALYSIS

Although the Bannockburn property is in an advanced state of exploration, it is still in the preliminary stages of mine evaluation. The nature of the deposit, combined with the limited amount of data from underground work, precludes confident economic estimates with the usual presentation of cash flows. The proportion of reserves assigned to the lower confidence classifications of Indicated and Inferred (69.3% and 24.8% respectively) is a reflection of this problem. The critical aspect of mineable grade versus current reserve grade has still be to resolved. Other factors are also difficult to assess at this time: nugget effect, overall reserve potential, mining costs, etc. However, a series of sensitivity analyses (Appendix I) have been done to indicate the effect of positive results on the potential viability of the property.

12.1 BASE CASE PARAMETERS

The base case parameters used for the sensitivity analyses are:

Gold price	$450 (U.S.)
Currency exchange rate	$1.00 (U.S.) = $1.20 (Cdn.)
Reserve tonnage	500,000 tons
Reserve grade	0.186 oz gold per ton
Production rate	200 tons per day
Operating days per year (Mill)	350 days
Development period	9 months
Metallurgical Recovery	98%
Refinery payment	99%
Capital Costs	$13,720,000
Operating Costs	$90.40 per ton
Cost of money	12%
Discount rate	15%
Debt retirement rate	100% of profit
Equity financing	$0

In addition, for the first three months of operation, operating costs have been increased (115%, 110%, 105%), and metallurgical recovery has been decreased (85%, 90%, 95%) from base case parameters, to reflect common start-up problems and inefficiencies.

A reserve of 500,000 tons has been used in order to impart more realism into the analysis. As noted in Section 13, small underground mines commonly cannot be economically justified on the basis of initial reserves; potential reserves also must be included to obtain more realistic evaluations.

One sensitivity analysis, using grade versus gold price, has been done at zero percent discount rate to indicate conditions that produce a positive cash flow after debt retirement (although not necessarily an acceptable return on investment).

12.2 SENSITIVITY ANALYSES

The sensitivity analyses results, for the most part, are quite negative at the base case parameters. However, they do indicate the effect of positive changes to some of the important economic factors. By far the most significant change is an increase in grade. Also, lowering operating costs by increasing the rate of production produces significant economic improvements.

Increasing the grade from the ore reserve average of 0.186 oz gold per ton is feasible. First, with no change in calculation parameters, the reserve grade is expected to increase because of the up-grading processes discussed in Section 6.3. Second, the cut-off grade can be increased somewhat without severely affecting the total reserve tonnage (Section 6.7.4), In combination, it is expected that an average reserve grade of about 0.25 oz gold per ton can be achieved.

A decrease in operating costs can be realized by increasing mining productivity and/or daily production. Mining productivity will increase if some larger (wider) stopes can be developed; local stockworks could be important in this regard. Overall productivity for the operation will be improved by a larger daily throughput because many fixed costs will show little or no corresponding increase, i.e. general overhead, mill and plant operating personnel, and so forth. The combination of increasing the average mineable grade and of decreasing operating costs (and increased daily production), both of which should be attainable goals, could result in an economically viable mine on the Bannockburn property.

Another factor, which is not dependent on property characteristics, but which has a profound effect on profitability, is equity financing. Three sensitivity analyses show the marked differences in project economics for $0, $7,000,000, and $13,720,000 equity financing.

To demonstrate the result of some of these positive changes, an analysis has been done using some hypothetical parameters. Base case parameters have been used except for: reserve grade of 0.25 oz gold per ton, 250 tons per day production, $85 per ton operating costs, and $13,000,000 capital costs. The Net Present Value at 15% DCF is -$2,219,000; at 0% DCF it is $4,763,000. The Rate of Return (before taxes) is 8.5%.

13.

CONCLUSIONS

13. CONCLUSIONS

A number of significant conclusions have evolved from this study of the Bannockburn property. They form the basis of the proposed feasibility program outlined in Section 14.

The most important conclusions are:

(1) The Bannockburn deposit is not economically feasible at current costs, gold price and reserves. However, it clearly exhibits potential to be economically feasible if some of the determining factors can be enhanced; furthermore, there is good reason to believe that some of the necessary enhancement can be achieved by further exploration and testing of the deposit.

(2) The reserve grade, as determined almost completely from diamond drilling results, is expected to increase significantly, particularly for the lower grade reserve blocks, when calculated from chip and muck sampling in the mineralized zones. This factor should have a major effect on the potential viability of the Bannockburn deposit.

(3) Some previous assaying problems due to the 'nugget effect' of free gold have resulted in a number of vein intersections being of apparently lower than actual grade (particularly in the North Area). Confirmatory testing by diamond drilling or mining is expected to result in up-grading of these intersections, and thereby to increase the reserve grade slightly and to add new ore reserve blocks.

(4) The probability of locating additional reserves is good. There are several areas that should be explored:

(i) Within the drilled areas, but most specifically in the North Area, for reasons noted in (3) above.

(ii) Downward extensions of veins in the Core and North areas; probably the best near-term source of reserves (and the most easily developed from the present workings).

(iii) New, possibly 'blind' zones 'stratigraphically' below the Core/North zones.

(iv) Extensions to the north and south of Core/North zones.

(v) Other, untested areas of the Bannockburn property.

(5) There are some indications of local stockworks in the Core Area; they could provide additional reserves and, more importantly, significantly lower production costs.

(6) A material increase in reserves could lead to an increase in the proposed daily production rate and a consequent decrease in total operating costs.

(7) Most of the exploration must be done by drifting and raising on veins. More intense diamond drilling will not confirm the reserves nor give accurate indications of mineable grade or vein continuity. However, diamond drilling will disclose new veins and vein zones in unexplored or poorly tested areas.

(8) Bearing in mind that for vein-type underground mines it is common to locate and mine ore for many years beyond initial reserve indications, it is suggested that equity financing be considered as a means of commencing operations before estimated returns are of a sufficient confidence level to satisfy complete debt financing.

14.

PROPOSED FEASIBILITY PROGRAM

14. PROPOSED FEASIBILITY PROGRAM

The program outlined below could provide sufficient positive results to justify a production decision. However, it is more likely that some further work will be required to fully satisfy such a decision. Such work may not be extensive but will be necessary to fill-in information gaps and clean up 'loose-ends'.

14.1 OBJECTIVES OF PROGRAM

(1) Confirm the up-grading and determine the amount of up-grading, of the drill-defined reserves, that can be expected during the course of mining operations. The comprehensive objective is to raise the average mine grade, as calculated with present reserve parameters, by being able to apply a grade enhancement factor, particularly to the lower grade drill-defined reserve blocks.

(2) Increase the reserve tonnage by:

(i) Drifting, raising and drilling on the 2nd Level

(ii) Drilling from underground and surface for downward extensions of the Core/North zones

(iii) Drilling to explore the 'Lower' Zone

(iv) Limited surface drilling elsewhere on the property, especially the northern and southern projections of the Core/North zones

(3) Enhance the reserve confidence levels, ie. transfer reserves from Inferred to Indicated to Possible/Probable/Proven categories.

(4) Confirm the North Area reserves which are based solely on diamond drill intersections.

(5) Check low grade intersections for which visible gold has been noted (resulting from the nugget effect on assaying), mostly in the North Area.

(6) Explore, by drifting and slashing, the possibility of mineable grade stockworks.

(7) Test the strike and dip continuity of veins.

14.2 WORK REQUIRED

14.2.1 Surface Exploration

Prospecting and geological mapping should be done on all areas of the property where such work has not previously been done. Any mineralized veins located should be trenched or stripped.

Diamond drilling is required to test the northern and, more particularly, the southern extensions of the Core/North zones, to explore the downward continuity of the Core/North zones, and to look for zones parallel to the Core/North zones (such as the 'Lower' Zone intersected in earlier drilling).

14.2.2 Underground Exploration

The proposed underground exploration and development program is designed to pursue the objectives set out in Section 14.1. This can be accomplished by extending the existing underground workings and by raising and drilling from the exploration drifts.

