THE DISTRIBUTION OF GOLD AND OTHER ELEMENTS IN THE URANIUM CONGLOMERATES OF ELLIOT LAKE, CANADA

By

Ⓒ DAVID IAN ROSS, B.Sc.

A Thesis
Submitted to the School of Graduate Studies
in Partial Fulfilment of the Requirements
for the Degree
Master of Science

McMaster University
September 1981

MASTER OF SCIENCE (1981)
(Geology)

McMASTER UNIVERSITY
Hamilton, Ontario

TITLE : The Distribution of Gold and Other Elements in the Uranium Conglomerates of Elliot Lake, Canada

AUTHOR : David Ian Ross, B.Sc. (Aberdeen University)

SUPERVISOR : Dr. J.H. Crocket

NUMBER OF PAGES : x, 104

ABSTRACT

A study has been undertaken to investigate the controls on the concentration and distribution of gold and platinum group elements in the uranium deposits of Elliot Lake.

Gold is present almost entirely in the pyrite and gold concentration in the ore varies with pyrite content. Sporadic high gold values can be related to specific sedimentary structures and may indicate the presence of original detrital gold. Gold is preferentially concentrated in pyrite compared to pyrrhotite. The distribution of gold in pyrite along the strike of the Lower Reef, Denison Mine, is difficult to explain by the placer hypothesis. It indicates that the pyrite now observed in the oligomictic conglomerates is not of direct detrital origin.

An attempt to determine the concentration and distribution of platinum group elements within the ore conglomerates failed due to experimental problems.

The distribution of the majority of major and trace elements can be related to the mineralogy of the ore reefs. Relationships with pebble size are due in part to depositional processes operating at the time the conglomerates were deposited, and possibly in part to processes operating after burial of the conglomerates.

ACKNOWLEDGEMENTS

The author wishes to express his appreciation to Dr. J.H. Crocket who suggested the study and supervised the research.

The author is also indebted to Mr. A. Kabir for his assistance and advice in the analytical procedures used to determine the noble metals, and Mr. O. Mudroch for his assistance in the X.R.F. analyses.

I would like to thank Rio Algom Mines Ltd. for granting permission to collect samples from their underground workings. Thanks in particular go to Mr. D. Sprague and his staff of the Geological Department, New Quirke Mine.

The financial support of the N.S.E.R.C. (Natural Sciences and Engineering Research Council) is gratefully acknowledged.

Finally, I wish to thank my wife for her help and understanding.

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

CHAPTER I

1.1 Introduction

In 1953 uraniferous quartz pebble conglomerates were discovered at the base of the Huronian System, along the north shore of Lake Huron, 20km east of the town of Blind River. This resulted in widespread exploration and in the development of the Elliot Lake mining camp, 40km to the northeast of Blind River.

Figure 1 : Location of Blind River - Elliot Lake area(From Robertson 1976)

The similarities between the Blind River - Elliot Lake ores and those of the Witwatersrand Basin, Republic of South Africa, were immediatly obvious. Although the conglomerates of the Witwatersrand Basin are the worlds' major gold producer, similar conglomerates from Elliot Lake have produced only trace or negligible amounts of noble metals and thus noble metals have played no part in the economic development of the Elliot Lake deposits.

Although the noble metal content is known to be low, the geological similarities with the Rand suggest that gold and platinum group elements could at least be potential by-products from the mining operations. This was the economic rationale behind this work.

The present study was conducted to investigate the controls on the concentration and distribution of gold and platinum group elements in the Elliot Lake ores, using the integrated input of sedimentological, major plus trace element and noble metal data.

Knowledge of the noble metal distribution should hopefully throw more light on the problems concerning the genesis of these deposits.

1.2 Geology of the Uranium Deposits

The Blind River - Elliot Lake uranium deposits are confined to Huronian sedimentary rocks. These have been weakly metamorphosed and folded into a broad east - west trending syncline and anticline, both of which plunge gently towards the west. The Huronian sequence is transgressive from south to north and unconformably overlies granitic rocks, metavolcanics and metasediments of Archean age.

Figure 2 is a generalized geological map of the Blind River - Elliot Lake area showing the location of the three major ore zones(areas outlined in solid black). Except for the Pronto deposit, all the uranium deposits occur in the Quirke Lake Syncline. Figure 3 is a geological cross - section through the New Quirke Mine, Quirke Zone.

Many descriptions of the regional geology of this area have been presented in the literature eg. Roscoe (1969), Robertson (1976), and a discussion will therefore not be presented here. Table 1 is a Table of Formations for the Blind River - Elliot Lake area using the nomenclature recommended by the Federal - Provincial Committee on Huronian Stratigraphy(J. A. Robertson et al, 1969).

The economic uranium deposits of the Blind River - Elliot Lake area are found as pyritic quartz pebble conglomerates in three major ore zones, the Pronto, Nordic and Quirke Zones. These strongly uraniferous conglomerates are confined to the Matinenda Formation which ~~is~~ the lowermost formation

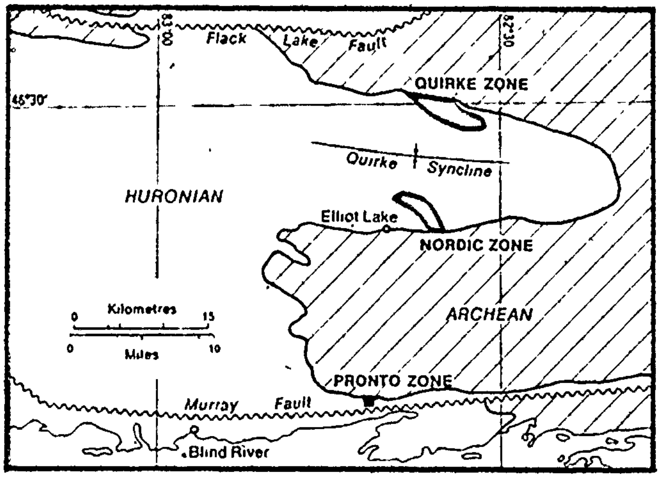


Figure 2 : Generalized geological map of the Blind River - Elliot Lake area. Areas outlined in solid black are the major ore zones.(Adapted from Theis 1979)

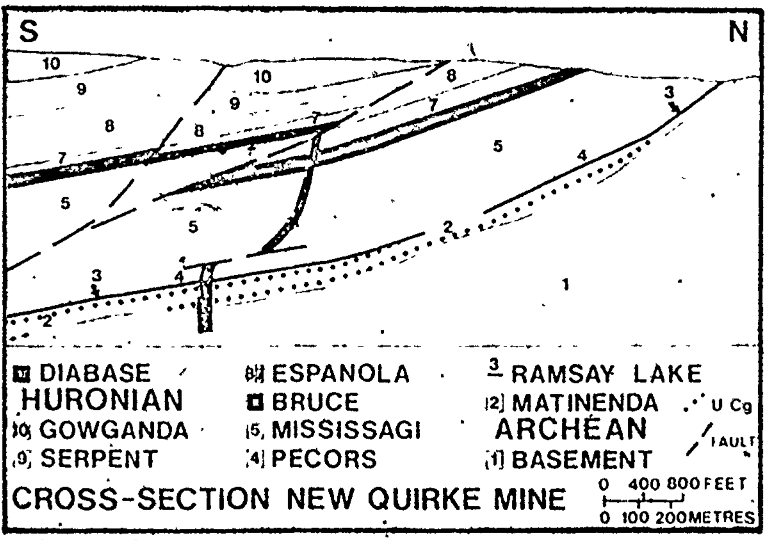


Figure 3 : Geological cross - section through the New Quirke Mine, Quirke Zone(From Robertson 1976)

UNIT	DOMINANT LITHOLOGY	AGE (Million Years)
PHANEROZOIC		
CENOZOIC		
PLEISTOCENE AND RECENT	Sand, gravel, till	
UNCONFORMITY		
PALEOZOIC		
ORDOVICIAN	Limestone	
PRECAMBRIAN		
PROTEROZOIC (LATE AND MIDDLE PRECAMBRIAN)		
KEWEENAWAN SUPERGROUP		
Sudbury Dikes	Olivine diabase	1 225
INTRUSIVE CONTACT		
Mount Lake Dike	Quartz diabase	
INTRUSIVE CONTACT WITH NIPISSING DIABASE		*
HUDSONIAN		
Croker Island Complex	Gabbro, granite	1,445
Cutler Batholith	Granite	1 750
INTRUSIVE CONTACT		
PENOKEAN		
Nipissing	Quartz diabase diorite	2 155
INTRUSIVE CONTACT		
HURONIAN SUPERGROUP		
Cobalt Group		
Bar River	Quartzite	
Gordon Lake	Siltstone, sandstone	
Lorrain	Quartzite, conglomerate, arkose	
Gowganda	Conglomerate, greywacke, quartzite	
UNCONFORMITY DISCONFORMITY		
Quirke Lake Group		
Serpent	Quartzite	
Espanola	Limestone, siltstone	
Bruce	Conglomerate	
LOCAL DISCONFORMITY		
Hough Lake Group		
Mississagi	Quartzite	
Pecors	Argillite	
Ramsay Lake	Conglomerate	
LOCAL DISCONFORMITY		
Elliot Lake Group**		
McKim	Argillite	
Matinenda	Quartzite ± U-conglomerate Conglomerate Arkose ± U conglomerate regolith	
UNCONFORMITY		
ARCHEAN (EARLY PRECAMBRIAN)		
LATE ARCHEAN INTRUSIVES	Diabase	2,500
INTRUSIVE CONTACT		
KENORAN (ALGOMAN)	Granite	2,500+
INTRUSIVE CONTACT		
EARLY ARCHEAN INTRUSIVES	Gabbro	
INTRUSIVE CONTACT		
KEEWATIN	Volcanic and sedimentary rocks	

* Mount Lake Dike may be 1 795 m y

** Volcanic rocks are found locally in the Elliot Lake Group (6), each occurrence has been given its own name

c 2 400

Note: Geological ages given are from a variety of sources

Table 1 : Table of Formations for the Blind River - Elliot Lake area (from Robertson, 1976)

within the Elliot Lake Group and lies at the base of the Huronian section. According to Robertson (1976) the Huronian succession was deposited with northward overlap and consequently the ore beds of the Nordic Zone are older than those of the Quirke Zone, with the Pronto Zone possibly being the oldest.

There is general agreement that the rocks of the Matinenda Formation were deposited by a system of southeasterly flowing braided rivers (McDowell 1957, Piennar 1963). Piennar (1963) used isopach maps to show that deposition of the Nordic and Quirke Zone sediments was controlled by depressions or valleys in the Archean basement, these striking northwest - southeast. Robertson (1966) has suggested that the positive and intermediate topographical areas in Figure 4 are cherty iron formations which stood as ridges on the Archean surface.

Each ore zone may have one (Pronto) or more (Nordic, Quirke) ore beds, commonly refered to as "reefs". These may be correlated between different mines within the same ore zone. Table 2 presents a correlation of ore beds between the Quirke, Denison and Stanrock Mines (Quirke Zone) showing the variable terminology applied to the same ore horizons in different mines.

The ore reefs are composed of coalesced lenses of sedimentary units. Individual units are discontinuous and may grade from well packed conglomerates to quartzites and may be truncated or scoured by later units (Theis 1979; Piennar 1963; Roscoe 1969). Figure 5 is a generalized diagram which illustrates these properties.

(Underlined beds have supported production)

	QUIRKE 1 & 2 (Rio Algom Mines) Ramsay Lake Formation		DENISON Denison Mines Ltd. Ramsay Lake Formation		STANROCK Denison Mines Ltd.
Matinenda Formation	Quartzite		Quartzite		
	Upper	————	F		
	Quartzite		Quartzite		
	A(Quirke 1)	————	E		
	Quartzite		Quartzite		
	B	————	D		
	Quartzite		Quartzite		
	C-L	————	Floater reef(s)		
	Quartzite		Quartzite		
			A_1		
			A Quartzite		Quartzite
			A_2	————	St_1
	C(Quirke 2)	————	I Q. (Interbedded quartzite)		Quartzite
			B	————	St_2
	Quartzite		Quartzite		Quartzite
	FWC (Scraps)	—— ? ——	Scraps		St_3
	Quartzite		Quartzite		Quartzite
	Basement		Basement		Basement

Table 2 : — Correlation of quartz-pebble conglomerate reefs, Quirke Zone
(Adapted from Robrtson 1976 in which the C Reef, New Quirke Mine is correlated with only the A Reef, Denison Mine)

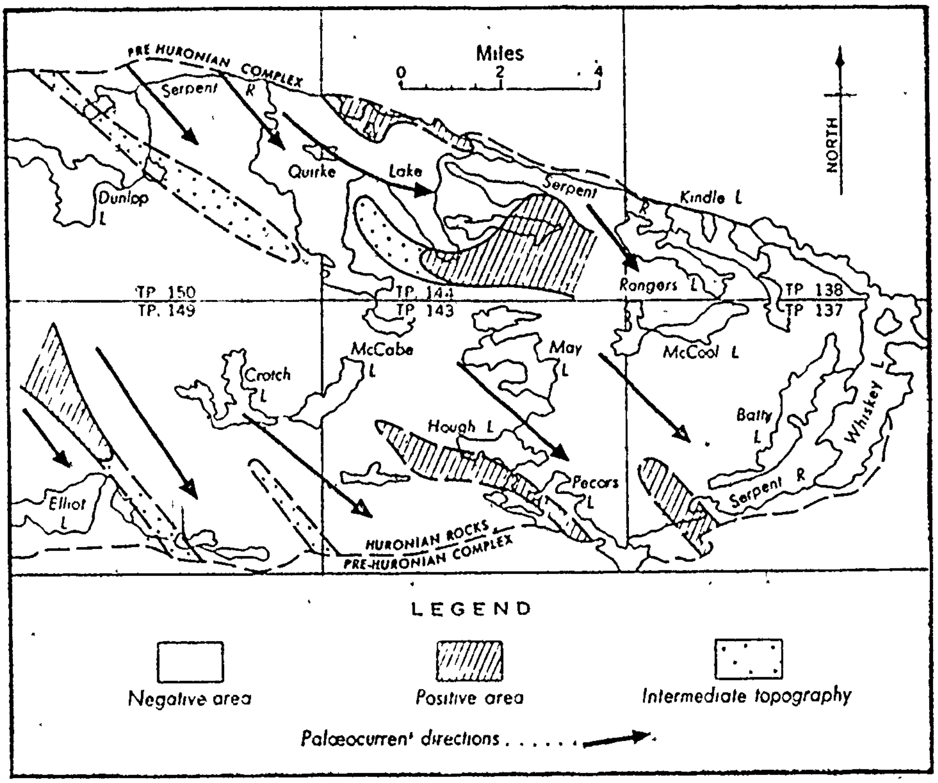


Figure 4 : Topography of surface on which Matinenda Formation was deposited, Quirke Lake Syncline. (from Piennar 1963)

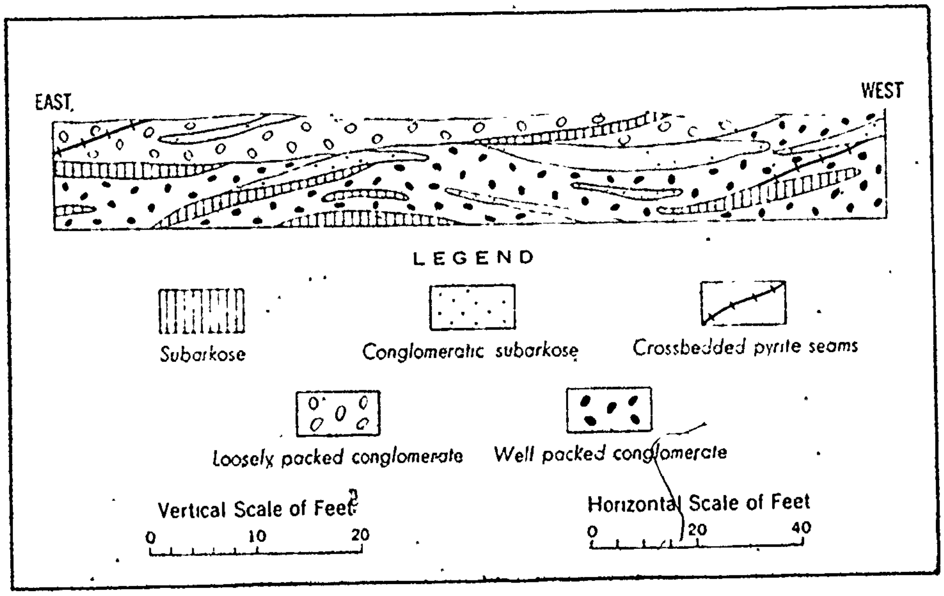


Figure 5 : Section showing intricate lensing within the ore conglomerate, Quirke Mine. (from Piennar 1963)

Robertson (1976) describes five types of boundaries to the ore beds:

i) Outcrop.

ii) Wedge out against basement highs.

iii) Lateral thinning of conglomerate accompanied by a drop in grade.

iv) Erosional unconformity with the Ramsay Lake Conglomerate.

v) Faulting.