Underground exploration will consist of the following categories and approximate quantities:

		Feet
(i)	Main access, 11 ft x 15 ft	
	Extend decline to second level	800
	Drive crosscut on second level	200
(ii)	First level, Core Area	
	Drifting on veins (7 ft x 6 ft)	600
	Raising to surface (7 ft x 5 ft)	400
(iii)	Second level, Core Area	
	Drifting on veins (7 ft x 6 ft)	500
	Raising to 2nd level (7 ft x 5 ft)	400
(iv)	Second level, North Area	
	Access drift and crosscut (7 ft x 6 ft)	800
	Drifting on veins (7 ft x 6 ft)	500
	Raising to surface or blind (7 ft x 5 ft)	400
	Totals: 11 x 15 decline and crosscut	**1,000**
	7 x 6 drift	**2,400**
	7 x 5 raise	**1,200**

The decline will be extended on an 11 ft x 15 ft cross-section to the second level, roughly 175 ft from surface. An 11 ft x 15 ft crosscut will be driven for 200 feet across the mineralized veins as projected from the first level. A stub decline will be driven below the second level to allow for truck haulage of muck produced on the second level. Sumps and diamond drill bays will be cut from the decline as required.

Drifting on the first level is intended to increase the grade of drill-indicated reserves. In particular, it is intended to investigate the possibility that an ore zone of multiple veins exists in the vicinity of the veins now known as No. 1 and No. 2.

Raising from the first level is intended both to test the dip, variability of dip, and continuity of the veins found in the Core Area and to provide ventilation and escape ways to surface.

Drifting on the second level is intended to confirm and investigate the downward extension of the Core Area and to give access to the drill-indicated North Area some 500 feet distant.

Raising from the second level has the same purposes as raising from the first level.

Diamond drilling will be done from the second level in the Core and North areas as a means of more closely defining the veins prior to vein drifting and raising, and to explore below the second level for extensions of present zones and for new parallel zones.

14.2.3 Metallurgical Testing

Further metallurgical testing is necessary in order to confirm the suitability of the cyanide recovery process on the Bannockburn ore. It should be done on underground samples taken from several locations; a range of grades should be tested as well. Other characteristics to be checked are Work Index and Filterability.

14.2.4 Permitting and Approvals

Application for some permits and approvals should get underway as soon as the required information is available. These matters can take considerable time and thereby cause delays in project work and additional costs.

14.2.5 Feasibility Report

Results of the feasibility work will have to be compiled and presented in a feasibility report. This will take three to six months after the underground exploration has been completed.

14.3 ESTIMATED COSTS

Costs have been estimated in some detail to arrive at the summary presented below. They have been estimated on the basis of charge-out rates for equipment, costs of services and supplies, applicable labour rates, and some profit for a contractor. Results of these estimates have been converted to costs per foot where applicable. However, although values are assigned to the major sections of the proposed work program, it does not follow that the overall project costs can be reduced by a corresponding amount if a section is deleted. For example, some distributable costs such as generator rental, are time dependent rather than footage dependent; a reduction in footage that is not on the critical path will not reduce the generator rental time or cost.

Cost Category			Cost
Decline and Crosscut			$ 900,000
Drifts:	Core Area - 1st Level	$ 400,000	
	Core Area - 2nd Level	350,000	
	North Area - 2nd Level	900,000	1,650,000
Raises:	Core Area - 1st Level	280,000	
	Core Area - 2nd Level	280,000	
	North Area - 2nd Level	280,000	840,000
Diamond Drilling:	Surface	200,000	
	Underground	300,000	500,000
Support (including assaying)			100,000
Consulting, Engineering, Supervision			100,000
Contingency (approx. 10%)			410,000
TOTAL ESTIMATED COST			**$ 4,500,000**

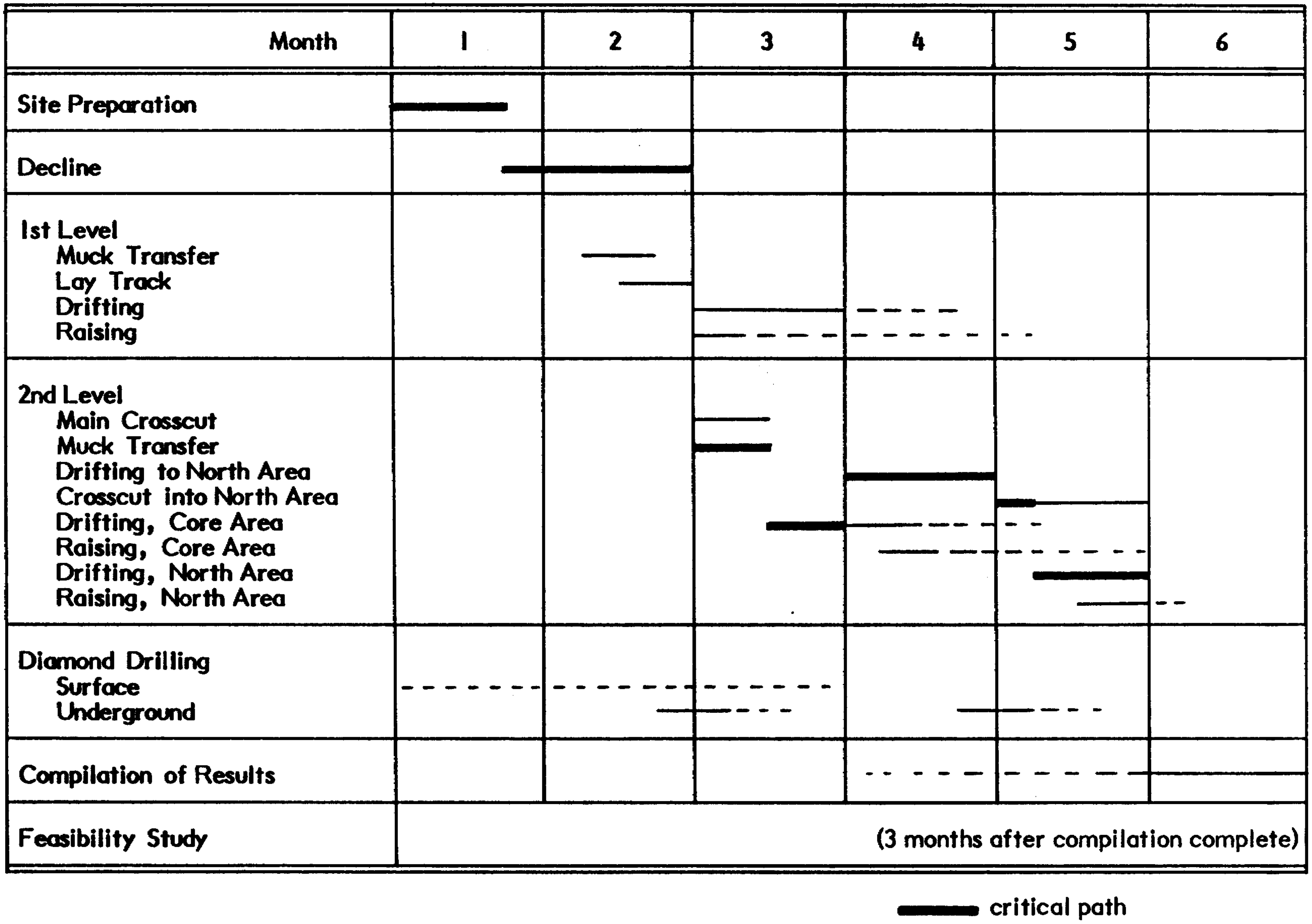
14.4 SCHEDULE

The probable schedule for the proposed feasibility program is laid out on Figure 14 (the critical path is marked). Although there is some possibility that some of the items on the critical path can be accelerated, the overall effect would not be very great and it would be unwise to predict markedly shorter times than those indicated on the bar chart.

Of particular importance is the collection and collation of the results of sampling and mapping at the end of the underground work. Failure to do this promptly and efficiently can have serious effects on the time needed to obtain a final report.

Micham Exploration Inc. - Bannockburn Project

FIGURE 14: WORK SCHEDULE - STAGE 2 UNDERGROUND



APPENDICES

APPENDIX I

SENSITIVITY ANALYSES

Micham Exploration Inc. - Bannockburn Project

SENSITIVITY ANALYSES
(Net Present Values)

Capital Costs	Reserve Grade 0.186	0.200	0.225	0.250	0.275	0.300
$11,000,000	($15,427,266)	($12,756,512)	($7,829,567)	($2,996,536)	$1,342,226	$5,181,852
$11,500,000	($16,205,850)	($13,535,097)	($8,608,152)	($3,739,679)	$687,669	$4,570,070
$12,000,000	($16,984,435)	($14,313,682)	($9,386,737)	($4,499,080)	$23,872	$3,951,534
$12,500,000	($17,763,020)	($15,092,267)	($10,165,321)	($5,272,765)	($649,913)	$3,325,712
$13,000,000	($18,541,605)	($15,870,852)	($10,943,906)	($6,053,075)	($1,334,286)	$2,692,326
$13,500,000	($19,320,190)	($16,649,437)	($11,722,491)	($6,831,659)	($2,029,665)	$2,049,081
$13,720,000	($19,662,767)	($16,992,014)	($12,065,068)	($7,174,237)	($2,338,578)	$1,764,131
$14,000,000	($20,098,775)	($17,428,022)	($12,501,076)	($7,610,244)	($2,735,455)	$1,399,425
$14,500,000	($20,877,360)	($18,206,606)	($13,279,661)	($8,388,829)	($3,452,154)	$742,162

DCF @ 15%; Equity Financing = $0

Operating Costs	Reserve Grade 0.186	0.200	0.225	0.250	0.275	0.300
$75.00	($13,735,510)	($10,992,845)	($6,028,354)	($1,483,114)	$2,565,845	$6,451,273
$77.50	($14,698,532)	($12,012,252)	($7,040,835)	($2,362,143)	$1,867,833	$5,689,881
$80.00	($15,676,480)	($12,952,413)	($8,053,317)	($3,258,018)	$1,057,954	$4,925,748
$82.50	($16,624,304)	($13,909,023)	($8,931,849)	($4,044,080)	$232,244	$4,168,824
$85.00	($17,600,171)	($14,880,898)	($9,951,256)	($5,006,893)	($607,013)	$3,485,083
$87.50	($18,547,613)	($15,866,850)	($10,874,982)	($6,011,821)	($1,464,424)	$2,693,995
$90.00	($19,505,144)	($16,823,591)	($11,900,919)	($7,019,436)	($2,315,101)	$1,894,229
$90.40	($19,662,767)	($16,992,014)	($12,065,068)	($7,174,237)	($2,338,578)	$1,764,131
$92.50	($20,440,863)	($17,774,449)	($12,848,096)	($7,890,261)	($3,095,449)	$1,080,805

DCF @ 15%; Equity Financing = $0

Micham Exploration Inc. - Bannockburn Project

SENSITIVITY ANALYSES
(Net Present Values)

Reserve Tonnage	Reserve Grade					
	0.186	0.200	0.225	0.250	0.275	0.300
200,000	($15,732,400)	($14,612,946)	($12,520,452)	($10,435,873)	($8,278,590)	($6,209,793)
300,000	($17,120,280)	($15,464,434)	($12,391,873)	($9,336,977)	($6,173,644)	($3,149,332)
400,000	($18,406,067)	($16,237,624)	($12,228,472)	($8,246,304)	($4,129,499)	($470,121)
500,000	($19,662,767)	($16,992,014)	($12,065,068)	($7,174,237)	($2,338,578)	$1,764,131
600,000	($20,862,323)	($17,719,183)	($11,929,119)	($6,184,474)	($925,606)	$3,532,154
700,000	($22,082,022)	($18,448,059)	($11,761,187)	($5,232,272)	$339,442	$5,110,049
800,000	($23,186,694)	($19,113,191)	($11,623,217)	($4,524,265)	$1,284,906	$6,292,969
900,000	($24,244,291)	($19,743,742)	($11,473,493)	($3,926,611)	$2,079,083	$7,283,669
1,000,000	($25,271,520)	($20,359,174)	($11,336,508)	($3,444,770)	$2,722,322	$8,088,306

DCF @ 15%; Equity Financing = $0

Micham Exploration Inc. - Bannockburn Project

SENSITIVITY ANALYSES
(Net Present Values)

Gold Price	Reserve Grade					
	0.186	0.200	0.225	0.250	0.275	0.300
$400	($22,882,059)	($21,201,598)	($16,992,014)	($12,557,735)	($8,171,562)	($3,868,050)
$425	($21,462,290)	($19,154,890)	($14,512,004)	($9,877,968)	($5,213,806)	($1,017,712)
$450	($19,662,767)	($16,992,014)	($12,065,068)	($7,174,237)	($2,338,578)	$1,764,131
$475	($17,643,414)	($14,802,067)	($9,593,567)	($4,397,230)	$159,487	$4,419,996
$500	($15,617,834)	($12,557,735)	($7,174,237)	($1,968,343)	$2,661,463	$6,887,894
$525	($13,580,679)	($10,446,770)	($4,666,341)	$392,112	$5,068,094	$9,391,258
$550	($11,520,400)	($8,171,562)	($2,338,578)	$2,661,463	$7,307,283	$11,864,167
$575	($9,493,503)	($6,057,647)	($309,115)	$4,853,204	$9,600,532	$14,305,420
$600	($7,374,694)	($3,868,050)	$1,764,131	$6,887,894	$11,864,167	$16,721,490

DCF @ 15%; Equity Financing = $0

Gold Price	Reserve Grade					
	0.186	0.200	0.225	0.250	0.275	0.300
$400	($5,670,828)	($3,990,367)	$219,217	$4,583,878	$8,233,163	$11,568,709
$425	($4,251,059)	($1,943,659)	$2,699,227	$6,885,081	$10,525,249	$13,959,653
$450	($2,451,536)	$219,217	$5,031,341	$9,040,247	$12,801,303	$16,367,393
$475	($432,183)	$2,409,164	$7,114,745	$11,152,510	$14,967,088	$18,747,746
$500	$1,593,397	$4,583,878	$9,040,247	$13,156,851	$17,165,066	$21,058,325
$525	$3,627,239	$6,420,994	$10,944,933	$15,167,423	$19,336,682	$23,399,439
$550	$5,503,630	$8,233,163	$12,801,303	$17,165,066	$21,447,423	$25,733,196
$575	$7,194,390	$9,891,637	$14,564,027	$19,140,370	$23,593,898	$28,063,853
$600	$8,844,763	$11,568,709	$16,367,393	$21,058,325	$25,733,196	$30,384,595

DCF @ 15%; Equity Financing = $7,000,000

Micham Exploration Inc. - Bannockburn Project

SENSITIVITY ANALYSES
(Net Present Values)

Gold Price	Reserve Grade					
	0.186	0.200	0.225	0.250	0.275	0.300
$400	$10,412,744	$12,089,525	$15,138,660	$18,187,795	$21,236,931	$24,286,066
$425	$11,832,513	$13,614,092	$16,853,799	$20,093,505	$23,333,211	$26,572,918
$450	$13,252,282	$15,138,660	$18,568,937	$21,999,215	$25,429,492	$28,859,769
$475	$14,672,051	$16,663,228	$20,284,076	$23,904,924	$27,525,772	$31,146,621
$500	$16,091,820	$18,187,795	$21,999,215	$25,810,634	$29,622,053	$33,433,472
$525	$17,511,589	$19,712,363	$23,714,353	$27,716,343	$31,718,334	$35,720,324
$550	$18,931,358	$21,236,931	$25,429,492	$29,622,053	$33,814,614	$38,007,175
$575	$20,351,126	$22,761,498	$27,144,631	$31,527,763	$35,910,895	$40,294,027
$600	$21,770,895	$24,286,066	$28,859,769	$33,433,472	$38,007,175	$42,580,878

DCF @ 15%; Equity Financing = $13,720,000

Gold Price	Reserve Grade					
	0.186	0.200	0.225	0.250	0.275	0.300
$400	($34,560,738)	($31,348,823)	($22,702,131)	($13,643,751)	($4,695,598)	$4,012,899
$425	($31,850,665)	($27,129,195)	($17,634,863)	($8,179,473)	$1,332,946	$9,533,523
$450	($28,171,993)	($22,702,131)	($12,641,828)	($2,664,747)	$6,985,659	$14,805,009
$475	($24,032,401)	($18,228,214)	($7,598,827)	$2,968,909	$11,773,120	$19,802,148
$500	($19,893,699)	($13,643,751)	($2,664,747)	$7,711,854	$16,494,948	$24,471,382
$525	($15,734,716)	($9,340,765)	$2,433,014	$12,214,094	$21,019,517	$29,180,010
$550	($11,529,206)	($4,695,598)	$6,985,659	$16,494,948	$25,260,428	$33,832,765
$575	($7,394,533)	($386,419)	$10,883,422	$20,616,025	$29,573,334	$38,430,154
$600	($3,068,674)	$4,012,899	$14,805,009	$24,471,382	$33,832,765	$42,983,510

DCF @ 0%; Equity Financing = $0

APPENDIX II

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REFERENCES

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1988 House, G.D.; Compilation Report on the Bannockburn Property, Madoc Township, Hastings County, Southern Ontario Mining Division; by Sawyer Consultants Inc. for Mono Gold Mines Inc., 17 March, 1988.