The ore bearing conglomerates are composed of well rounded, well sorted quartz pebbles with a few pebbles of chert, set in a matrix of quartz, sericite, pyrite and minor feldspar. Pebble size within the ore reefs decreases to the southeast, the direction of sediment transport (Piennar 1963).

The pyrite grains are anhedral to euhedral, with many being rounded ("buckshot pyrite"). Roscoe (1969) has described the following forms of pyrite in the Elliot Lake ores: 1) anhedral grains commonly about 1mm in size that contain abundant small inclusions of other minerals; 2) euhedral to subhedral grains 0.3 to 0.6mm in diameter that contain few inclusions but embay quartz pebbles or other minerals and larger grains that include normal sized grains of other minerals; 3) spheroidal grains about 0.5mm in diameter with abundant inclusions in their central part but not in their rims which may represent overgrowths; 4) anhedral grains with curved markings that suggest boundaries of overgrowths; 5) tiny anhedral and euhedral grains disseminated in the sericitic matrix of some conglomerates; 6) aggregates of small grains commonly 0.3 to 0.6mm but up to 12mm in diameter; 7) brecciated grains evidently squeezed between pebbles during compaction of the conglomerate.

Rounded grains are considered by some workers to be indicative of detrital transport (eg. Robertson and Steenland 1960; Roscoe 1969; Theis 1979), euhedral grains arising from overgrowths on rounded grains. Theis (1979) attempts to show that the size of large pyrite grains is related to quartz pebble size within a given sample despite the variability of pyrite grain morphology. He concludes that the deposition of much of the pyrite was controlled by the same mechanism that controlled the deposition of the quartz pebbles i.e. that most of the pyrite is detrital.

In sampling well packed conglomerates, the size of the interstices would be related to pebble size. The negative correlation between potassium (sericite) and pebble size, also indicated in Theis study, indicates there is less fine grained matrix in the interstices of coarser conglomerates. Thus, if the pyrite had been introduced, the maximum crystal size would be related to the size of the interstices in the conglomerate and the amount of matrix occupying those interstices. Another possibility is that the pyrite is a replacement of a common detrital iron mineral, such as magnetite (Arnold 1954), which might have originally been in hydraulic equivalence with the quartz pebbles.

The principal ore minerals are uraninite, "brannerite" and monazite, and these minerals are confined to the matrix. The major ore mineral at the Pronto Mine and at the A Reef, Quirke Mine (Theis 1973) is "brannerite". Uraninite is the second most important mineral at the Pronto Mine, and apparently the most important in the Nordic Zone and the C Reef at the Quirke Mine (Theis 1973).

The uraninite generally occurs as black subangular grains approximately 0.1mm across. They are more commonly found in samples containing medium to larger pebbles, and are scarce in samples with a pebble size less than 3cm (Theis 1979). The uraninite carries about 6% ThO_2 (Derry 1960; Thorpe 1963) which is more typical of pegmatitic uraninite than vein type. Ramdohr (1958) and Robertson (1962) have described rounded grains which together with the thorium content would tend to favour a detrital origin. Theis (1979) states, "The occurrence of uraninite as discrete grains, the distribution of these grains within the ore conglomerate, their pronounced association with large pebbles, their position at or near the base of accumulations of detrital grains, their occurrence in cross-beds and their minor element chemistry all point conclusively to the detrital nature of the uraninite". Robertson (1962), Piennar (1963), Roscoe (1969) and Robertson (1976) also support a detrital origin for the uraninite. Rice (1958) believes uraninite was formed as a result of uranium leached from "brannerite" and then deposited as gummite and finally recrystallised as uraninite, in response to heat provided by intruded diabase. Patchett (1960) supports a secondary origin for uraninite and indicated the cores are carbon rich. Davidson (1957, 1965) has used the lack of uraninite in modern placers and geochemical principles along with the concept of uniformitarianism as an argument against the detrital origin of uraninite.

"Brannerite" is typically found as ovoid red-brown to black grains in the metamict state. Grains range in size from 0.25 to 1.50mm. The "brannerite" is a composite grain aggregate showing bladed rutile and anatase surrounded by uranium and rare-earth oxides. It occurs more abundantly in samples with a medium to smaller pebble size (Theis 1979). Most workers who have studied the mineralogy of these ores agree that "brannerite" in its present delicate and metamict state is unlikely to be transported as detrital grains. However the shapes of the grains are typically rounded and debate arises as to whether they were original detrital grains subsequently altered, or whether the grains were originally some titaniferous mineral into which uranium has been adsorbed. Robertson and Steenland (1960) interpreted the rounding to be caused by transportation and that the "brannerite" is detrital. Rice (1958) and Piennar (1963) also believe the "brannerite" was deposited as detrital grains but that alteration has taken place since. Ramdohr (1957) has suggested that the "brannerite" is an alteration product and has proposed the "Pronto Reaction" in which uranium was adsorbed onto titania collectors. Investigations by Ferris and Rudd (1971) and by Theis (1973, 1979) concluded that the "brannerite" formed at low temperatures during diagenesis as a result of uranium migrating to decomposing ilmenite.

Monazite grains are normally rounded to subangular and less than 0.3mm in diameter. Roscoe (1959) has described the monazites and has pointed out that monazite can contain considerable uranium and is, therefore, one of the ore minerals. Monazite is less abundant in coarse pebble conglomerates than in smaller pebble conglomerates (Theis 1979). It is generally accepted to be an original detrital mineral.

Other minerals present in minor quantities within the ore reefs are zircon, pyrrhotite, galena and thucolite. Robertson (1976) has listed a large number of heavy detrital minerals which have been identified from the ore reefs, but these occur in very small quantities and are not present in all samples.

The genesis of these deposits and others similar to it, eg. Witwatersrand (S. Africa), Jacobina (Brazil), has been much debated. Bateman (1955), Joubin and James (1957), Davidson (1957) and Heinrich (1958) have cited the high uranium to thorium ratios, the high titanium to iron ratio and the association of Ti, Co, Ni, Th and U in a deposit carrying the characteristic minerals "brannerite", uraninite and pyrite as evidence of a hydrothermal origin. Joubin (1954) suggested the "Keeweenawan" diabase as a source for the ore fluids, but later admitted mining evidence indicated that diabase intrusion post-dated uranium mineralization. Davidson (1957) suggested granite lying to the southeast of the Blind River - Elliot Lake area was the probable source for the ore fluids, but much of the granite formerly considered to be of post - Huronian age has since been proved to be older than the Huronian. Davidson (1965) proposed a new hypothesis in which uranium was leached from the sedimentary pile by ground waters. The mineralized ground waters eventually sank to the lowest permeable horizons, the oligomictic conglomerates, where uranium would be reprecipitated. Kimberley (1978) has proposed that the ore metals may have been introduced into shallowly - buried fluviatile sediments, but that there may have subsequently been some redistribution of the ore by placer processes.

Abraham (1953), Holmes (1957), McDowell (1957), Piennar (1963), Roscoe (1969), Robertson (1962), Robertson and Steenland (1960), Robertson (1966, 1969, 1976), Theis (1979) have all indicated a preference for a placer or modified placer origin. Evidence presented has included variation in the U/Th ratios, the absence of quartz veins or other signs of intense hydrothermal activity, the overall distribution of ore beds and the behaviour of ore minerals within the ore reefs. One of the principle arguments of the placerist theory is that the Precambrian atmosphere was essentially anoxygenic, so as to allow the release and transportation of uraninite and pyrite to the site of deposition. Both of these minerals are unstable at the earths surface today. Derry (1960) and Joubin (1960) have both suggested a syngenetic origin for the uranium mineralization with the possibility that uranium was carried in solution and precipitated in gravel banks.

CHAPTER II

Methods and Procedures

2.1 Sampling Rational and Procedure

Samples were collected from the following locations:

1) Quirke Zone
    i) New Quirke Mine
        a) C, A and Upper Reefs
        b) Ramsay Lake Conglomerate: at the contact with the C Reef in New Quirke Mine
    ii) Denison Mine
        Upper and Lower Reefs: these samples were collected by R. Martindale (see Martindale 1968)
2) Nordic Zone
    Nordic Mine: these samples were collected by W. Mercer. The reef and location are unknown.

The C Reef, New Quirke Mine, and the Upper and Lower Reefs, Denison Mine, are sometimes collectively called the Rio Algom - Denison Reef.

Sample descriptions are given in Table 3. Location maps are given in Appendix IV.

Pretorius (1981) described five sites in which exploitable gold and uranium occur in the Witwatersrand:

Table 3

Sample Descriptions

* = Sample illustrated in Figure 6
Cong. = Conglomerate
Qtzite = Quartzite

(Positions of samples from the Upper Reef, Denison Mine (Martindale 1968) were originally measured in inches and have not been converted to metric).

Sample	ROCK TYPE	COMMENTS
C Reef - New Quirke Mine(Quirke Zone)		
1	Cong.	
2	Cong.	
3A	Cong.	Graded Bed
3B	Cong.	
3C	Cong.	
4	Cong.	
5	Cong.	
6	Cong.	
6A	Qtzite.	Pyrite bands
6A1	Qtzite.	Pyrite cross beds
7	Cong.	
8	Cong.	
9	Cong.	Cong. resting on qtzite
10	Cong.	
10A	Cong.	Pyrite band seperating cong. and qtzite
10A1	-	
10A2	Qtzite.	
11	Cong.	
12	Cong.	
12A	Cong.	Cong. resting on qtzite
13	Cong.	Pyrite cross beds
14	Cong.	
30A	Cong.	
30B	Cong.	
30C	Cong.	
30D	Qtzite.	
[illegible]	Argillite.	
30F	Qtzite.	
30G	Cong.	Pyrite cross beds
30H	Cong.	
30I	Qtzite.	Pyrite cross beds
30J	Qtzite.	
[illegible]	Vein.	Calcite, sphalerite, Chalcopyrite, pyrite
50	Cong.	
4	Cong.	Pyrite cross beds

Sample	ROCK TYPE	COMMENTS
56	Cong.	Sample dark. Taken just above reef-basement contact
56B	Qtzite.	Pyrite cross bed
Upper Reef - Denison Mine(Quirke Zone)		
M44	Qtzite.	6" above HW. Pyrite cross beds
M45	Cong.	Hanging wall (HW)
M46	Cong.	6" below HW
M47	Cong.	12" below HW
M48	Cong.	18" below HW. Pyrite cross beds
M49	Cong.	24" below HW
M50	Cong.	30" below HW
M51	Cong.	36" below HW
M52	Cong.	42" below HW. Pyrite cross beds
M53	Cong.	48" below HW
M54	Cong.	54" below HW. Pyrite cross beds
M55	Cong.	60" below HW
M56	Cong.	66" below HW
M57	Cong.	72" below HW
M58	Cong.	12" above footwall (FW)
M59	Cong.	6" above FW
M60	Cong.	FW
M61	Qtzite.	6" below FW
Lower Reef - Denison Mine(Quirke Zone)		
M67	Qtzite.	6" below FW
M68	Cong.	FW
M69	Cong.	6" above FW
M70	Cong.	12" above FW
M11	Cong.	Taken near
M26	Cong.	
M21	Argillite.	Approx. 1cm thick pyrite band running through
M22	Cong. + Qtzite.	HW
M23	Cong.	
A Reef - New Quirke Mine(Quirke Zone)		
15A	Cong.	

Sample	ROCK TYPE	COMMENTS
15B	Cong.	
15C	Cong.	
16	Cong.	
17	Cong.	
18	Cong.	
19	Cong.	
19A	Cong.	Pyrite cross beds
19B	Cong.	
21A	Cong.	
21B	Cong.	
21C	Cong.	
22A	Cong.	
23	Cong.	At contact with dyke (pyrrhotite only)
23A	Cong.	
24A	Cong.	
25	Cong.	
31	Cong.	Pyrite cross beds
32	Cong.	
Upper Reef - New Quirke Mine(Quirke Zone)		
20	Cong.	Pyrite bands
Nordic Zone		
N1	Cong.	
N2	Cong.	
N3	Cong.	
N4	Cong.	
N5	Qtzite.	Pyrite cross beds
N6	Qtzite.	Pyrite cross beds
N7	Cong.	
N8	Cong.	
Reworked C Reef(Ramsay Lake Conglomerate)		
41	Cong.	Pyrrhotite only
45	Cong.	Pyrrhotite only
51	Cong.	
51A	Cong.	3cm thick pyrite band in
52	Cong.	
C Reef Footwall Beds - New Quirke Mine		
53	Cong.	Abundant silt scraps in
Fault cutting C Reef		
49	Fault gouge	Galena and Thucolite present

i) In the matrix of the conglomerates.

ii) In pyritic quartzites where the gold and uranium particles lie on the foresets of the cross-bedded sands.

iii) On quartzites along the planes of unconformities that seperate two cycles of sedimentation.

iv) On shales along the planes of unconformity that seperate succeeding cycles of sedimentation.

v) In carbon seams that are developed on, or immediately adjacent to, planes of unconformity.

These then are the sites in which one might look for gold in the Elliot Lake uranium ores. They might also be favourable sites for grains of the platinum group elements.

Besides the normal conglomeratic ore, various specific sedimentary structures were chosen for analysis. These are shown diagrammatically in Figure 6.

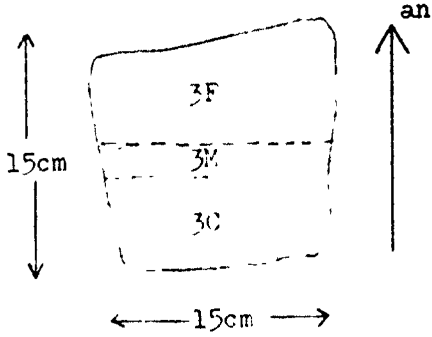
Figure 6

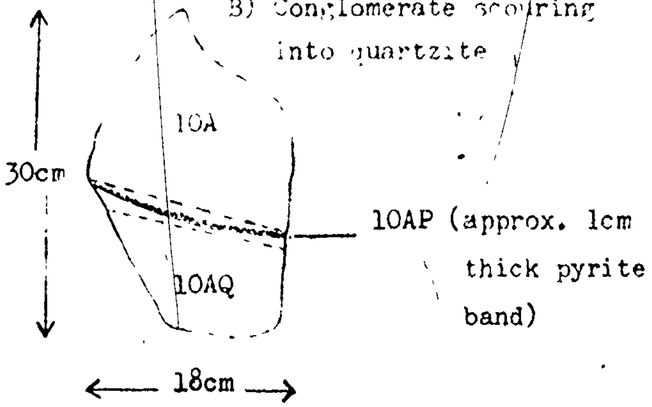
Sedimentary structure samples for major and trace element analyses.