APPENDIX III

ACKNOWLEDGEMENTS

ACKNOWLEDGEMENTS

The following people have been directly involved in the prefeasibility study:

C. R. Saunders, P.Eng., geological engineer - project co-ordinator
Reserves, mining, cost estimates and analyses, general subjects.

R. S. Adamson, P.Eng., geological engineer - alternate project co-ordinator
Geology, mineral deposits, general subjects

T. A. Morrison, P.Eng., mining engineer
Mining, cost estimates, ancillary services, planning

J. W. Fisher, P.Eng., metallurgist
Milling, cost estimates

D. S. Rogers, geologist
Reserves

Micham Exploration personnel, both at the mine site and in the Toronto head office, provided most of the data used in this study. They were very helpful in getting additional or missing information and in expediting the study through good cooperation with Orcan personnel.

APPENDIX IV

CERTIFICATES

CERTIFICATE

I, C. Raymond Saunders of 666 St. Ives Crescent, North Vancouver, Canada, do hereby certify that:

1. I am a graduate of the University of British Columbia, (B.A.Sc. in Geological Engineering, 1956).

2. I am a registered Professional Engineer of the Province of British Columbia (registration number 6498).

3. From 1956 until 1967, I was engaged in mining and mining exploration in Canada for a number of companies; positions included mine geologist, mine engineer and and chief geologist for underground and open pit operations. Since 1967 I have been practicing as a consulting geological engineer in minerals exploration, property development and deposit evaluation in Canada and other countries.

4. I have examined the Bannockburn property reported upon herein.

5. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the properties or securities of Micham Exploration Inc. or any associate or affiliate of Micham Exploration Inc.

6. I do not have a direct or indirect interest in, nor do I beneficially own, directly or indirectly, any securities of Micham Exploration Inc. or any associate or affiliate of Micham Exploration Inc.

Respectfully submitted,

PROFESSIONAL PROVINCE OF C. R. SAUNDERS BRITISH COLUMBIA ENGINEER

Vancouver, Canada

C. Raymond Saunders, B.A.Sc., P. Eng.

CERTIFICATE

I, **Robert S. Adamson**, with business and residential addresses in Vancouver, British Columbia, do hereby certify that:

1. I am a consulting geological engineer.

2. I am a graduate of the University of British Columbia, (B.A. Sc. in Geological Engineering, 1957).

3. I am a registered Professional Engineer of the Province of British Columbia.

4. From 1957 until 1967, I was engaged in mineral exploration in Canada for a number of companies. Positions included Senior Geologist, Chief Geologist, and Vice-President, Exploration. Since 1967 I have been practising as a consulting geological engineer and, in this capacity, have examined and reported on numerous mineral properties in Africa, Europe, and North and South America.

5. I have not examined the Bannockburn property reported on herein.

6. I have not received, directly or indirectly, nor do I expect to receive any interest, direct or indirect, in properties of Micham Exploration Inc. or any affiliate thereof, nor do I beneficially own, directly or indirectly, any securities of Micham Exploration Inc. or any affiliate thereof.

Respectfully submitted,

Vancouver, Canada

Robert S. Adamson, B.A.Sc., P.Eng.

CERTIFICATE

I, Thomas A. Morrison, of 375 - 1440 Garden Place, Delta, British Columbia, Canada, do hereby certify that:

1. I am a graduate of the Camborne School of Mines, England (ASCM, 1976).

2. I am a registered Professional Engineer of the Province of British Columbia (registration number 14007).

3. From 1976 to 1986 I was engaged in underground mining, tunnelling, and other aspects of the mineral industries in Canada and the United States, in both the private and public sectors. Positions included shiftboss, mine captain, engineer, contract administrator, and project manager. Since 1986, I have been practising as a consulting mining engineer in Canada.

4. I have examined the Bannockburn property reported upon herein.

5. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the properties or securities of Micham Exploration Inc. or any associate or affiliate of Micham Exploration Inc.

6. I do not have a direct or indirect interest in, nor do I beneficially own, directly or indirectly, any securities of Micham Exploration Inc. or any associate or affiliate of Micham Exploration Inc.

Respectfully submitted,

PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
T. A. MORRISON

Vancouver, Canada

Thomas A. Morrison, ACSM, P. Eng.

ORCAN MINERAL ASSOCIATES LTD.
CONSULTING ENGINEERS

SUITE 1417 - 409 GRANVILLE STREET
VANCOUVER, CANADA V6C 1T2
TELEPHONE (604) 682-3722

12 July, 1988

Mr. David R. Bell
President
Micham Exploration Inc.
104 - 18 King Street, East
Toronto, Ontario
M5C 1C4

Dear Mr. Bell:

Re: Preliminary Results from the Current Engineering Study of the Bannockburn Project, Madoc, Ontario

The purpose of this letter is to present the preliminary findings and estimates of the Engineering Evaluation that is currently underway on the Bannockburn property. The estimates presented should be comparable within ten percent to the figures that will be in the final report. The reserves are essentially fixed; there should be no significant variations in the final report.

The Bannockburn Project comprises a Precambrian vein-type gold deposit situated in southern Ontario. Exploration results suggest that it could have the potential to support a small (eg. 200 tpd) underground mining operation with economics comparable to similar well-located gold mines. The veins strike predominantly north-south and dip 35°-60° east.

Exploration to date has consisted of surface diamond drilling, ramping down to the 75 ft. level in one area, and initial diamond drilling and drifting on this level.

Reserves

Reserves have been calculated mostly from diamond drilling results but with some support from underground sampling (chip and muck sampling). The reserves are classified as 'Modified Geological' (except for some 15,000 tons of Probable) because,

(1) some mining parameters have been employed in making the calculations, but (2) for the most part the reserves have not been verified by underground openings.

Using a minimum mining width of six feet, the cutting of individual high grade assays to 3.0 oz Au per ton, and a zone cut-off of 0.05 oz Au per ton, the total estimated reserves are 248,000 short tons at a cut grade of 0.186 oz Au per ton. Uncut grade is 0.267 oz Au per ton. The decision to cut individual high grade assays to 3.0 oz Au per ton is based on an analysis of all drill hole assays of 0.01 oz Au per ton and greater. For zone cut-off grades of 0.05 (the reserve grade basis), 0.08 and 0.10 oz Au per ton, the results are:

Cut-off Grade	Reserve Tonnage	Cut Grade oz Au/t	Uncut Grade oz Au/t
0.05	248,000	0.186	0.267
0.08	211,000	0.208	0.302
0.10	187,000	0.224	0.330

From the limited drifting done to date, it appears that mining grades, at least for the lower grade zones, will be higher than the grades calculated from diamond drilling results. In addition, an assaying problem involving several hundred samples from the North Area appears to have caused the downgrading of over 100 possible reserve grade assays.

Objectives of Further Exploration

Exploration to date indicates the presence of potentially economic gold mineralization on the Bannockburn property. Most of the mineralization (gold-bearing quartz veins) is present in two closely adjacent areas, the 'Core Area' and the 'North Area'; only the Core Area has been tested underground in a limited manner. The apparent absence or paucity of gold mineralization in other areas may be partly due to restricted exploration.

The recognized economic potential of the Bannockburn property dictates that comprehensive exploration be undertaken. The objectives of this exploration are:

(1) to verify and, thereby, upgrade the classification (to mineable categories) of a major portion of the present reserves; (2) to possibly increase the reserves in the Core and North areas; (3) to locate additional reserves peripheral to these areas; (4) to explore for other mineralized areas in the general vicinity of the present zones. In the last case, there are indications from previous exploration results of at least one such set of veins 'stratigraphically' below the Core Area mineralization.

Proposed Exploration Program

To pursue the objectives set out above, the following exploration work is proposed:

(1) Core Area

(a) 75 Level - further drifting on veins; drifting and slashing of possible stockwork zones; raising.

(b) 175 Level - ramp down from the 75 Level; diamond drill; drift and raise on veins.

(2) North Area - drift from the Core Area on the 175 Level for approximately 500 ft. to get into the North Area; diamond drill; initial drifting and raising.

(3) Surface Drilling

(a) Close to the present zones to determine lateral and dip extensions.

(b) Peripheral drilling to locate other mineralized areas.

(4) Ancillary Studies - metallurgical testing, preliminary environmental surveys, services requirements, etc.

Cost and Time Estimates

Cost estimates and scheduling are based on the following preliminary work quantities:

Decline and main 175 Level cross-cut	1,200	ft.
Drifting	3,000	ft.
Raising	1,500	ft.
Diamond drilling (surface and underground)	20,000	ft.
Ancillary studies		

The initial estimate of total cost, including engineering, overhead, contingency and so forth, is \$4,500,000.

The on-site time required to complete this program will be a minimum of six months.

If this program is at least moderately successful, a substantial portion of the physical work necessary for a feasibility study will have been completed.

Respectfully submitted,

ORCAN MINERAL ASSOCIATES LTD.

PROFESSIONAL PROVINCE OF C. R. SAUNDERS BRITISH COLUMBIA ENGINEER

C. Raymond Saunders, P.Eng.

PROFESSIONAL PROVINCE OF R. S. ADAMSON BRITISH COLUMBIA ENGINEER

Robert S. Adamson, P.Eng.

PROFESSIONAL PROVINCE OF T. A. MORRISON BRITISH COLUMBIA ENGINEER

T.A. Morrison, P.Eng.

/cc

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Mr. David R. Bell
President
Micham Exploration Inc.
Suite 1104
18 King Street East
Toronto, Ontario
M5C 1C4

Dear Mr. Bell

We are pleased to submit our report entitled:

Preliminary Feasibility Study
of the
BANNOCKBURN PROPERTY
Madoc, Ontario

Should you have any questions concerning any part of the report or require more elaboration, we will be available for discussion at your convenience.

Yours very truly,
ORCAN MINERAL ASSOCIATES LTD.

C. R. Saunders

C. Raymond Saunders,
Vice President

Comments on a Review

Of a

Pre-Feasibility Study

For

MICHAM EXPLORATION INC.

on it's

BANNOCKBURN GOLD PROJECT

By

D. S. Rogers

September 1988

TABLE OF CONTENTS

PREFACE

The Preliminary Feasibility Study of the Bannockburn Property represents a complete and comprehensive report on the Madoc area project. The underground program to-date has been capably directed and the management team has done a thorough job in recording the development data. Geological mapping of the underground development, detailed chip and muck sampling and detailed diamond drilling have been completed and recorded. In future, the same attention to detail must be exercised in an attempt to determine the geological parameters of the deposit.

INTRODUCTION

On May 5th in company with C.R. Saunders, Vice President of Orcan Minerals Associates Ltd. I visited the Bannockburn property and reviewed the geological, sampling and diamond drilling data with the Project Manager, .J.C. Dadds. The visit also included a surface tour of the major vein systems which had been exposed by stripping the core area in the vicinity of the ramp portal. The cores from several underground diamond drill holes were also examined, notably holes 7-1 and 3-2 which were drilled through the core area and were believed to represent typical sections through the multiple vein zone. The ramp portal was sealed for safety reasons following the completion of the underground program, and no underground examination was possible.

Subsequent to the property visit I spent a period of ten days in June at the Vancouver office of Orcan Mineral Associates in the preparation of the Preliminary Feasibility Study of the Bannockburn Property. My principal involvment dealt with the section entitled 6.0 Reserves, which included sampling and assaying procedures, the comparison of various sampling results, ore reserve methodology and classification and finally, the blocking out of "ore" on individual cross sections for subsequent tonnage and grade calculations.

CRITIQUE OF PRELIMINARY FEASIBILITY STUDY

Summary

Personal comments on the report have been limited to a more critical assessment of sections 6, 7, 12 and 13 which include the Reserves, Mining, Economic Analysis and Conclusions sections of the study and which relate more closely to my own experience and current envolvement with the project. My recommendations, of a more specific nature, re-emphasize the objectives of the proposed feasibility program but place a higher priority on the further definition of the ore on the 75 metre level before another lower elevation is tested (eg. 2nd level). Ultimately, however, the deposit will require development at successive lower elevations to confirm the downward extension of the veins.

6.0 RESERVES

6.3 Comparison of Sampling Results

While I concur with the approach taken by the authors in their analysis of various types of sampling comparisons, I feel the inferences are drawn from too small a sample population to be reliable, particularly in the case of the chip/muck comparisons to diamond drill core samples. To date the chip and muck samples show relatively close agreement, however more comparisons are required before any relationships can be determined.

6.4 Cutting High Grade Assays

The traditional practise of cutting high grade assays to some arbitrary figure based on the experience of each individual deposit has been addressed by the authors. To acknowledge the validity of this exercise, at the present stage of development of the Bannockburn property is of more importance than the arbitrary figure itself. The selection of the arbitrary 3 ounce cutting figure is reasonable for the deposit based on the percentage of high assays obtained to-date; only 6.5% of the assays are above 1.0 ounce. Fine free gold is common in the deposit, however course gold occurences are not. It is recommended to continue to calculate the ore grades, drift averages etc. carrying both a cut and an uncut value until a reconcilliation with milled tonnage is available.

7.0 MINING

7.3.6 Stoping

The authors have recognized the limitations of the mining options on essentially a narrow vein low tonnage type stoping operation. Open stoping in a pre-determined panel and rib pillar arrangement using scrapers should prove the most effective in dealing with dip variations in the ore. Where the advanced knowledge of steeper dipping veins (>45°) is determined by raises, shrinkage stoping methods are proposed. This method will result in hang-ups and necessitate scraping if local flattening or drag folding occurs along the vein.

As indicated by the results of recent underground diamond drilling, several holes intersected continuous sections of low grade mineralization whose widths far exceeded the 6 foot minimum stoping limit established for narrow vein mining. My recommendations include proposed development to assess the potential of possible low grade stringer zone type mineralization which could be mined by less costly bulk mining methods. In addition to having a beneficial effect on overall stoping costs it would add flexibility to the mining operation.

12.0 ECONOMIC ANALYSIS

12.2 Sensitivity Analyses

The possibility exists for increasing the grade of the ore reserve tonnage from the stated 0.186oz/ton, however the expectations that it will approach 0.25oz/ton is somewhat unrealistic for two reasons.

1. Granted the diamond drilling indicated grade may understate the actual stope grade, as is characteristic of so many gold deposits, however the effects of mining dilution may negate any increase. The experience of mining narrow irregular shallow dipping veins presents little margin for error despite allowing for a 6 foot mining width.

2. The breakdown of the ore reserve total based on cut-off grades is as follows.

TONNAGE		GRADE	CUT-OFF RANGE
187,300 tons	@	0.224oz/ton	0.10oz/ton
60,860 tons	@	0.069oz/ton	0.050z/ton to 0.09oz/ton
248,160 tons	@	0.186oz/ton	

From the above chart if appears possible to up-grade the ore reserve grade for the portion 0.10oz/ton cut-off however, when the total includes the ore to a 0..05oz/ton cut-off the up-grading to 0.25oz/ton appears unlikely.

13.0 CONCLUSIONS

I would heartily endorse the conclusions which have been stated in the report regarding the future potential of the property and to those areas where additional exploration is required and where additional reserve tonnage can be developed. As previously stated, I fail to share the degree of optimism the authors exhibit for the up-grading of the deposit. Nevertheless, the true test of the grade will only be realized in actual test stoping situations.

RECOMMENDATIONS FOR FUTURE DEVELOPMENT

CORE AREA - 75 METRE LEVEL Figure - 6

1. Extend # 1 D.D. Bay (X/C) 100 feet northeasterly to investigate the zone containing the value 0.047oz/ton for 57.9 feet in hole 1-8. Of particular interest, at the end of this low grade section a vein assayed 0.182oz/ton for 5 feet. The possibility exists for drifting north and south on the strike extension of this vein.