3C, 56CB etc = Sample numbers

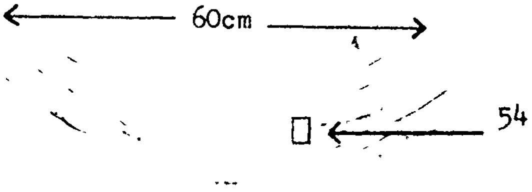
⟵ ⟶ = Sample dimensions in centimetres

A) Graded Bed

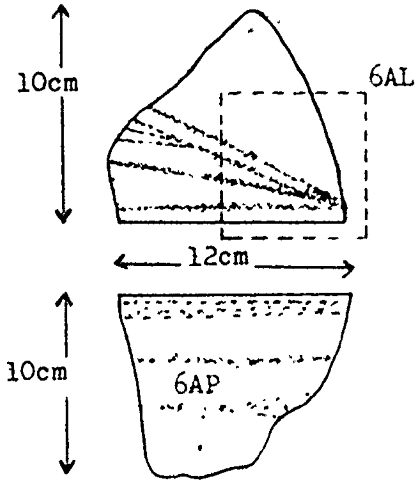




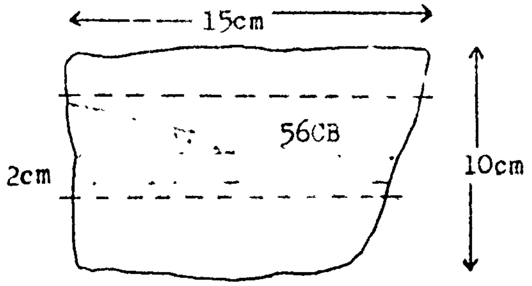
B) Conglomerate scouring into quartzite



C) Channel defined by pyrite

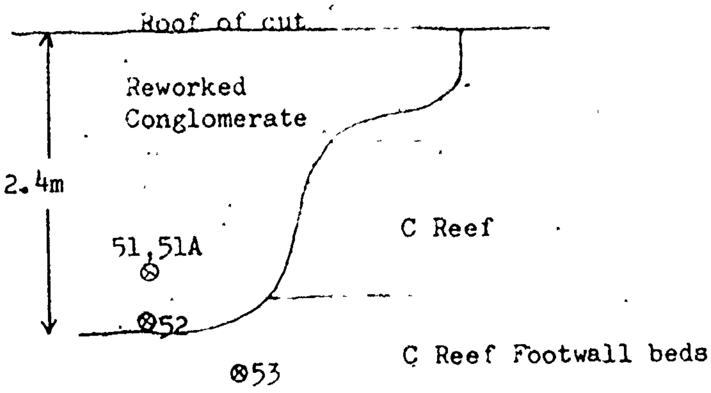


D) Pyrite banding and cross beds in quartzite



E) Pyrite cross bed

F) Large channel cutting into C Reef



Channel approx. 6m wide

2.2 Sample Preparation

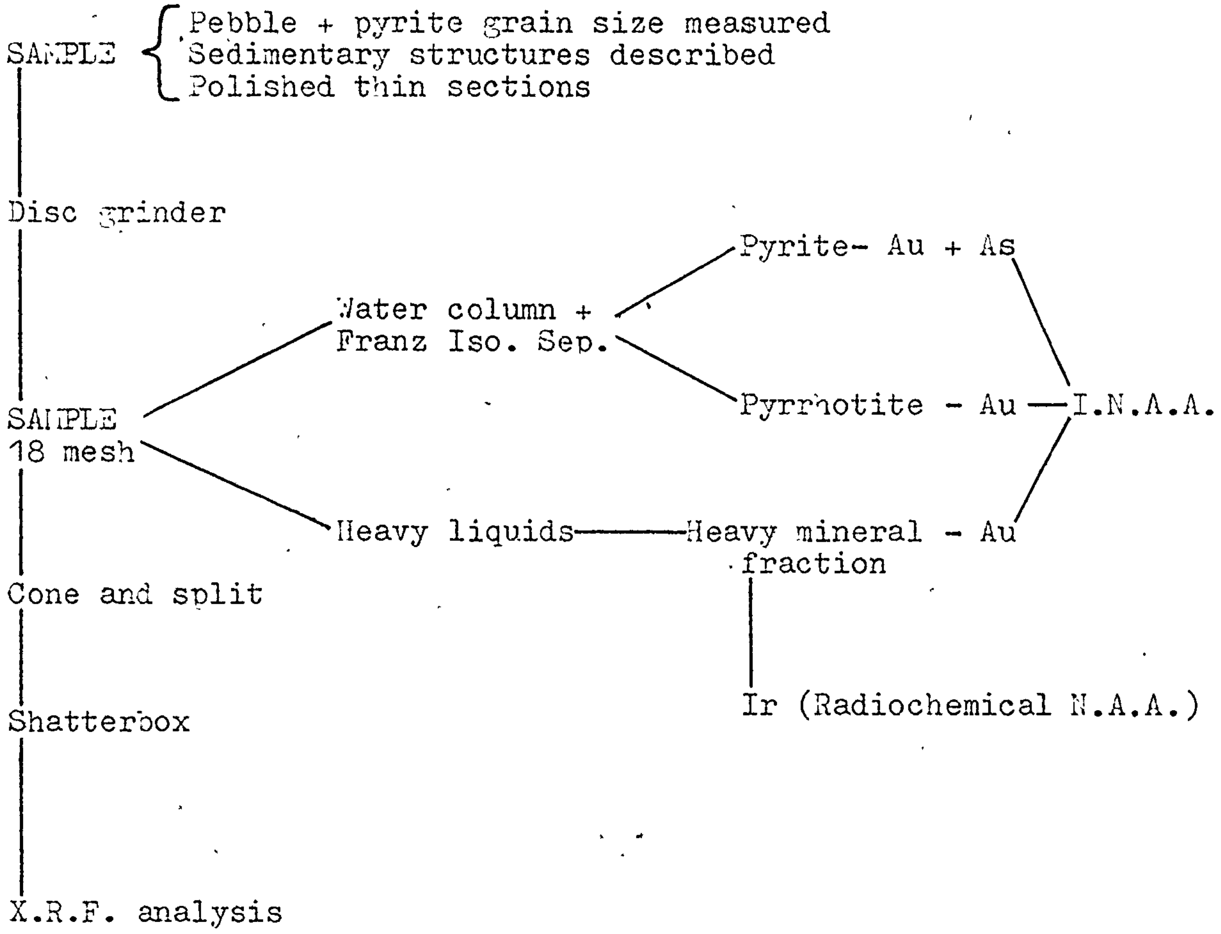
In this study the objective was to analyse mostly whole rock samples to determine the distribution of gold in relation to the sedimentology and bulk geochemistry of the ore reefs. Smaller numbers of analyses were carried out on heavy mineral, pyrite and pyrrhotite concentrates. All these separates were analysed for gold, and the pyrite separate was analysed for both gold and arsenic.

Figure 7 is a flow chart showing the experimental procedure followed in this study.

Polished thin sections were made from selected samples, and these were used for microprobe examination of the pyrite.

Before crushing and analysis, the average size of the ten largest pebbles in each sample was measured where possible. No selection or discrimination of samples according to sample size or packing index was made, but all samples, except one (ME2), consisted of single lithologies, conglomerate with varying degree of packing. Pebble size was used by Theis (1979) to characterize the depositional energy environment within a given conglomeratic unit. The pyrite grain size of a sample was recorded as being either large or small. If the majority of the pyrite grains had diameters >1mm then they were classified as large; if ≤1mm they were classified as small. Pebble and pyrite grain sizes are reported in Appendix I and II respectively.

Samples were broken with a hammer and jaw crusher, fed



Instrumental Neutron Activation Analysis - Au
(I.N.A.A.)

Figure 7: Flow chart of Experimental Procedure

into a disc grinder and ground to at least 18 mesh. A portion of this sample was taken for separation of pyrite, pyrrhotite and heavy minerals. Samples were then coned and split and sub - samples were further crushed in a shatterbox for 10 minutes and reduced to a fine powder. The resulting powders were used for X-ray fluoresence analysis (X.R.F.) and the bulk of the neutron activation analyses.

Heavy mineral concentrates were obtained by centrifuging a portion of the 18 mesh powder with tetrabromoethane, thoroughly washing the resultant separate with acetone and drying in an oven for two days. After drying, the separates were crushed in a shatterbox. Thorough drying is necessary to eliminate bromine which otherwise will generate a large 35.4 hour ^{82}Br peak upon neutron irradiation.

To obtain the pyrite and pyrrhotite separates the 18 mesh powder was seived to separate the 50-100 mesh size range. This material was put into a pressurized water column initially operated at sufficient pressure to keep all material suspended. A hand magnet was passed down the outside of the column and any pyrrhotite present was dragged down and collected in the separating funnel at the bottom. The pyrrhotite was dryed and passed through a Franz Isodynamic Separator at various slopes, but with the current switched off, to separate out non-magnetic impurities.

To obtain the pyrite the flow of water in the column was adjusted to allow the largest and heaviest grains (which were mainly pyrite) to fall to the bottom. After drying they were fed through the Franz Isodynamic Separator at various slopes and currents to separate out any paramagnetic and/or uranium bearing minerals.

The pyrite and pyrrhotite separates were checked for purity under the microscope. Minor quartz was the only contamination visible. Finally they were ground to a fine powder in an agate mortar and analysed for Au and As by instrumental neutron activation. The heavy mineral concentrates were analysed for Au and Ir.

2.3. Analytical Procedures

i) X.R.F.

The following elements were determined by X.R.F. analysis of rock powders:

Majors: SiO_2, Al_2O_3, Fe_2O_3, MgO, CaO, Na_2O, K_2O, TiO_2, MnO, P_2O_5, S.

Trace: U, Th, Zr, Ce, La, Y, Ba, As, V, Cr, Pb, Co, Cu, Zn, Ni, Nd, Nb, Rb, Sr.

(Fe_2O_3 was recalculated as FeS_2 once sulphur had been determined.)

ii) Neutron Activation.

Au and As were analysed by instrumental neutron activation. About 0.4gm of sample powder were weighed into plastic vials and irradiated in the McMaster Nuclear Reactor at high flux (10^{13}n/cm^2/sec) for half an hour along with known standards. Standards were prepared by accurately weighing an aliquot of solution, containing known concentrations of Au (≑10ppb) and As (≑9ppm) in a pre-weighed plastic vial. After reweighing pure quartz powder was added to absorb or soak up the standard fluid and thereby distribute it over the same volume as the sample. The vial was placed in a drying oven at approximately 70°C to dry the powder. When the powder had dryed the vials were sealed.

After irradiation, samples and standards were allowed to decay for two days and were then counted for one hour on a Ge Li detector (resolution of 2.1 KeV at 0.662 MeV; peak to background 15:1). Au and As were determined by counting the is on the order of a few parts per billion.

Radiochemical neutron activation analysis was carried out for iridium using the technique outlined by Crocket et al (1968). Iridium was chosen to represent the platinum group elements as it has been shown (Cousins 1973), that iridium and osmium minerals are the most abundant platinum minerals present in the Rand. Grains containing these elements are considered better survivors of sedimentary transport than grains containing Rh, Ru and Pt.

The analysis of iridium in the ore conglomerates failed due to the high uranium content of the rocks. On irradiation, relatively large amounts of fission product ruthenium were formed. Ruthenium followed iridium through the chemical separation procedure and completely dominated the gamma spectrum in the ^{192}Ir region. Attempts to remove the ruthenium by heating in a tube furnace, first at 300 C for one hour and then at 500 C for two hours, were unsuccessful.

Analyses by Martindale (1968) gave an average iridium content in pyrite of about 4ppb. Boyle (1979) quotes the average platinum and palladium content in the Elliot Lake conglomerates as being 0.005ppm and 0.006ppm respectively.

Whole rock analyses are reported in Appendix I, duplicates in Appendix III.

CHAPTER III

3.1 Major and Trace Element Distributions

Table 4 shows the correlation coefficients for the elements and parameters measured in this study. The correlation coefficients basically confirm those of Theis (1979) as shown by a comparison of some of the correlation coefficients, Table 5.

	Correlation Coefficients	
	This study	Theis(1979)
Ce - Th :	.85	.84
La - Zr :	.73	.85
U - Pb :	.94	.92
Y - Th :	.71	.85
Fe - K :	-.49	-.55
Zr - Ti :	.81	.81
Ti - Fe :	.00	-.05
Pebble size - U/Th:	.60	.79

Table 5 : Comparison of Correlation Coefficients showing Good Agreement with Theis (1979)

There are some significant differences however, with many of the correlation coefficients in this study being lower,

Table 4 : Correlation Coefficients

(Correlation coefficients $\geqslant .70$ are underlined. Samples 30E, ME1 and 23 were excluded from the calculations. Correlation coefficients with values $> .36$ are significant at a .001 level of significance)

	SiO$_2$	Al$_2$O$_3$	FeS$_2$	MgO	CaO	Na$_2$O	K$_2$O	TiO$_2$	MnO	P$_2$O$_5$	U	Th	Zr	Ce	La	Y	Ba	Pb	Cr	As
SiO$_2$	—	.09	-.33	.05	.14	.05	-.11	.43	-.13	-.31	-.07	.02	-.08	-.11	-.24	[illegible]	.04	-.40	-.19	[illegible]
Al$_2$O$_3$		—	-.55	.30	-.19	.39	.90	.19	.18	-.12	-.45	-.24	.13	-.16	-.15	-.42	.55	-.49	.04	[illegible]
FeS$_2$			—	-.07	.04	-.31	-.49	0	-.44	.17	.48	.18	-.07	.15	.16	.41	-.33	.58	.16	[illegible]
MgO				—	.11	.13	.33	.40	.27	.16	-.18	.03	.28	.16	.17	.03	.47	-.10	-.18	[illegible]
CaO					—	.07	-.15	.43	.12	.68	.12	.59	.56	.51	.47	.33	.02	.15	-.04	[illegible]
Na$_2$O						—	.45	.06	.11	-.01	-.05	.03	0	0	.01	-.08	.19	-.11	.07	[illegible]
K$_2$O							—	.21	.03	-.11	-.44	-.24	.10	-.17	-.16	-.44	.75	-.49	-.10	[illegible]
TiO$_2$								—	.04	.68	-.18	.53	.81	.68	.68	.20	.26	-.11	-.03	[illegible]
MnO									—	.06	-.13	.04	.16	.08	.07	-.06	-.02	-.15	.01	[illegible]
P$_2$O$_5$										—	.14	.84	.80	.87	.35	.47	-.01	.15	.02	[illegible]
U											—	.49	-.04	.15	.11	.77	-.32	.94	.08	[illegible]
Th												—	.68	.85	.80	.71	-.11	.49	.02	[illegible]
Zr													—	.75	.73	.41	.18	-.04	-.06	[illegible]
Ce														—	.99	.49	-.03	.17	-.03	[illegible]
La															—	.44	-.09	.14	0	[illegible]
Y																—	-.21	.77	-.06	[illegible]
Ba																	—	-.33	-.29	[illegible]
Pb																		—	.09	
Cr																			—	[illegible]
As																				
V																				
Co																				
Cu																				
Zn																				
Nd																				
Nb																				
Rb																				
Sr																				
Ni																				
Pebble size																				
Au																				
U/Th																				

30

U	Th	Zr	Ce	La	Y	Ba	Pb	Cr	As	V	Co	Cu	Zn	Nd	Nb	Rb	Sr	i	[illegible]	Au	[illegible]
-.07	.02	-.08	-.11	-.24	[illegible]	.04	-.40	-.19	-.55	-.31	-.24	-.21	.02	-.05	-.49	-.15	.22	-.61	.01	[illegible]	[illegible]
-.45	-.24	.13	-.16	-.15	-.42	.55	-.49	.04	-.60	.36	-.54	-.12	-.15	-.17	-.51	.82	.16	-.40	-.43	[illegible]	-.49
.48	.16	-.07	.15	.16	.41	-.33	.58	.16	.75	.08	.46	.24	.06	.12	.66	-.28	-.27	.70	.18	.67	[illegible]
-.18	.03	.28	.16	.17	.03	.47	-.10	-.18	-.16	.49	-.16	-.09	.36	.15	-.11	.29	.17	0	[illegible]	.12	-.19
.12	.59	.56	.51	.47	.33	.02	.15	-.04	.06	.35	.09	-.06	.08	.60	.29	-.23	.55	.11	.02	[illegible]	-.15
-.05	.03	0	0	.01	[illegible]	.19	-.11	.07	-.23	.16	-.05	-.12	-.16	0	-.16	.42	.13	.12	-.10	-.09	-.11
-.44	-.24	.10	-.17	-.16	-.44	.75	-.49	-.10	-.47	.37	-.49	-.13	-.15	-.16	-.51	.83	.28	.26	-.41	[illegible]	-.43
.18	.53	.81	.68	.68	.70	.26	-.11	-.03	-.04	.89	-.14	-.08	.28	.67	.21	.10	.47	.07	-.45	-.12	-.60
-.13	.04	.16	.08	.07	-.06	-.02	-.15	.01	-.37	-.08	-.02	-.19	.11	.10	-.24	-.05	.28	.07	0	[illegible]	-.18
.14	.94	.80	.87	.85	.47	-.01	.15	.02	.18	.57	.06	-.05	.03	.91	.35	-.15	.61	.44	-.38	.06	[illegible]
—	.49	-.04	.15	.11	.77	-.32	.94	.08	.61	-.11	.43	.01	-.06	.21	.78	-.10	-.04	[illegible]	.13	.69	.60
	—	.68	.85	.80	.71	-.11	.49	.02	.27	.42	.70	-.06	.01	.91	.59	-.18	.46	[illegible]	-.38	[illegible]	-.15
		—	.75	.73	.41	.18	-.04	-.06	-.08	.67	-.09	-.06	.11	.77	.20	-.02	.55	[illegible]	-.45	[illegible]	[illegible]
			—	.99	.49	-.03	.17	-.03	.20	.51	.20	-.08	.07	.98	.38	-.22	.43	.76	-.37	.04	-.40
				—	.44	-.09	.14	0	.17	.52	.16	-.03	.07	.94	.35	-.21	.38	.37	[illegible]	.03	-.41
					—	-.21	.77	-.06	.46	.13	.43	.04	.09	.56	.72	-.28	.18	.51	[illegible]	[illegible]	[illegible]
						—	-.33	-.29	-.31	.38	-.30	-.08	-.11	-.04	-.29	.65	.42	[illegible]	-.16	[illegible]	-.31
							—	.09	.61	.04	.45	.07	.10	.22	.84	-.19	-.09	.59	[illegible]	.74	.54
								—	-.06	.01	-.19	.02	-.13	-.06	.04	.02	.23	.06	-.17	[illegible]	0
									—	0	.57	.02	.09	.19	.63	-.26	-.12	.74	.26	.56	.47
										—	-.16	-.02	.14	.51	.21	.33	.39	.16	[illegible]	.02	.47
											—	.03	.07	.21	.47	-.44	.03	.42	[illegible]	.47	.33
												—	-.02	-.08	.04	-.11	-.13	.33	[illegible]	.15	.12
													—	.06	.05	-.20	-.02	.11	.07	[illegible]	.07
														—	.41	-.22	.54	.75	[illegible]	.06	-.36
															—	-.28	0	.71	.10	.60	.30
																—	.14	.03	[illegible]	-.12	-.20
																	—	-.22	[illegible]	-.18	-.29
																		—	[illegible]	.59	[illegible]
																			—	.13	.40
																				—	.41
																					—

2 OF 2

Table 6.

	Correlation Coefficients	
	This study	Theis (1979)
La - [illegible] :	-.13	.45
U - Zr :	-.04	-.46
Pb - Ce :	.17	-.45
Pebble size - Y :	.00	-.40
Pebble size - Ti :	-.45	-.80
Pebble size - Ce :	-.37	-.85

Table 6 : Comparison of Correlation Coefficients showing Poor Agreement with Theis (1979)

This is probably because Theis (1979) sampled only well packed conglomerates from one ore reef (Rio Algom - Denison Reef), while in this study a far more heterogenous sample population was employed. A number of reefs were sampled and no sample selection based on sample size or degree of packing were used. Also some quartzite data were included in most calculations.

Figures 8 and 12 are plots of selected element pairs from which it can be seen that all the reefs investigated appear to follow similar trends and distributions, indicating they have a common mode of origin.

In as much as part of this study overlaps the work done by Theis (1979), a brief summary of his results will be presen

-ted.

Theis attempts to relate element analyses to mineralogical and sedimentological features within the Rio Algom - Denison Reef. He bases his conclusions on the following assumptions. Ce and La, the two most abundant rare-earth elements commonly found in the mineral monazite, are dominantly controlled by the variation in the abundance of monazite. Variations in the content of Zr and Ti reflect the abundance of zircon and "brannerite" respectively. Fe reflects the pyrite content and K the sericite content.

Theis findings indicate that the abundance of Th and Y are controlled by uraninite when the pebble sizes are large and by monazite when small. Similarly, U abundance is controlled by uraninite when pebble sizes are large and by "brannerite" when small. All Pb is radiogenic decay product of uranium. The U/Th ratio increases with increasing pebble size and is explained as a result of the more dense uranium minerals being deposited in a higher energy environment, while the less dense thorium minerals are carried to a lower energy environment. This is also the explaination given for the decrease in the U/Th ratio down the paleo-slope. Theis divides the elements analysed in his study into two groups. One group includes Zr, Ce, La, Ti and to a lesser extent Th, Y and K. The elements in this group correlate with each other, and the contents of these elements increase with decreasing pebble size i.e. the content of monazite, zircon, "brannerite" and sericite increase with decreasing pebble size. U, Pb and the U/Th ratio make up the second group. These parameters increase with a general increase in pebble size. Theis summarized his findings by noting that all elements determined, with the exception of Fe, showed some degree of correlation with depositional environment as reflected by the pebble size.

The findings obtained in this study tend to confirm most of the above relationships. A relationship exists between the distribution of Zr, Ce, La, TiO_2 and Th as shown by their high, positive correlation coefficients in Table 4. K_2O and Y however, correlate very poorly with this group of elements. Y in fact correlates better with Theis second group of elements, U, U/Th and Pb, the existance of which is confirmed by the correlation coefficients obtained in this study. The present findings however, do not stress relationships of element abundance with pebble size to such a large extent. This could be due to the more heterogenous sample population. Results of this study would also tend to confirm some of Theis assumptions. The strong correlation of P_2O_5 with Ce and La, .87 and .85 respectively, indicates most Ce and La are present in the mineral monazite. Correlation of K_2O with Al_2O_3 (.90) confirms most potassium is present in the sericitic matrix.

Interpretation of the findings obtained in this study are still based on the assumptions the Fe (quoted as FeS_2) represents the pyrite content, Zr the zircon content and TiO_2 the "brannerite" content. From the element analyses completed in this work, the following relationships are established: SiO_2 and FeS_2 are mutually exclusive as shown by their strong negative correlation (-.88). Al_2O_3 correlates very well with K_2O (.90), as does Rb (.83) and Ba (.75), indicating these elements all occur in the same mineral, sericite. MgO shows no clear relationship with any of the other elements. Its strongest correlations are with TiO_2 (.40) and Ba (.47) indicating that it may occur both in "brannerite" (present in the rutile phase?) and sericite (possibly minor chlorite present). CaO and Sr correlate strongest with P_2O_5, with correlation coefficients of .68 and .61 respectively, suggesting most calcium and strontium are present in the mineral monazite. There may also be minor apatite present (Robertson 1976). Na_2O correlates most strongly with K_2O (.45) indicating most sodium is probably present in the sericitic matrix of the ore conglomerates. Little can be said about MnO except that it is exceptionally uniform throughout the ore reefs.

The rare-earth Nd correlates well with Ce (.98) and La (.94) as expected. In these ores Nb follows U, the correlation coefficient being .78. Nb appears unrelated to TiO_2 as shown by their poor correlation of .21. The correlation coefficients of As and Ni with FeS_2, .75 and .70 respectively, indicates the distribution of these elements is controlled by the distribution of pyrite and that they are mostly contained within the pyrite. The same might be argued for Co, but the poor correlation with FeS_2 (.46) may indicate the presence of

Co in some other minerals e.g. cobaltite (Robertson 1976). V shows a strong correlation with TiO_2 (.89) and is present mostly in the rutile phase of the "brannerite". Cu and Zn show no relationship with any of the other elements and probably represent trace constituents within the sericite and/or pyrite. Where high values are recorded they could indicate minor amounts of chalcopyrite and sphalerite. The distribution of Cr is problematic. It was thought that Cr might reflect the distribution of chromite within the ore reefs, but this does not appear to be the case as no relationship is apparent with heavy minerals such as zircon and monazite. In fact Cr shows no relationship with any of the other elements. Theis (1979) reports that his Cr analyses showed marked contamination from use of a disc grinder and the same problem may have occurred in this study. According to Boyle (1979) Cr in this type of deposit occurs mainly in the micas (sericite) and pyrite with small amounts in a variety of other minerals including chromite.

To summarize: The distribution of

a) Fe, S, As, Ni (and Co) are controlled by pyrite.
b) Al, K, Rb, Ba (and Na) are controlled by sericite.
c) P, Ce, La, Nd, Ca and Sr are controlled by monazite.
d) Ti and V are controlled by "brannerite".
e) Zr is controlled by zircon.
f) U, Nb and Pb are controlled by uraninite and "brannerite".
g) Y and Th are controlled by uraninite and monazite.
h) Mg is probably controlled by "brannerite" and sericite.

The mineralogical controls on Cu, Zn, Cr, and Mn have not been discerned from the correlation matrix.

Figures 8A to 8E illustrate some of the relationships described above.

Figures 8A - E

Plots of : Ce versus P_2O_5

V versus TiO_2

Ba versus K_2O

Rb versus Al_2O_3

As versus FeS_2

× = C Reef + Lower and Upper Denison Reefs

○ = A Reef

▽ = Upper Reef, New Quirke Mine

□ = Nordic Zone

△ = Reworked C Reef

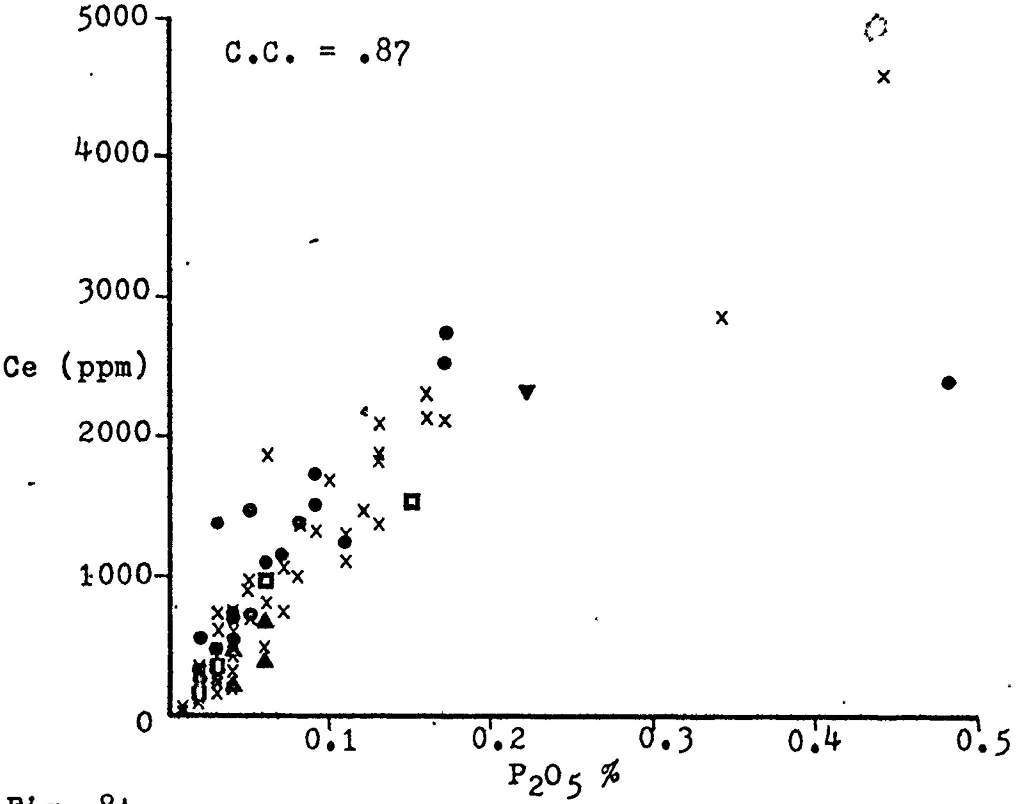
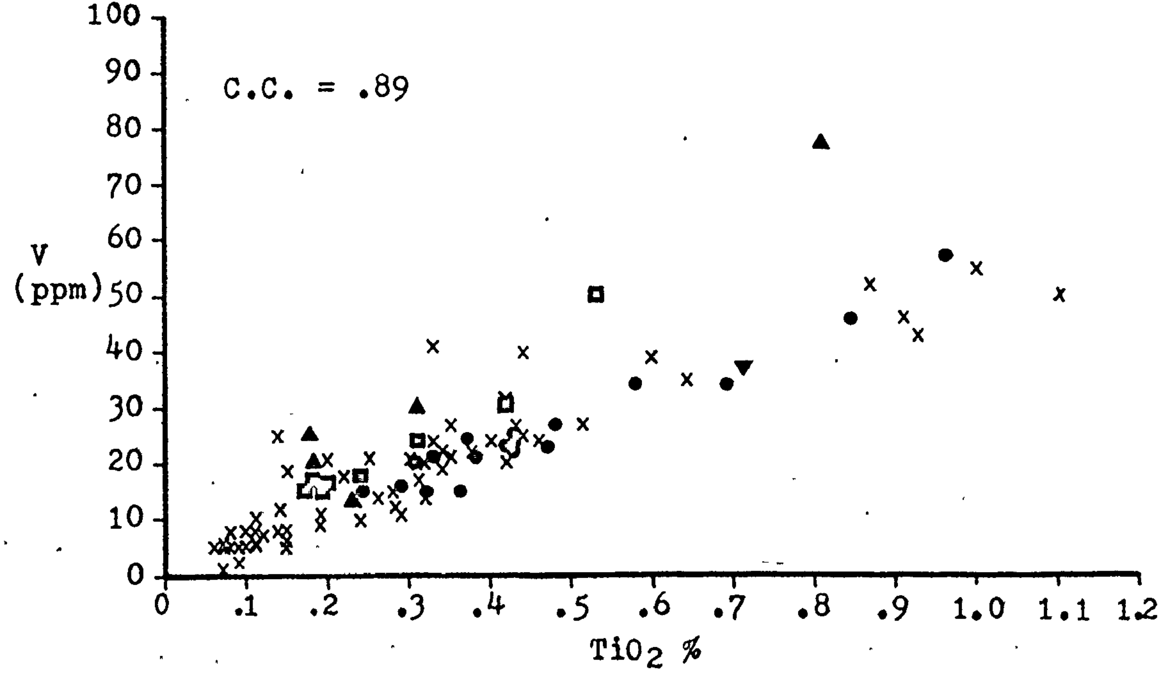
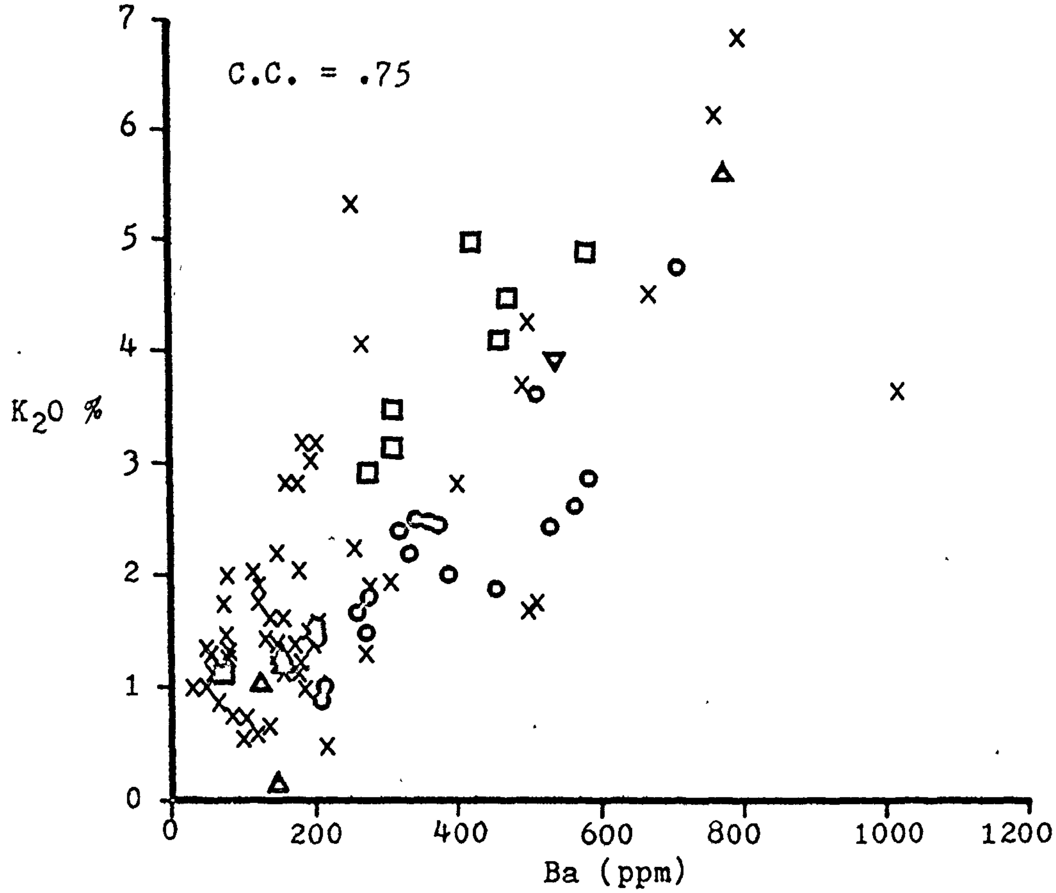