2. Extend #3 D.D North Drift 50 feet + to explore the intersection 0.189oz/ton for 14 feet in hole 1-7.

3. Extend #3 D.D Bay (X/C) 75 feet northeasterly along drill hole 3-4 to investigate intersections, 1.032oz/ton for 1.5 feet and 0.213oz/ton for 4.2 feet, respectively. Drive south on either or both of these veins.

4. Recent surface stripping (August 1988) has opened up a new quartz vein which strikes northwest and dips 33° to the northeast. The projection of this vein to the 75 level appears to fall between the projection of the #1 North and the #2 North drift vein structures. (ie. 0.182oz/ton for 5 feet. See Recommendation 1)

NORTH ZONE Figure 5

5. The North Zone underground exploration, as recommended in the Preliminary Feasibility Study, was to be deferred until the 2nd level was developed (175 metre level). The continued extension of the #3 North Drift is favorably located to explore the gap, which on the basis of limited surface diamond drilling, exists between the core area and the North Zone. If encouraged by the results of Recommendation 2 extend this drift towards the North Zone area.

6. Exploration diamond drilling underground is recommended from all the above drift extensions, with detailed drilling in areas where results warrant.

SURFACE DRILLING

7. A minimum of 5,000 feet of surface drilling is recommended on the area referred to as The Plantt Option, to explore for the continuation of parallel quartz structures in the easterly portion of the North Zone.

Respectfully submitted,

Dean S. Rogers

Dean S. Rogers

Micham Exploration Inc.

ESTIMATED COSTS
PHASE II
BANNOCKBURN PROJECT
Madoc, Ontario

6 October, 1988

Orcan Mineral Associates Ltd. Vancouver, Canada

PURPOSE

The purpose of this study is to outline and cost an exploration program for the Bannockburn project that will have a total cost in the order of $2.0 million. It is based on information set out in the "Preliminary Feasibility Study of the Bannockburn Project" by Orcan Mineral Associates Ltd., dated July, 1988, and on discussions with Mess'rs. G. D. House and L. J. Manning of Mono Gold Mines Inc.

OBJECTIVE

The objective of the proposed underground program is to further explore the Bannockburn Gold deposit.

The budgetary constraint will confine underground work to the 75-ft. level from the existing workings and to raises between that level and surface.

The components of this objective are:

(i) Establish whether the core area should be regarded for mining purposes as a stockwork, a series of veins, or some combination of the two.

(ii) Explore the lateral continuity of veins.

(iii) Explore the dip continuity of mineralized zones between the 75-ft. level and surface.

(iv) Explore latteraly and to depth by limited diamond drilling.

PROPOSED UNDERGROUND WORK

Level development will be to a minimum practical size for the equipment in use. Headings used for haulage such as No. 2 X-C will be larger (10 ft. x 10 ft.) than vein drifts (9 ft. x 9 ft. or 8 ft. x 9 ft.). Raises will be 7 ft. x 5 ft.

For the following descriptions, reference is made to the attached plan.

No. 2 X-C will be declined to the level and then driven at +1% to break into the north end of the existing No. 3 Dr.N. Development will then turn north following No. 3 Vein.

It is intended that 2 or 3 veins which will probably be cut in No. 2 X-C will be drifted on North and South of the crosscut. These may correspond to No. 1 and No. 2 veins in the existing workings. These drifts will be driven under geological control but one connection to No. 1 or No. 2 Dr.N. will be necessary for drainage. Parts of the pillar between No. 1 and No. 2 Dr.N. will be slashed to allow further investigation of this apparent network of veins.

No. 1 Dr. S. will be driven on vein from the existing No. 1 X-C. No. 3 X-C will be driven east through the vicinity of the vein intersections in DDH 85-9 and 3-7. A connection will be made between the end of this X-C and the existing No. 3 Dr.S. No. 3 Dr.S. will then be extended south on vein. These excavations will be driven at +1%.

Raises will be driven to investigate favourable veins between the 75-ft. level and surface. These will facilitate ventilation and provide escapeways in addition to their primary function. The number of these raises will, to some extent, depend on the funds available.

The approximate footages involved are as follows:

No. 2 X-C	210	ft.
No. 1 Dr.N.	150	
No. 2 Dr.N.	130	
No. 3 Dr.N.	100	
No. 1 Dr.S.	180	
No. 3 X-C	90	
No. 2 Dr.S.	40	
No. 3 Dr.S.	320	
Slash Equivalent	200	
Drifting	1,420	
Raising	600	
Total Footage	**2,020**	**ft.**

Diamond drilling will comprise approximately 2,000 feet on surface and 2,000 feet underground.

ESTIMATED COST

Costs have been estimated on the basis of rental or purchase of individual items and then summarized into the categories which follow.

No.	Item	Total Expenditure	Possible Salvage	Final Cost	Percent
1	Heavy Equipment	280,000	0	280,000	14.19%
2	Small Powered Equipment	132,000	49,000	83,000	4.21%
3	Linear Goods	52,000	0	52,000	2.64%
4	Tanks & Receivers	18,000	4,000	14,000	0.71%
5	Electrical Equipment	83,000	1,000	82,000	4.16%
6	Small tools & Equipment	28,000	3,000	25,000	1.27%
7	Consumables	354,000	0	354,000	17.94%
8	Manpower	461,000	0	461,000	23.37%
9	Buildings	12,000	0	12,000	0.61%
10	General Expense	120,000	0	120,000	6.08%
11	Diamond Drilling	120,000	0	120,000	6.08%
12	Overhead	191,000	0	191,000	9.68%
13	Contingency	179,000	0	179,000	9.07%
	Total Estimated Cost - Bannockburn	**$2,030,000**	**$57,000**	**$1,973,000**	**100.00%**

Respectfully submitted,

ORCAN MINERAL ASSOCIATES LTD.

C. R. Saunders

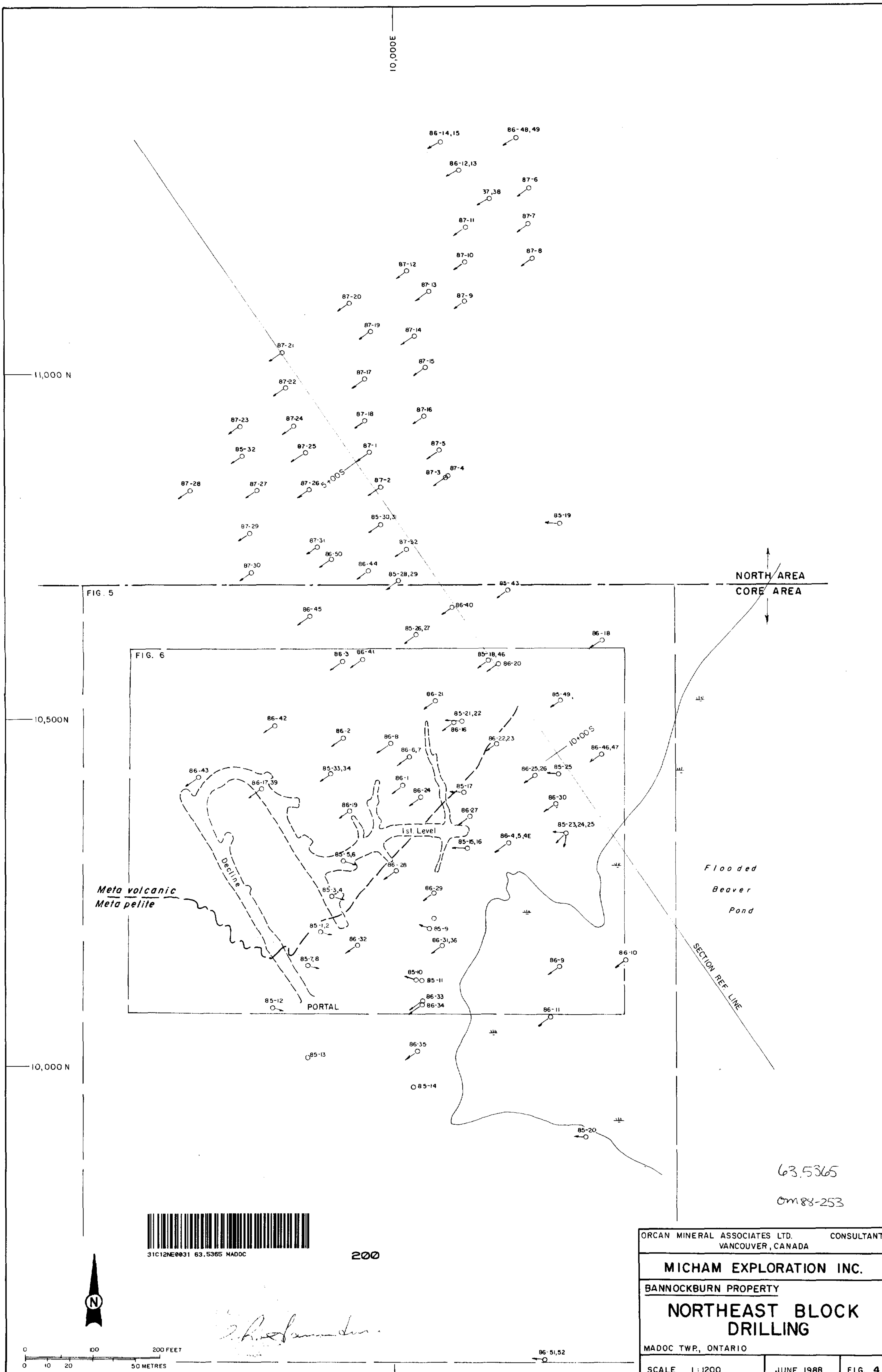
PROFESSIONAL ENGINEER PROVINCE OF BRITISH COLUMBIA C. R. SAUNDERS