C.C. = .87
5000
4000
3000
2000
1000
0
Ce (ppm)
0.1
0.2
0.3
0.4
0.5
P_2O_5 %

Fig. 8A

Fig. 8B

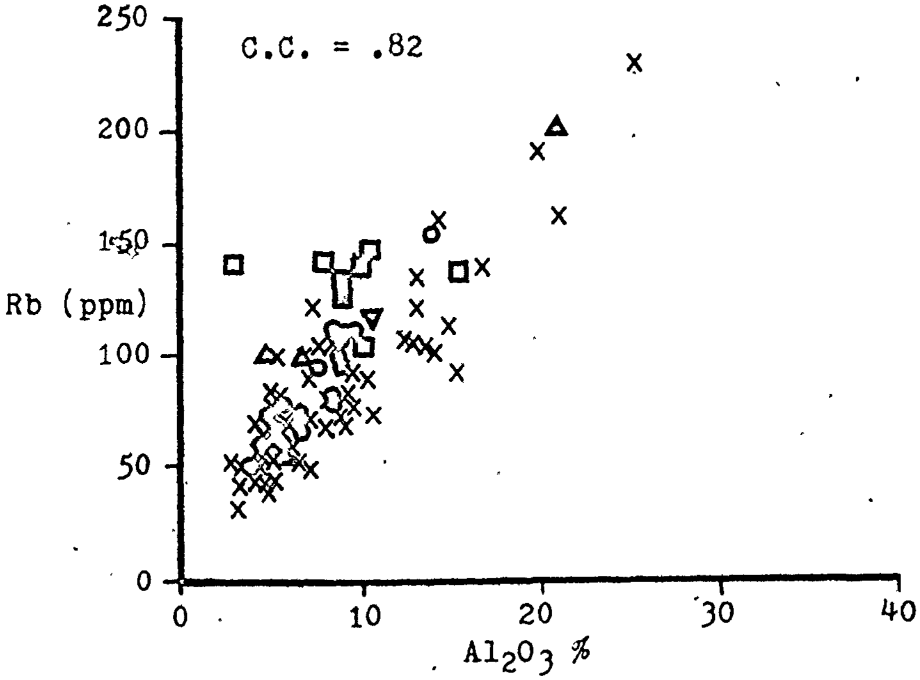

C.C. = .89
100
90
80
70
60
50
40
30
20
10
0
V (ppm)
0
.1
.2
.3
.4
.5
.6
.7
.8
.9
1.0
1.1
1.2
TiO_2 %



C.C. = .75
K$_2$O %
7
6
5
4
3
2
1
0
0
200
400
600
800
1000
1200
Ba (ppm)

Fig. 8C

Fig. 8D



C.C. = .82
Rb (ppm)
250
200
150
100
50
0
0
10
20
30
40
Al$_2$O$_3$ %

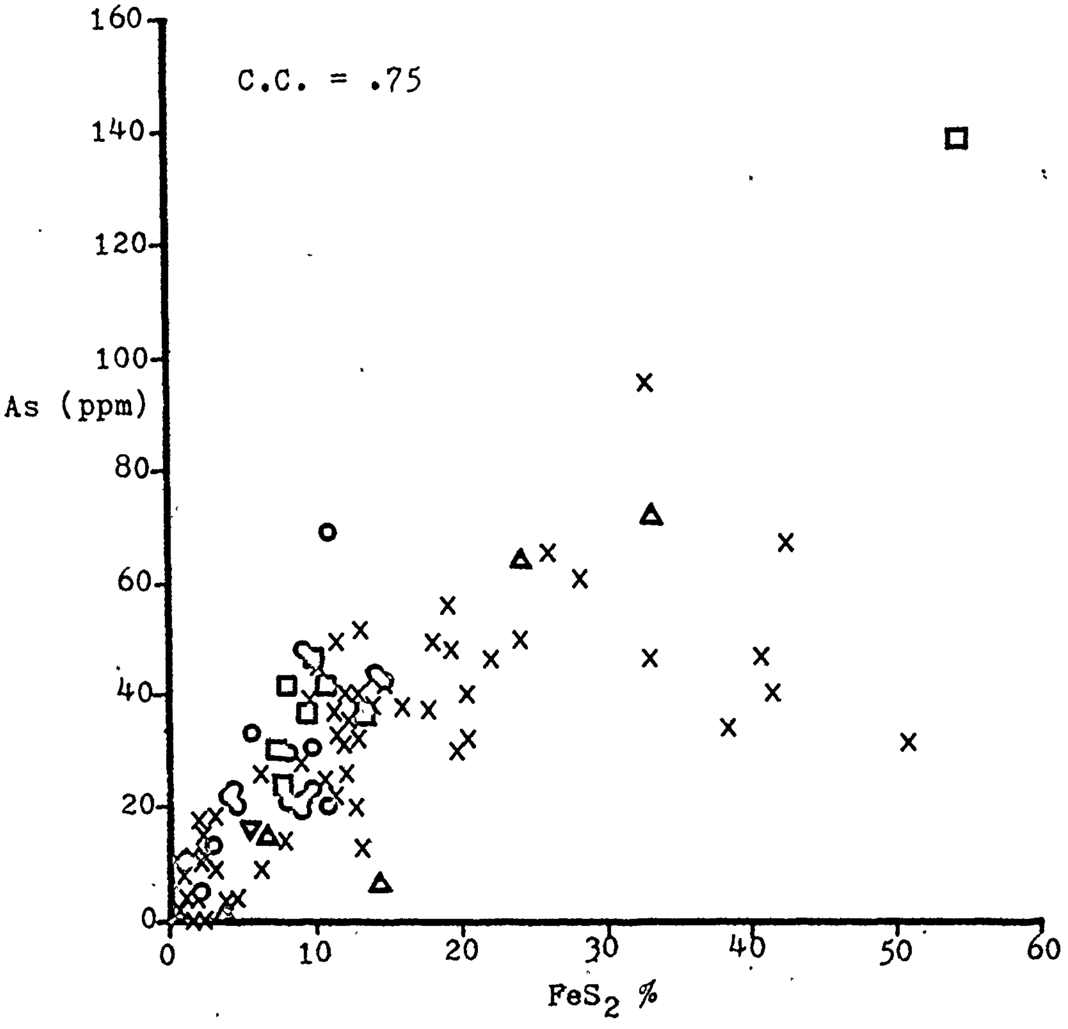


Fig. 8E

3.2 Microprobe analyses of pyrite

Electron microprobe (Acton Lab. Inc. MS 64) analyses were carried out on a number of pyrite grains to see if any zoning was discernable. Ni, Co, Ti and S were chosen for analysis.

If the rounded pyrite grains showed any distinct symmetrical zoning, it could be taken as evidence for a non - detrital origin. If detrital, the break up and resultant release of pyrite from the source rock should not result in any symmetrical zoning within grains.

Three rounded grains and two euhedral (cubic) grains were studied. Although all four elements could be detected in the pyrite above background levels in the quartz - sericite matrix, no zoning was discernable in either type of pyrite in that only flat traces were obtained (see Figure 9). In the case of Co, Ni, and Ti, this was at the lowermost limits of detection.

The lack of any zoning in the rounded pyrite might be taken as evidence for a detrital origin, but the non-detrital euhedral pyrite shows exactly the same flat distribution. In reality the sensitivity of the instrument used was probably too low to detect any change in element content within the pyrite.

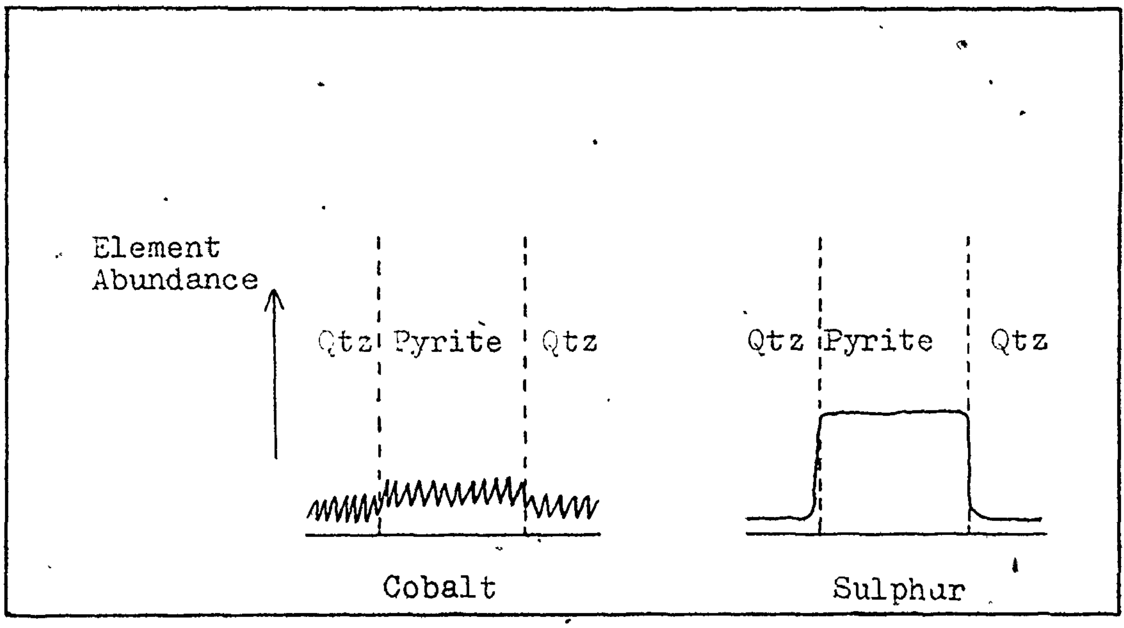


Figure 9 : Microprobe traces of Co and S in pyrite
(The traces for Ni and Ti were similar to Co)

CHAPTER IV

Distribution of Carbon

Carbon in the Elliot Lake ores has received relatively little attention. Carbonaceous material in this type of deposit is generally referred to as "radioactive hydrocarbon" or "thucolite" (Robertson 1976).

The importance of carbonaceous material in localizing gold and uranium in the Witwatersrand has been stressed in the literature (eg. Pretorius 1974). Several possible mechanisms have been proposed, including physical trapping, biogenic fixation and transportation by organic colloids. Hallbauer and van Warmelo (1974) claim to recognize structures of biological origin in the seams of hydrocarbon, which are thought to be the remains of ancient algal mats. Muir (1978) disputes this, arguing the method of sample preparation was exceptionally severe, and Cloud (1976) has been able to demonstrate that identical structures can be produced in the laboratory abiogenically. The channel facies of a fluviatile system is also unfavourable for the growth of algal mats. The argument that the algal mats grew on the shales at the top of the preceding sedimentary cycle is mitigated by the fact that many seams of hydrocarbon are actually contained within the conglomerates. Hoefs and Schidlowski (1967) found that the isotopic composition of carbon is very similar to present day crude oils and bituminous substances of obvious biological derivation. Thus the evidence favours the hypothesis that the hydrocarbon now present in the conglomerates represents remobilized biogenic carbon which migrated there in either a gaseous or liquid phase.

Although different types of carbonaceous material have been reported from the Elliot Lake ores (Roscoe 1957; Ruzicka and Steacy 1976), occurrences are quite rare. The most common occurrence is as nodular, warty thucolite occurring in fault gouge, fractures, and vugs or cavities at the margins of diabase dykes (observation by the author) cutting the ore reefs. Other rare occurrences are as seams within the ore reefs (Ruzicka and Steacy 1976). The presence or non - presence of hydrocarbon has most often been made simply on the premise of visual observation. From the fact most thucolite seen, occurs in faults, fractures and dykes cutting the ore reefs, it is logical to assume these locations have acted as points of concentration for hydrocarbon, and that this hydrocarbon has been scavenged from the adjacent ore reefs.

Based on the assumption the ore conglomerates might contain more hydrocarbon than was immediatly obvious in hand speciman or in underground mine exposures, the decision was taken to analyse the ore for carbon. The analyses were attempted using a Leco induction furnace and WR12 carbon determinator. Due to the high sulphur content of the samples however, carbon could not be determined. The sulphur dioxide released on combustion swamped the manganese dioxide trap on the determinator and gave anomalous results.

Patchett (1960) studied the radioactive minerals from the Elliot Lake ore conglomerates and noted individual grains of uraninite had a characteristic texture. This texture was distinguished by very minute curving and irregular fractures filled with another phase. The number of these imperfections increased towards the centre of a grain giving the appearance of porous grains with purer rims. (See Figure 10) Patchett restricted his study to only three samples, but it is well established that many uraninite grains show this texture, (Hart et al 1955; Heinrich 1958; Ramdohr 1969 figure 615; Theis 1979).

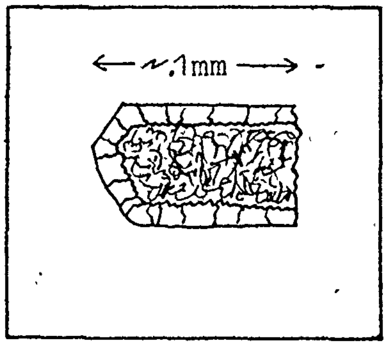


Figure 10 : Sketch of uraninite grain (from Patchett 1960, figure 19)

Analyses by Theis (1979) indicate that the porous areas within uraninite grains are lower in uranium, but richer in thorium, than the rims (see Table 7). This is in contr

-adiction to Ramdohrs (1969) suggestion that the intact rims of the uraninite grains are due to their higher thorium content as compared to the cores.

Rims		Porous areas	
UO_2	ThO_2	UO_2	ThO_2
65.44	7.17	45.35	14.71
63.51	5.53	32.40	16.51
62.62	6.35	39.46	19.18
64.39	7.75	34.64	13.88
62.67	5.31		
62.80	5.97		

Table 7 : Examples of UO_2 and ThO_2 contents of uraninite rims and porous areas within uraninites (Data from Theis 1979)

Patchetts examination of the uraninites by reflected light revealed the impurities remained completely isotropic under crossed nicols indicating they were not quartz. Microprobe analyses by Theis (1979) on other Elliot Lake uraninites support this observation. X-ray powder patterns of uraninite obtained by Patchett, showed only the diffraction lines of UO_2 and sometimes those of galena and pyrrhotite. Difficulty was experienced in converting UO_2 to U_3O_8 by heating, a reaction that normally proceeds rapidly, leading to the conclusion that some factor had retarded the oxidation. On analysis, the uraninite was shown to contain on the average 15% carbon by volume. No carbonate diffraction lines were observed on the powder patterns and CO_2 inclusions could not have supplied that much carbon. Patchett concluded that carbon or hydrocarbon was a major impurity within the uraninite grains and that the cores are carbon or hydrocarbon rich. Chemical analyses of the low uranium areas by Theis (1979) yielded totals well below 100% due to the presence of elements undetectable by the electron microprobe. Carbon may have been one of the elements undetected. It is of interest to note that on a microprobe analysis of a very dark phase in the low uranium area within a uraninite grain, Theis had to move the beam continuously to avoid burning. This indicates the mineral phase under the beam was easily volatilized - possibly hydrocarbon.

Patchett (1960) favours a hypothesis in which uraninite was precipitated from solution by carbon. The disseminated nature of uraninite is believed to be related to minute scattered amounts of carbon in the conglomerates and the thorium content as due to the incorporation of thorium released from altered monazite.