C. Raymond Saunders, P.Eng.

T. A. Morrison

PROFESSIONAL ENGINEER PROVINCE OF BRITISH COLUMBIA T. A. MORRISON

Thomas A. Morrison, P.Eng.



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ORCAN MINERAL ASSOCIATES LTD. CONSULTANTS
VANCOUVER, CANADA

MICHAM EXPLORATION INC.

BANNOCKBURN PROPERTY

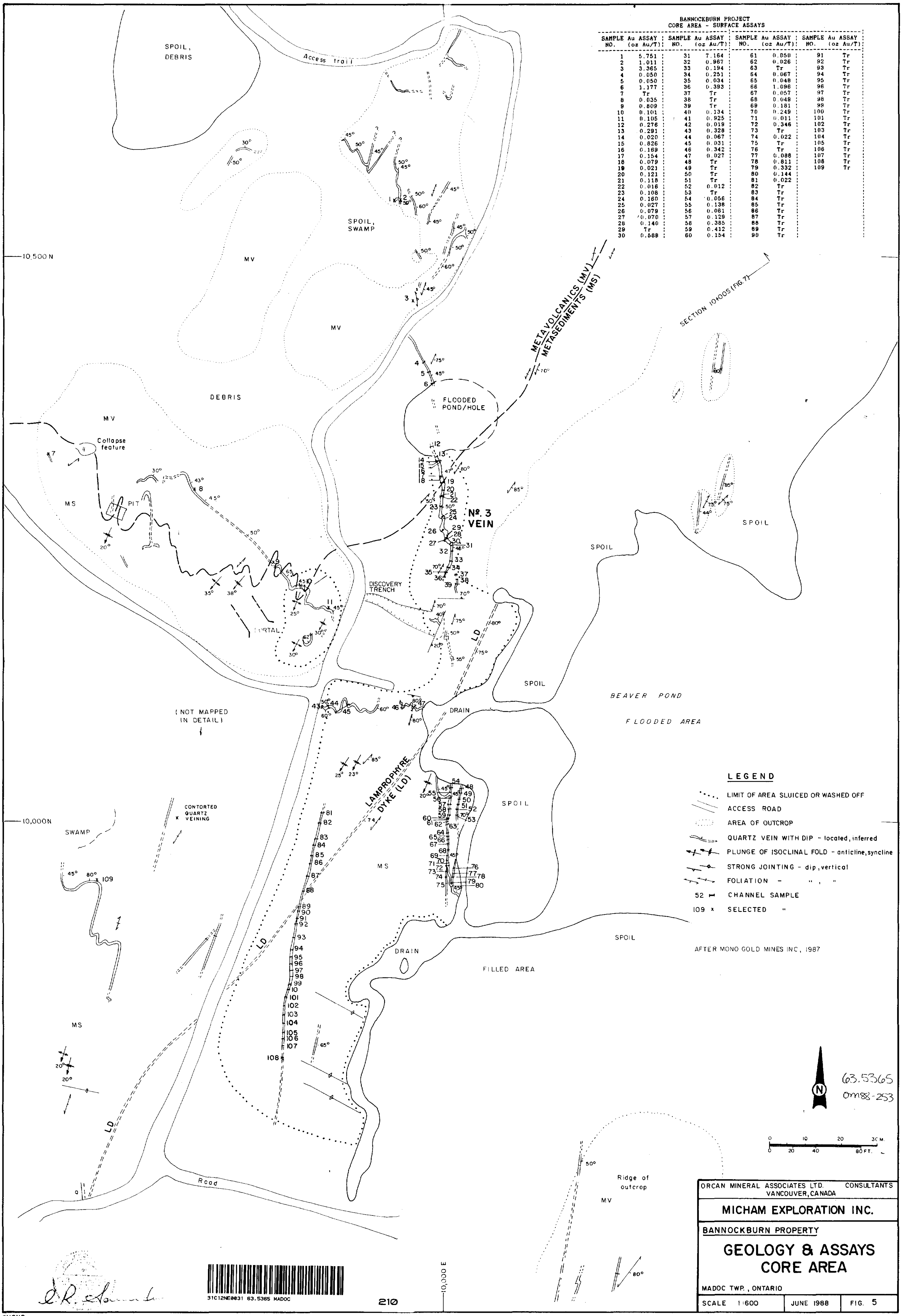
NORTHEAST BLOCK DRILLING

MADOC TWP., ONTARIO

SCALE 1:1200	JUNE 1988	FIG. 4

BANNOCKBURN PROJECT
CORE AREA - SURFACE ASSAYS

SAMPLE NO.	Au ASSAY (oz Au/T)	SAMPLE NO.	Au ASSAY (oz Au/T)	SAMPLE NO.	Au ASSAY (oz Au/T)	SAMPLE NO.	Au ASSAY (oz Au/T)
1	5.751	31	7.164	61	0.050	91	Tr
2	1.011	32	0.967	62	0.026	92	Tr
3	3.365	33	0.194	63	Tr	93	Tr
4	0.050	34	0.251	64	0.067	94	Tr
5	0.050	35	0.034	65	0.048	95	Tr
6	1.177	36	0.393	66	1.096	96	Tr
7	Tr	37	Tr	67	0.057	97	Tr
8	0.035	38	Tr	68	0.049	98	Tr
9	0.809	39	Tr	69	0.181	99	Tr
10	0.101	40	0.134	70	0.249	100	Tr
11	0.105	41	0.925	71	0.011	101	Tr
12	0.276	42	0.019	72	0.346	102	Tr
13	0.291	43	0.328	73	Tr	103	Tr
14	0.020	44	0.067	74	0.022	104	Tr
15	0.826	45	0.031	75	Tr	105	Tr
16	0.169	46	0.342	76	Tr	106	Tr
17	0.154	47	0.027	77	0.088	107	Tr
18	0.079	48	Tr	78	0.811	108	Tr
19	0.021	49	Tr	79	0.332	109	Tr
20	0.121	50	Tr	80	0.144		
21	0.118	51	Tr	81	0.022		
22	0.016	52	0.012	82	Tr		
23	0.108	53	Tr	83	Tr		
24	0.160	54	0.056	84	Tr		
25	0.027	55	0.138	85	Tr		
26	0.079	56	0.061	86	Tr		
27	0.070	57	0.129	87	Tr		
28	0.140	58	0.385	88	Tr		
29	Tr	59	0.412	89	Tr		
30	0.588	60	0.154	90	Tr		



ORCAN MINERAL ASSOCIATES LTD. CONSULTANTS
VANCOUVER, CANADA

MICHAM EXPLORATION INC.