If uraninite formed by adsorption of uranium onto hydrocarbon collectors, no place is more likely for this to happen than in the ore conglomerates themselves. Hydrocarbon could have migrated into the conglomerates and been distributed preferentially in rocks of higher permeability and porosity, the better sorted, better packed conglomerates with larger pebbles. The negative correlation between K_2O and pebble size indicates there is less fine grained material in the interstices of the coarser conglomerates. Uranium would be concentrated by hydrocarbon in the coarser conglomerates while thorium would mostly follow monazite, and as previously shown, the concentration of monazite increases with smaller pebble sizes and poorer packing (Robertson 1962; Theis 1979). In effect the U/Th distribution could be the result of two processes - one during deposition of the conglomerates and the other after burial. It would be of interest to see how the U/Th ratio varied with the carbon/phosphorus ratio. The presence of carbon and titanium in the conglomerates of the Matinenda Formation may have acted as "collectors" for any uranium in solution which was moving through the Huronian rocks. It need not be postulated that the oligomictic conglomerates were any more permeable than other rocks in the Huronian section, only that they contained the highest amounts of carbon and altered ilmenite and were thus able to trap large amounts of uranium.

That carbon is important in the formation of this type of deposit is of little doubt and the distribution and relation of carbon to the other elements warrents further study.

CHAPTER V

The Distibution of Gold

5.1 Previous Work

Comparitively little work has been done on the occurrence of gold in the Elliot Lake mining camp, which is suprising, since the analogies of Elliot Lake to the Witwatersrand would tend to suggest the possibilities of finding gold at Elliot Lake.

Hart et al (1955) noted gold was present in the radioactive conglomerates, but that although the occassional good assay was obtained, the overall gold content was low and the distribution erratic. Davidson (1957) described the occurrence of gold at Elliot Lake as being "patchy". Heinrich (1958) stated the gold values were low and erratic, 0.02 to 0.03oz Au/ton. Robertson and Steenland (1960) noted gold was not present at Elliot Lake except in sporadic traces and its manner of occurrence was unknown.

Martindale (1968) obtained an average gold content for pyrite of 656ppb (0.02oz Au/ton) in a study of the gold content of pyrite from the ore reefs. Roscoe (1969) noted that although gold was concentrated in the conglomerates most samples were reported to contain "trace", "nil" or 0.005oz Au/ton. However, many samples contained about 0.01oz Au/ton and a few up to 0.04 oz Au/ton. The highest gold assays reportedly came from his

Type I conglomerate, $U_3O_8 > Ti > ThO_2 > Zr$.

More recently Boyle (1979) described the gold values as erratic and at present of no consideration in the economic value of the ores. He quotes an average gold content for most samples as being 0.09ppm (0.003oz Au/ton), with locally high assays from samples where sulphides such as galena, sphalerite, cobaltite etc occur. He states most of the gold (and silver) are present in the pyrite and other sulphides, an average figure being 0.64ppm (0.019oz Au/ton), which is in good agreement with Martindales average of 0.656ppm.

Native gold has been reported sporadically in some of the ores (Robertson 1976).

This general paucity of gold in the Elliot Lake ores is often explained as due to a lack of gold deposits in the provenance area to the northwest. Roscoe (1969) noted gold deposits are common through a belt extending east from Michipicoten, about 100 miles north of Elliot Lake, but are rare or absent in greenstone belts further south. A more recent hypothesis is that proposed by Pretorius (1980). If the Elliot Lake quartz pebble conglomerates are assumed to have been deposited on the shallow limb of an asymmetrical sedimentary basin, any gold present in the provenance area may have been eroded, transported and deposited deeper within the sedimentary pile, long before deposition of the Elliot Lake conglomerates, i.e. any gold deposits which might exist, may possibly lie deeper in the succession, to the south under Lake Huron.

5.2 Concentration and Distribution of Gold within the Ore Reefs

The results of the whole rock gold analyses are reported in Appendix I. Duplicates are reported in Appendix III. In respect to gold, far greater heterogeneity exists in the sample after coning and splitting than after crushing in the shatterbox.

The ore reefs investigated gave a range in gold values from 5ppb to 987ppb (0.0002 to 0.03oz Au/ton). Figure 11 shows a plot of sample frequency versus gold content for all the data.

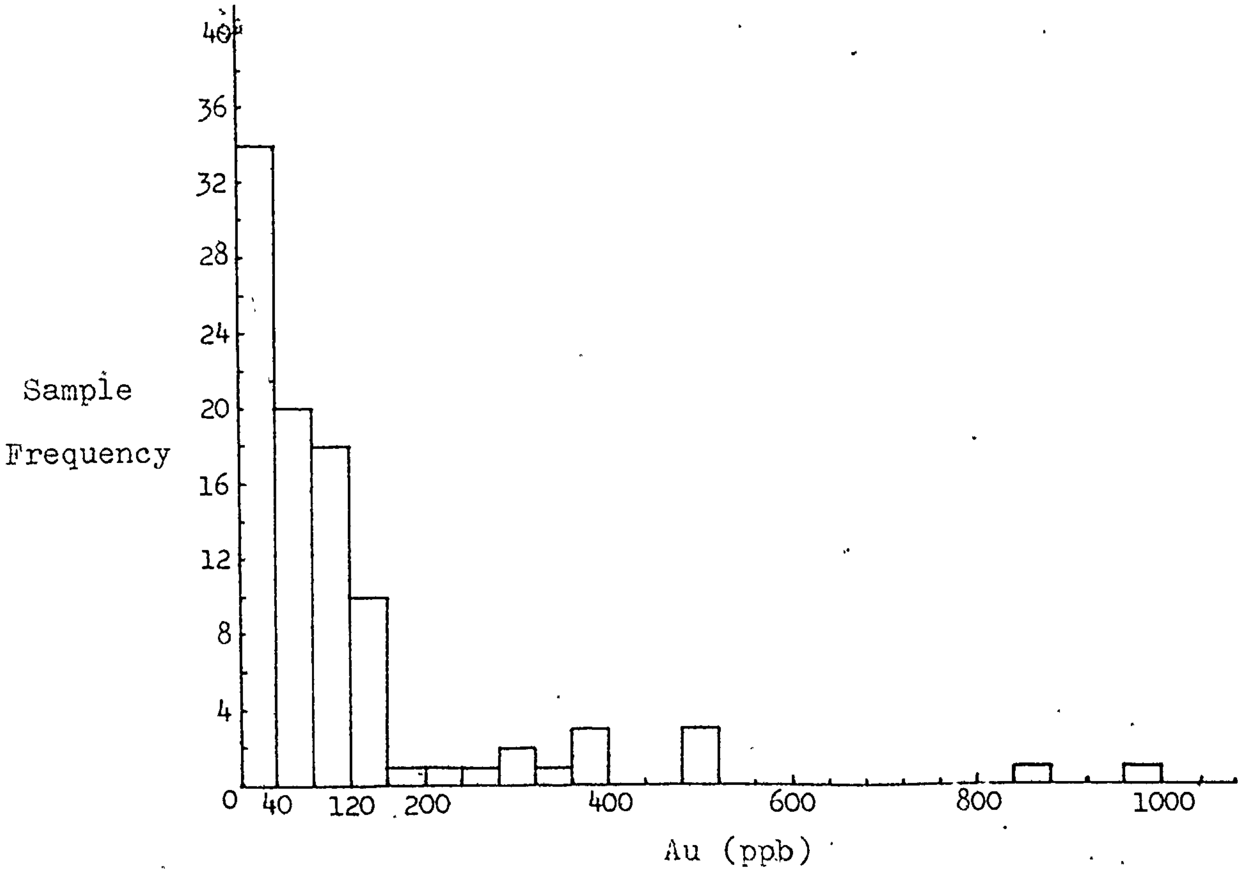


Figure 11 : Sample Frequency versus Gold Content

The ranges and averages for the different reefs are shown in Table 8.

Reef	Range(ppb)	Average(ppb)
C Reef + Lower and Upper Reefs (Denison Mine)	5 - 510	111 (63)
A Reef	16 - 130	73 (19)
Nordic Zone	16 - 388	117 (8)
Reworked C Reef (Ramsay Lake Cong.)	19 - 987	400 (5)
Upper Reef (New Quirke Mine)	- - -	46 (1)

Table 8 : Ranges and average gold contents for the different reefs (Number of analyses in brackets)

The average for the Reworked C reef is in reality probably much lower than the figure quoted as two out of five samples had anomalously high gold contents.

From the correlation matrix (Table 4), the elements correlating most strongly with gold are Pb, Nb, U, FeS_2, SiO_2, Y, Ni, As and to a lesser extent Co. Their correlation coefficients with gold are shown in Table 9. Plots of Pb, Nb, U, FeS_2 and SiO_2 versus gold are illustrated in Figures 12A to E.

It is apparent from the graphs that although FeS_2 versus Au only gives a correlation coefficient of .67, it appears to give the strongest correlation. The reason for the apparent lower correlation is five points lying well above the main distribution, thus considerably lowering the correlation coefficient. If these five points are considered to be anomalous in respect to gold and are excluded from the calculation, a strong FeS_2 versus Au correlation of .88 is obtained. The coefficients for the other elements are recalculated under the same conditions and are shown in Table 9. U and its associated elements Pb, Nb, and Y decrease, as do Ni and Co. SiO_2 and As increase.

Element	Corr. Coeff. with Au	Corr. Coeff. omitting 5 samples (see text)
Pb	.74	.59
Nb	.69	.63
U	.68	.48
FeS_2	.67	.88
SiO_2	-.59	-.80
Y	.59	.34
Ni	.59	.53
As	.56	.62
Co	.46	.41

Table 9 : Correlation coefficients of selected elements against gold

Thus, FeS_2 gives the strongest correlation with gold and suggests that most of the gold in the ore conglomerates is present in the pyrite and the concentration of gold is related to pyrite content. The correlation of gold with U and its

Figures 12A - E

Plots of : Au versus Pb
Au versus Nb
Au versus U
Au versus FeS_2
Au versus SiO_2

× = C Reef + Lower and Upper Denison Reefs
○ = A Reef
▽ = Upper Reef, New Quirke Mine
□ = Nordic Zone
△ = Reworked C Reef
◌ = Samples omitted in recalculation of correlation coefficients

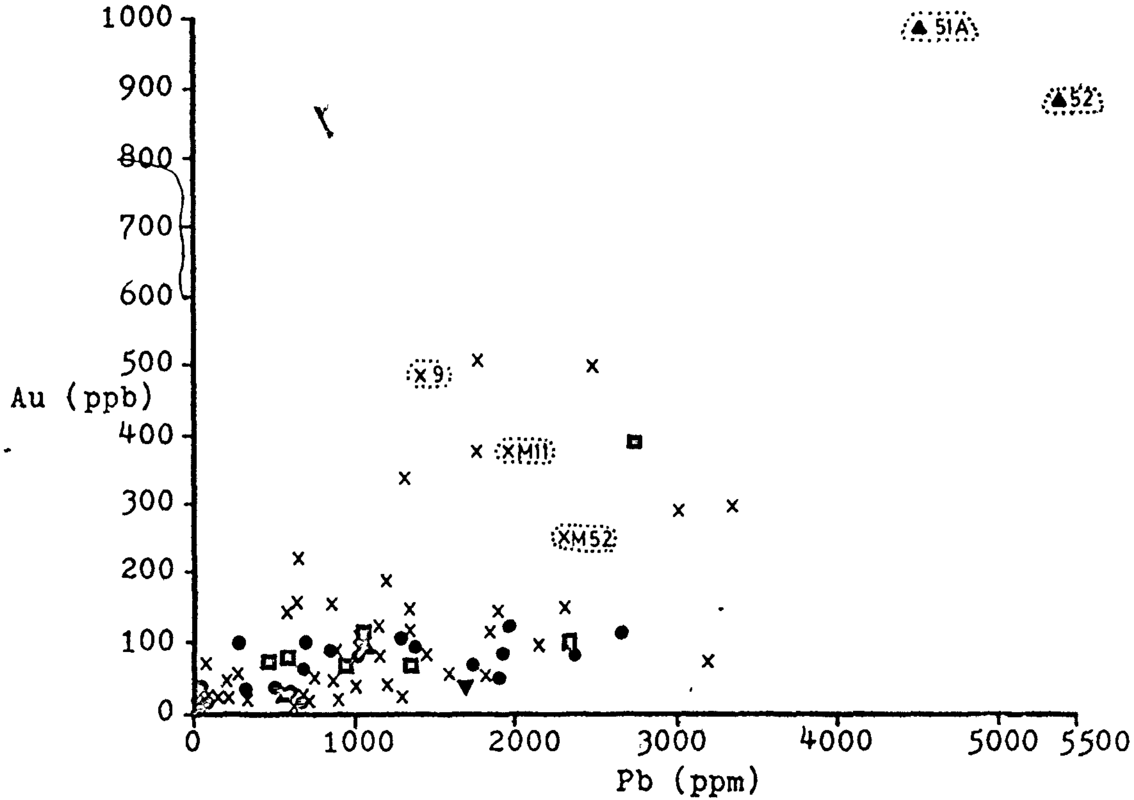
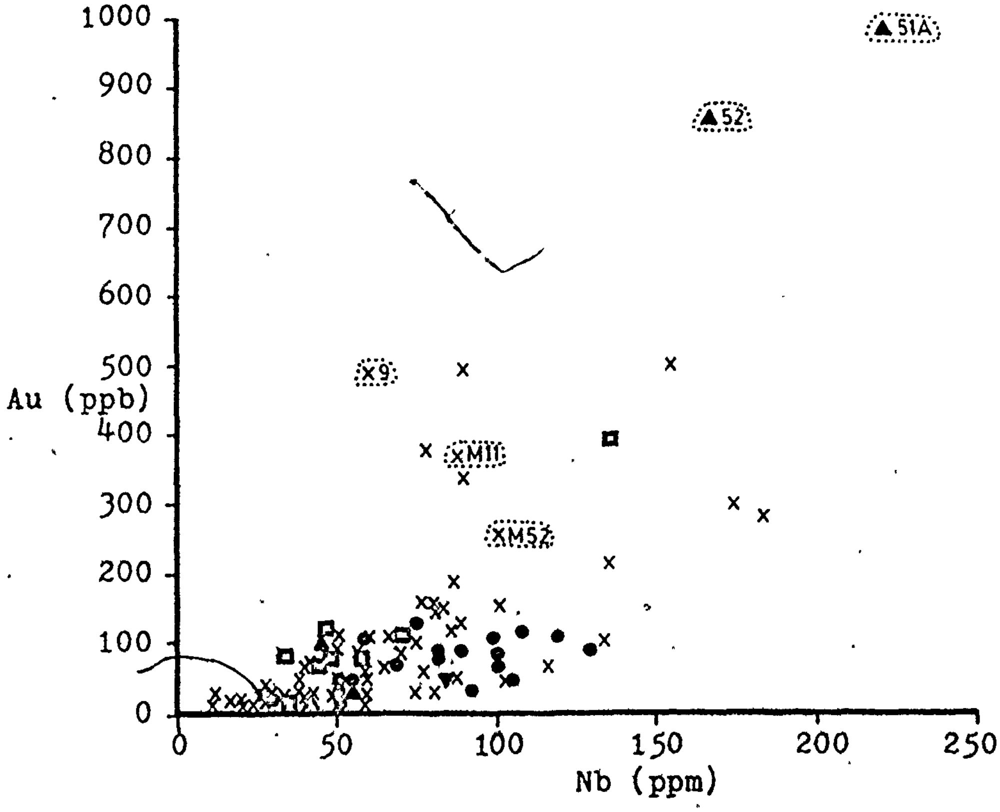