BANNOCKBURN PROPERTY

GEOLOGY & ASSAYS CORE AREA

MADOC TWP., ONTARIO

SCALE 1:600	JUNE 1988	FIG. 5

31C12NE0031 63.5365 MADOC

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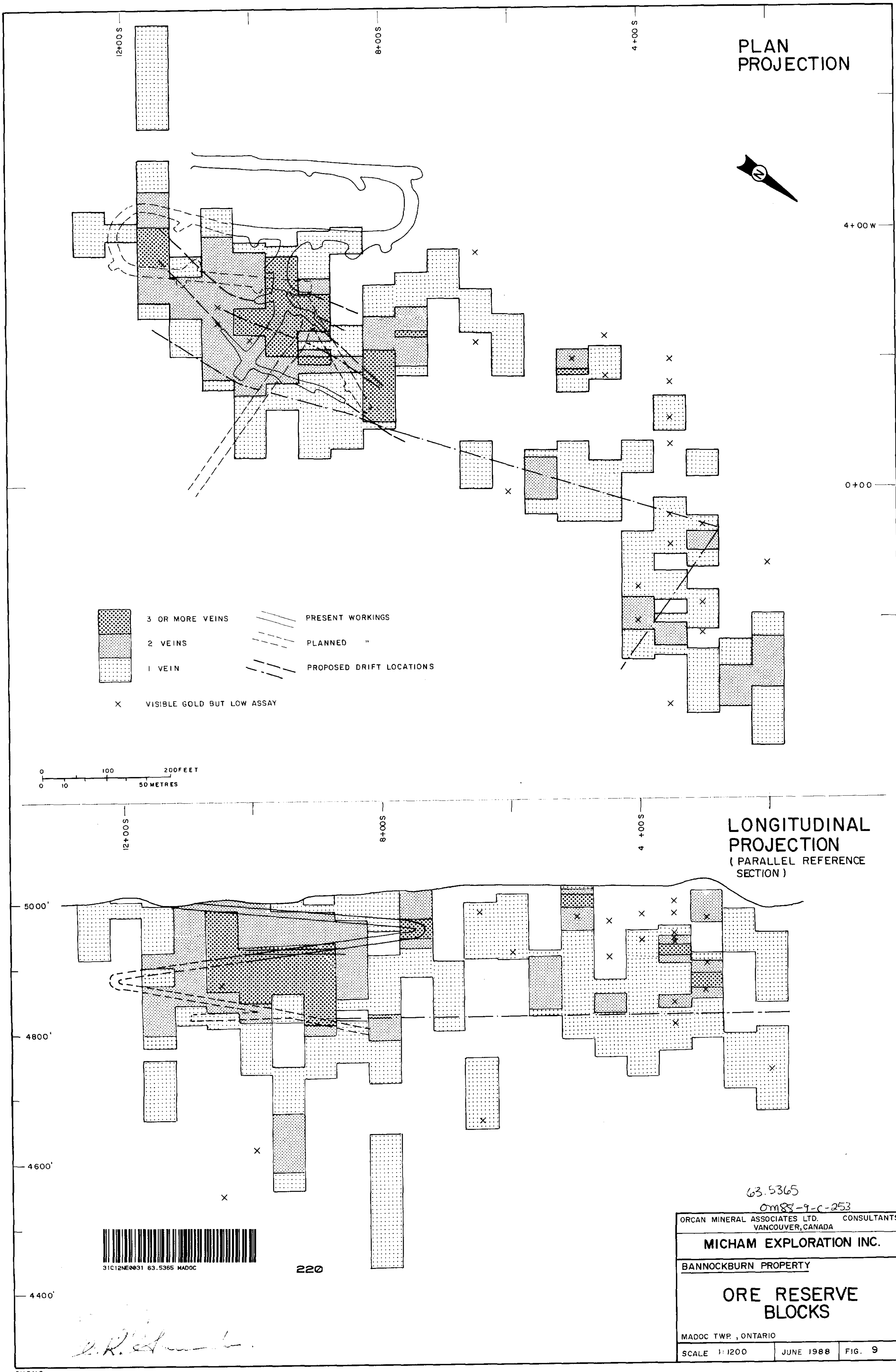
CHONG



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ORCAN MINERAL ASSOCIATES LTD. CONSULTANTS
VANCOUVER, CANADA

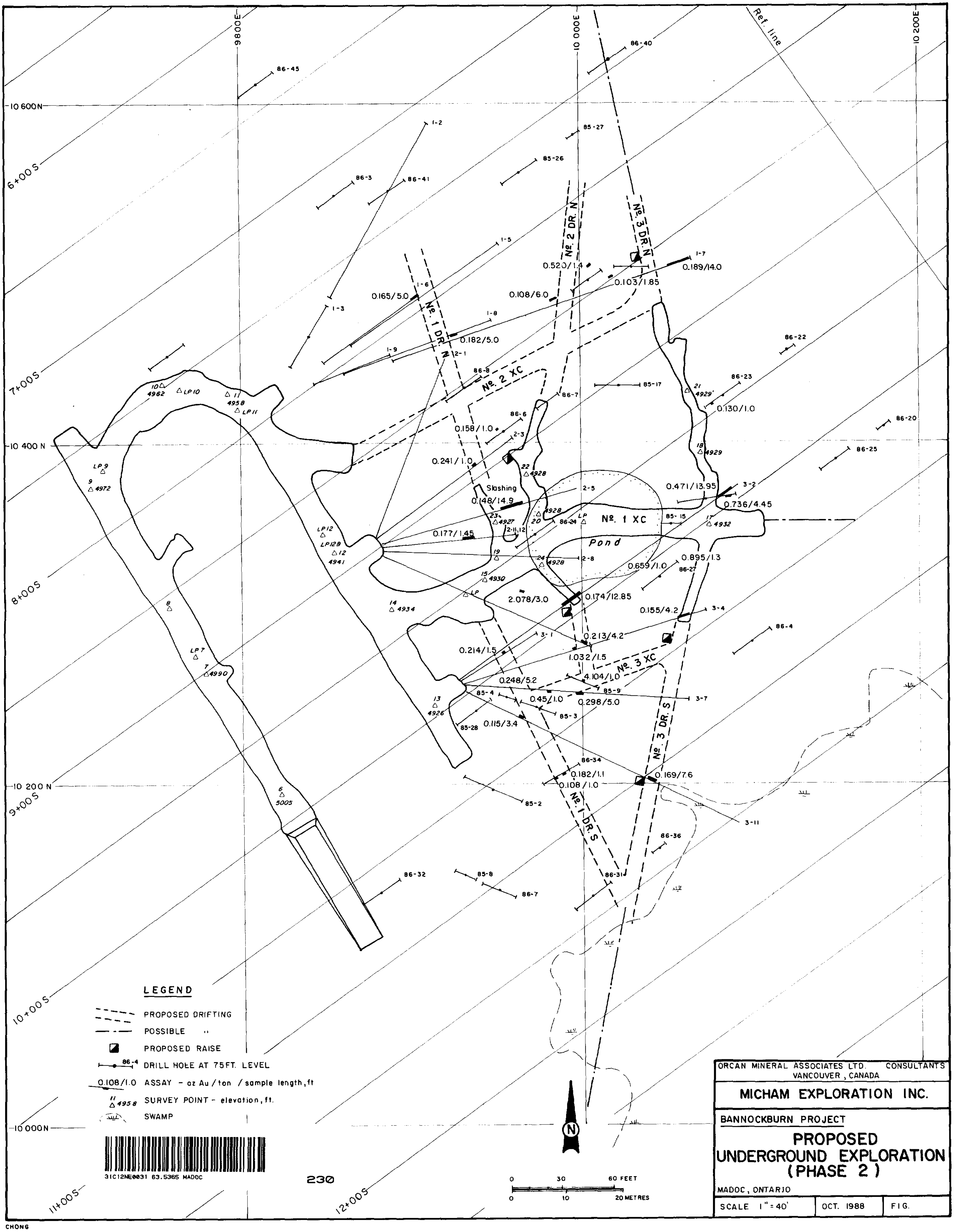
MICHAM EXPLORATION INC.

BANNOCKBURN PROPERTY

ORE RESERVE BLOCKS

MADOC TWP., ONTARIO

SCALE 1:1200	JUNE 1988	FIG. 9



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CHONG