Fig. 12A

Fig. 12B



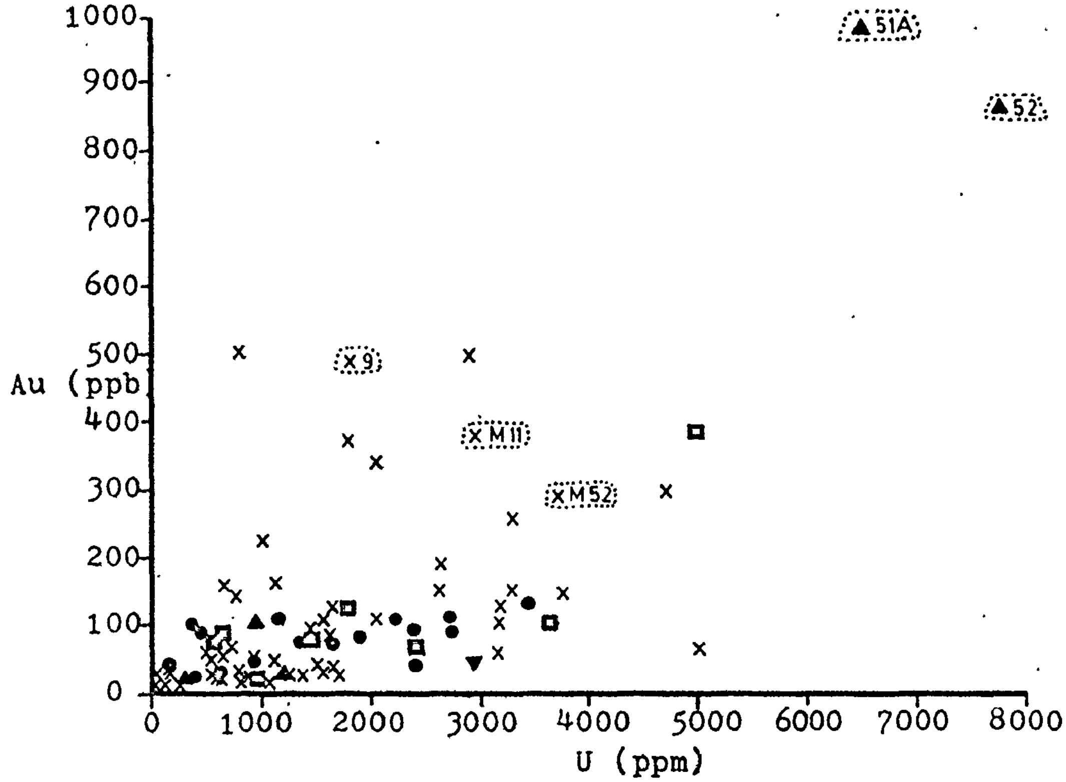
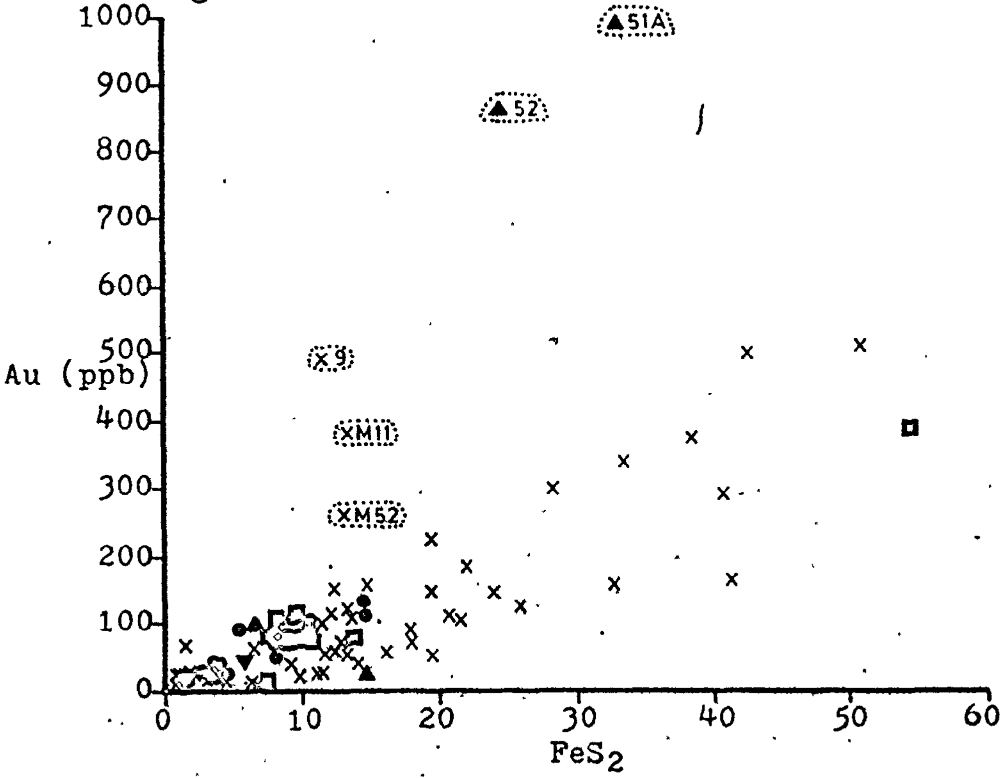


Fig. 12C

Fig. 12D



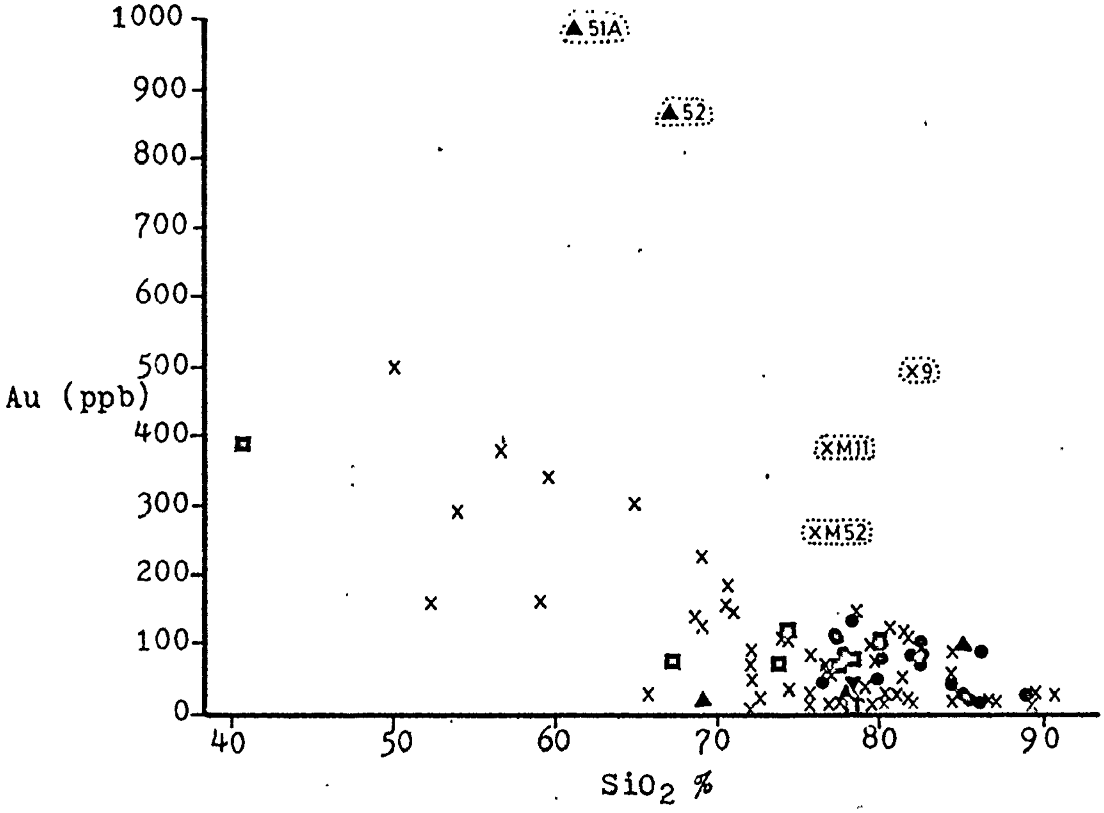

1000
900
800
700
600
500
400
300
200
100
0
Au (ppb)
40
50
60
70
80
90
SiO2 %
51A
52
X9
XM11
XM52

Fig. 12E

associated elements Pb, Nb and Y is an artifact of the weak correlation existing between U and FeS_2 (in this study .48), a fact which mine geologists use to roughly tell grade of ore. The correlation of As, Ni and Co with gold arises by preferential association of all these elements with the same mineral, pyrite. The negative correlation of FeS_2 versus SiO_2 results in the negative correlation between gold and SiO_2.

If free detrital gold were present in the conglomerates in large amounts, it might be expected to give a relationship with pebble size and undisputed heavy detrital minerals such as zircon (Zr) and monazite (Ce, La, P_2O_5), but no such relationship exists.

Figures 13 and 14 show lithological cross-sections through the C Reef (New Quirke Mine) and the Upper Reef (Denison Mine) respectively, togethor with variations in Au, FeS_2 and U. In general all three parameters appear to vary sympathetically through the vertical extent of the reefs. The conglomerates contain the highest gold and uranium values, the quartzites and argillite the lowest. In Figure 13 the calcite - sphalerite vein (30V) contains only traces of gold (21ppb) and no uranium.

Some additional points of interest include the following. Sample 49, obtained from a fault cutting the C Reef, contains galena and thucolite, but did not yield anomalously high gold values, only 19ppb. Neither did the C Reef Footwall Beds

Figure 13

Cross - section through the C Reef showing the variation in Au, FeS_2 and U

30A etc = sample numbers
∘∘∘ = conglomerate
⁚ = quartzite
▬ = argillite
⫽ = pyritic cross - bedding
═ = calcite - sphalerite vein

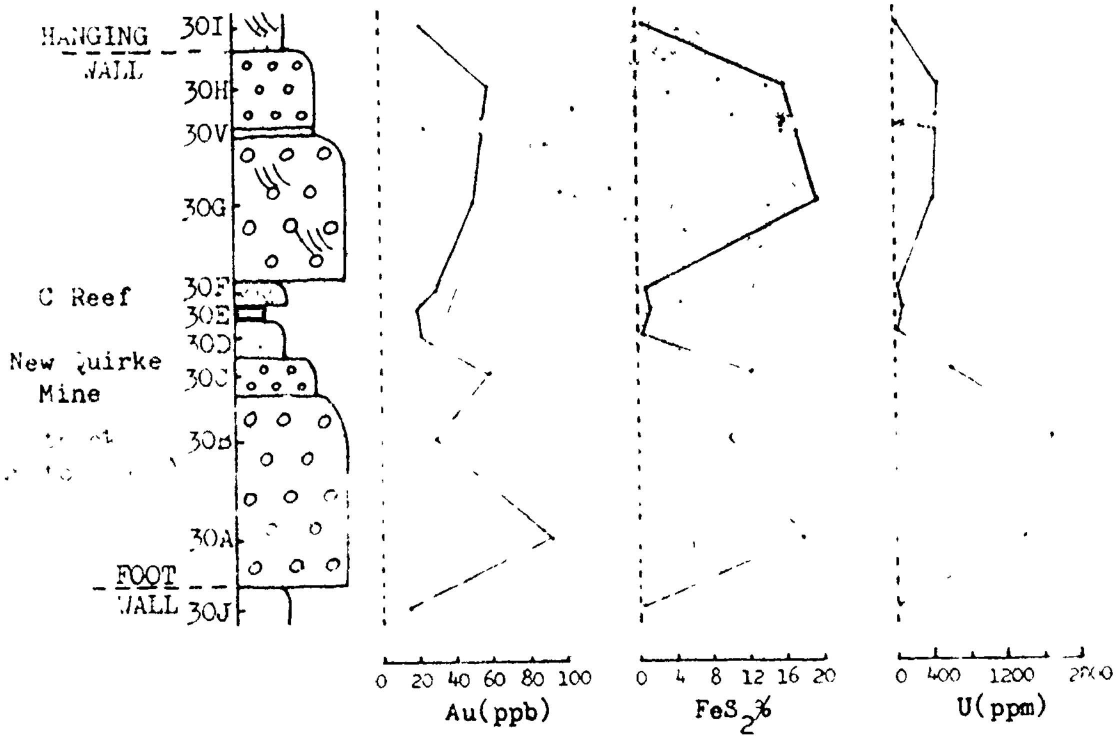

HANGING WALL
30I
30H
30V
30G
C Reef
30F
30E
30D
New Quirke Mine
30C
30B
30A
FOOT WALL
30J
0 20 40 60 80 100
Au(ppb)
0 4 8 12 16 20
FeS2%
0 400 1200 2000
U(ppm)

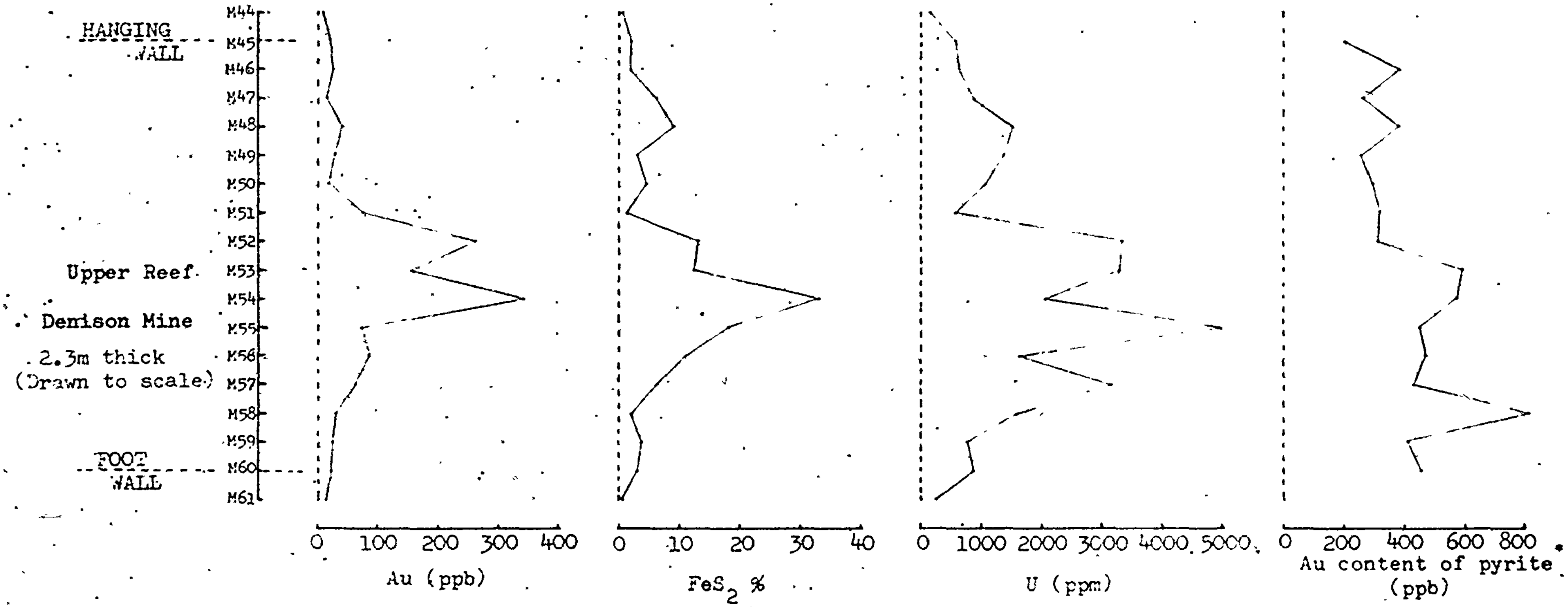
Figure 14

Cross - section through the Upper Reef, Denison Mine showing the variation in Au, FeS_2 and U.

M61 etc = sample numbers

All samples are conglomerates except M44 and M61, which are quartzites.

* = Data from Martindale (1968), see page 78 of this study


HANGING WALL
Upper Reef
Denison Mine
2.3m thick
(Drawn to scale)
FOOT WALL
M44
M45
M46
M47
M48
M49
M50
M51
M52
M53
M54
M55
M56
M57
M58
M59
M60
M61
0
100
200
300
400
Au (ppb)
0
10
20
30
40
FeS$_2$ %
0
1000
2000
3000
4000
5000
U (ppm)
0
200
400
600
800
Au content of pyrite (ppb)

(Scraps - see Table 2) - 69ppb Au. Sample 23, taken immediately adjacent to a dyke, contained only minor gold, that which is in proportion to its FeS_2 content. Gold appears to have been neither added or removed by the dyke intrusion.

5.3 Comparison of heavy mineral and whole rock gold values

The gold values used to calculate the correlation coefficients presented in Table 4, were obtained from the whole rock analyses. To test whether the whole rock sample size was large enough to adequately sample the average gold content, the heavy mineral fraction was separated from various samples. This would hopefully concentrate particulate gold from a much larger sample than taken for whole rock analysis and give a higher chance for detection of erratic particulate gold. The heavy mineral fraction represents that portion of the sample which sank in tetrabromoethane (see flow chart, Figure 7). The results are shown in Table 10 and in Figure 15.

After conversion of the heavy mineral fraction gold values back to whole rock values, using the heavy/light mineral weight ratios, a correlation coefficient of .88 was found for Au and FeS_2. The heavy/light mineral weight ratios were calculated by weighing the initial sample before separation and the heavy mineral fraction after separation. The difference between actual and recalculated gold values is due to error in in the heavy/light mineral weight ratios - sample mixing and sample loss in recovering heavy liquids.

For most samples, the heavy mineral fraction yields recalculated gold values which are within a factor of two of the whole rock values, although there are some exceptions. This suggests a relatively uniform distribution of gold and that 400mg whole rock sample adequately represents the larger

Sample	Heavy mineral Au value(ppb)	Heavy/Light mineral ratio	Recalculated Au(ppb)	Actual whole rock Au(ppb)
1	790	0.16	126	157
3C	346	0.11	38	116
5	728	0.19	138	148
9	308	0.09	28	489
10AP	358	0.15	54	100
12	337	0.20	67	115
12A	469	0.38	178	125
14	424	0.23	98	226
30A	383	0.23	88	94
56	510	0.14	71	87
15B	319	0.12	38	111
21C	278	0.17	47	84
23A	341	0.26	89	114

Table 10 : Recalculated whole rock gold values from heavy mineral fractions (see text)

sample from which the heavy fraction was taken. Sample 9 however, gives a far higher whole rock gold value than its heavy mineral concentrate indicating that by chance, sampling has isolated some particulate? gold. Concentrating the heavy mineral fraction may be worthwhile if time is available and a large enough initial sample is used.

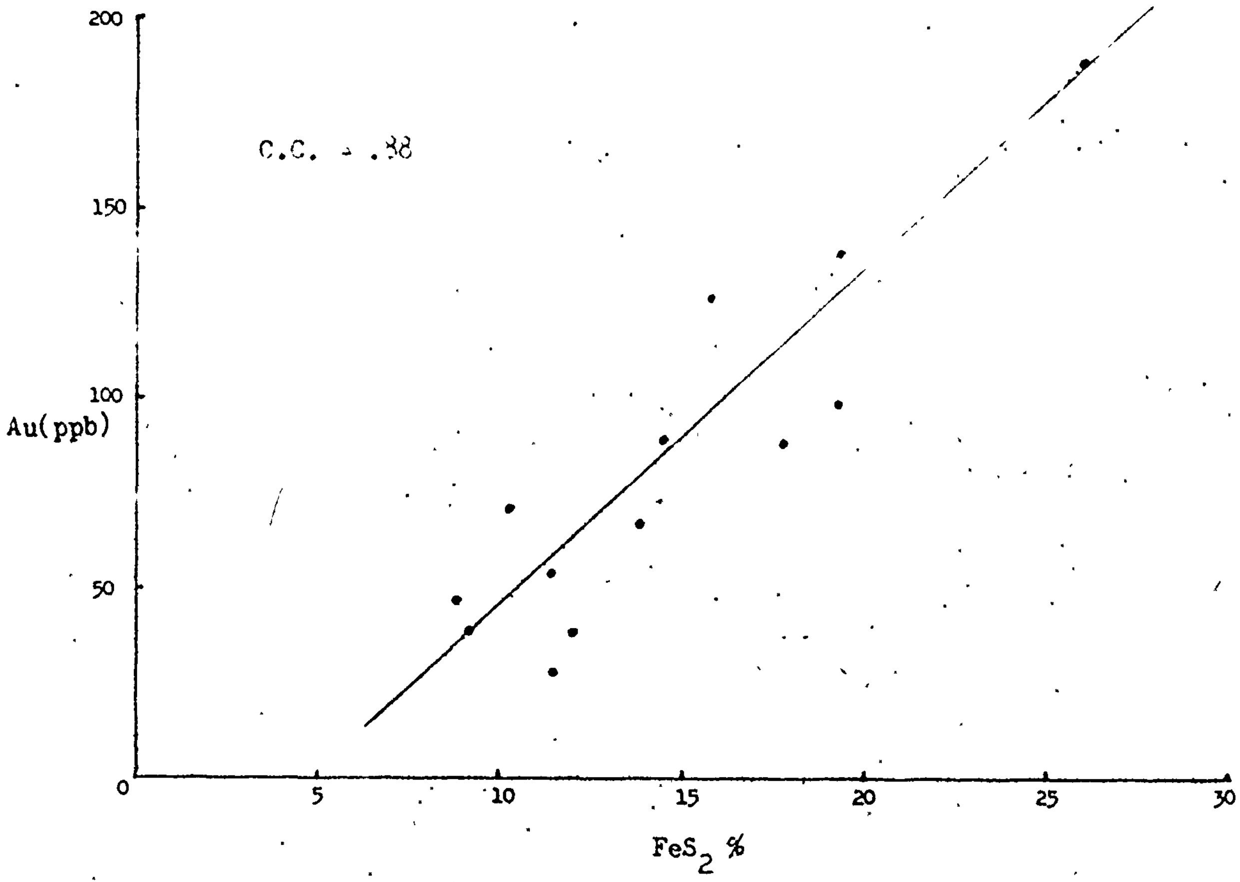


Figure 15 : FeS_2 versus recalculated gold values

5.4 Sedimentary control of gold

As stated previously five samples in Figure 12D plot well above the main distribution. These samples contain excess gold over and above the amount expected in pyrite alone. They possibly contain particulate gold of detrital origin.

The two samples with the highest gold values 51A (987ppb) and 52 (864ppb), occur at or near the base of a large channel cutting into the C Reef. This is a most likely place to find particulate gold. The present day channel bottoms of streams and rivers are favourable locations to find concentrations of placer gold. An attempt was made to detect particulate gold in sample 52. A portion of the 18 mesh powder was placed in hydrofluoric acid and allowed to digest for two days. However on inspection of the residue no gold particles were observed.

Sample 9 was a conglomerate lying directly on quartzite and as such could fit into No. iii in the classification scheme of Pretorius (1981) - see page 19 of this work. Samples M11 and M52 (Martindale 1968) were not collected by the author and their relationship to any sedimentary structure is incompletely known. Sample M11 is recorded as having been taken from near the footwall of the Lower Reef and sample M52 as having minor cross-bedding.

Though "high" gold values have been detected and can be related to specific sedimentary structures, many of the potentially favourable sites investigated yielded low gold values. The majority of conglomerates and quartzites recorded as having pyritic cross-beds did not yeild high gold values e.g. 30G

M48, 6AL, 56CB. Footwall and hanging wall contacts likewise yeilded low values e.g. ME2, M60, M45. Similarly sample 10AP, the base of a small scour, sample 54, a small channel, and sample 12A, a conglomerate resting on quartzite, contained only normal gold contents. Sample 30E, an argillite, yielded low gold values, as did the quartzites above and below it, 30D and 30F. All were sampled as complete units and included the bedding planes between them.

The Au/FeS_2 ratios for sample 3, the graded bed, are shown in Table 11.

Sample	Au/FeS_2	Pyrite grain size
3F	6.0	Small $\leqslant$ 1mm
3M	8.6	Mixture
3C	9.7	Large $>$ 1mm

Table 11 : Au/FeS_2 ratios for the graded bed

These values suggest that larger pyrite grains are richer in gold than smaller pyrite grains. To test this further the average Au/FeS_2 ratio of a number of samples with large pyrite was compared with the average for an equal number of samples with small pyrite from the same reef. The results are shown in Table 12. The averages are close enough togethor to indicate pyrite grain size is not a controlling factor in the gold content of the C Reef.

Samples with small pyrite	Au/FeS_2	Samples with large pyrite	Au/FeS_2
3F	6.0	1	10.6
10	11.7	2	5.9
10A	12.6	3C	9.7
10AP	8.7	4	5.5
14	11.7	5	7.7
30C	4.8	6	4.6
30G	2.6	7	10.6
30H	3.6	8	9.4
M45	10.0	11	4.9
M46	13.3	12	8.4
M47	2.2	12A	4.8
M48	4.4	13	4.9
M49	9.0	30A	5.3
M50	3.8	50	7.1
ME2	13.9	54	9.8
Average	7.9		7.3

Table 12 : Average Au/FeS_2 ratio of large and small pyrites from the C Reef (all samples are conglomerate, except ME2 which is conglomerate plus quartzite)

5.5 Distribution of gold in pyrite and pyrrhotite

Table 13 reports gold values for pyrite, pyrrhotite and chalcopyrite taken from the ore conglomerates. In most samples the amount of pyrrhotite is almost negligible and not enough was obtained for analysis.

Sample	Pyrite	Pyrrhotite	Chalcopyrite
7	402	799	-
51A	2085	186	-
52	6279	3636	-
53	452	61	-
56	527	154	-
R71-73*	1420	-	1398
Denison E.*	452	-	338

Table 13 : Gold values (ppb) for pyrite, pyrrhotite and chalcopyrite (* = from Martindale 1968)

Except for sample 7, gold is preferentially concentrated in pyrite compared to pyrrhotite. Martindale (1968) concluded the gold content of chalcopyrite is of the same order as pyrite from the same location. Samples 51A and 52 have been shown to contain anomalous gold, possibly particulate. The high gold values of pyrite and pyrrhotite (52 only) may indicate local assimilation of gold in or onto pyrite and pyrrhotite grains, or that in separation of the sulphides using the water column, gold has also been preferentially separated. If assimilation is chosen, the fact pyrite contains more gold than pyrrhotite is still true. If an artifact of separation, the difference in values could be due to gold being non-magnetic.

5.6 Gold and arsenic content of pyrite

Table 14 reports gold and arsenic values obtained from pyrite and provides further evidence that gold and arsenic in the ore are preferentially concentrated in the pyrite. The correlation coefficient between gold and arsenic in pyrite is poor (-.32), even if samples 51A and 52 are excluded from the calculation (.19). This may be due to the low number of data points.

Sample	As (ppm)	Au (ppb)
2	221	418
5	159	780
6	197	537
7	163	402
12	197	244
12A	185	305
14	281	340
15B	189	256
22A	189	218
23A	179	241
30A	175	318
51A	68	2085
52	108	6279
53	186	452
56	552	527
Average:	203	887

Table 14 : Gold and arsenic content of pyrite

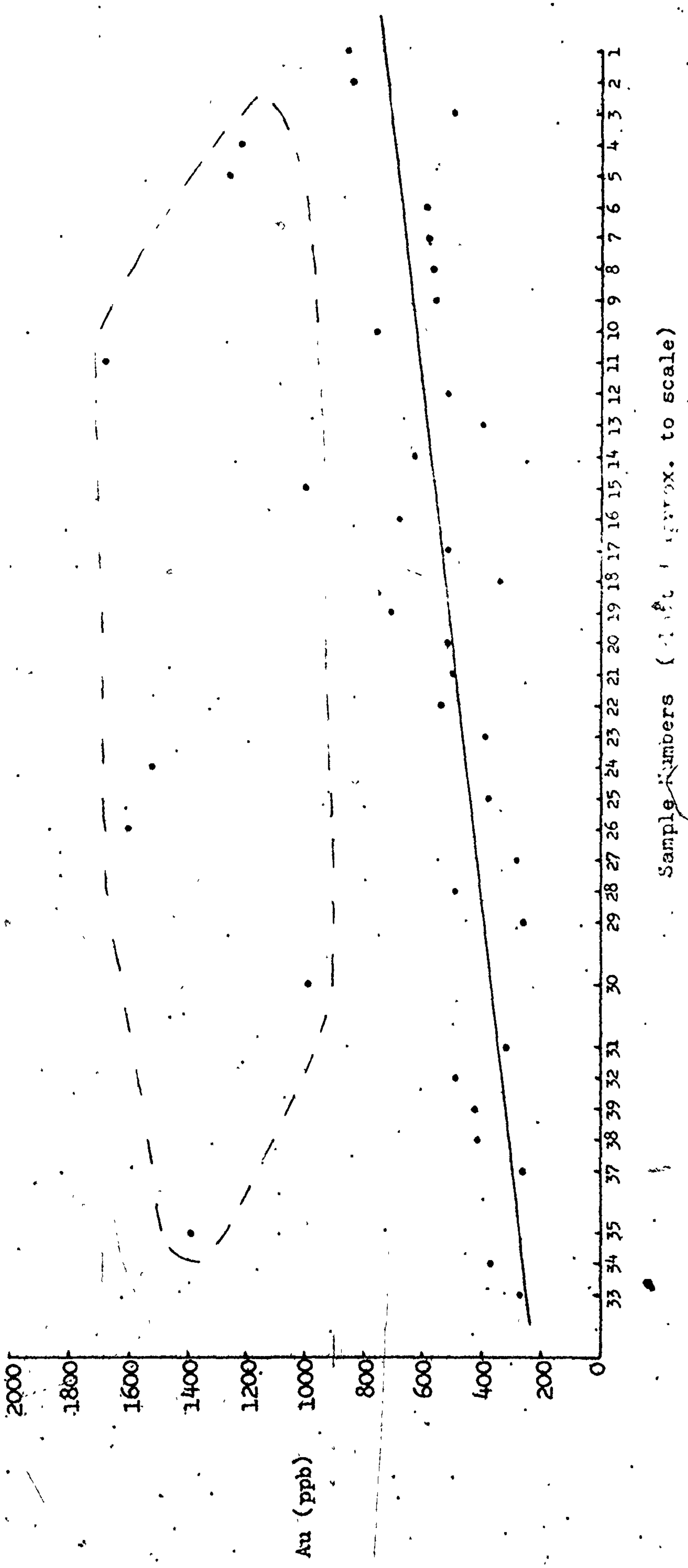
5.7 Geographic trends of pyritic gold in the Lower Reef, Denison Mine

Martindale (1968) studied the concentration and distribution of gold in pyrite from the reefs in the Denison Mine, and found a range of 202 to 2084ppb, with an average of 656ppb. From 38 samples collected along the strike of the Lower Reef over a distance of approximately 1.4km, he concluded no trend or systematic variation in gold concentration existed. Plotting his results graphically in Figure 16, reveals that 30 out of 38 points lie on or close to a straight line, with increasing gold values to the southeast. The remaining 8 points lie well above the line and are considered anomalous.

If this trend is real, it has important implications regarding the origin of the pyrite and the gold. From the economic standpoint it indicates higher gold values, at least in the pyrite, down strike to the southeast, towards the centre of the Quirke Lake Syncline.

Detrital sediment transport would not sort pyrite grains according to varying gold contents, where the gold variation is on the order of fractions of a ppm. The pyrite grains, if detrital, would be sorted by size and, as previously shown, the gold content of pyrite is not dependent on grain size. Therefore, whether or not there is a systematic change in pyrite grain size along strike, the increase in gold content down strike is not due to an increase in the percentage of coarser or finer grained pyrite. Gold may have been swept into

Figure 16 : Variation in gold content of pyrite along the strike of the Lower Reef, Denison Mine (Data from Martindale 1968. Least square line $Y = .29X + 289.47$)


Au (ppb)
0
200
400
600
800
1000
1200
1400
1600
1800
2000
1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16 17 18 19 20 21 22 23 24 25 26 27 28 29 30 31 32 39 38 37 35 34 33
Sample Numbers (not approx. to scale)

the conglomerates as free detrital particles during sedimentation, but suffered partial dissolution during or after lithification of the conglomerate with localized redistribution of soluble gold by adsorption onto pyrite. Though there is evid -ence for this happening on a local scale (eg. sample 52 - where coexisting pyrite and pyrrhotite are both anomalously high in gold), there is no clear evidence as to the scale (strike length) over which such a process might operate. If detrital gold had been incorporated into pyrite, perhaps by pyrite overgrowth during diagenesis, the gold would have to be of a very fine grain size to give the low values now found. As the original distribution of detrital gold would have been irregular, large scale redistribution and assimilation must have taken place to give the generally systematic trend now observed in the Denison data. It seems highly unlikely that the Denison trend could be produced solely by primary deposition of particulate gold and it argues against a simple placerist origin for the pyrite gold in the Elliot Lake conglomerates.

The observed trend in gold values is difficult to explain by the placer hypothesis. If it cannot be explained by sedimentary processes the implication is some other process or processes have been operating to form these deposits. The Denison trend may result from partial dissolution of very fine grained detrital gold with redistribution by groundwater or connate fluids subject to essentially unidirectional flow.

This is essentially a modified placer hypothesis. Fluid flow is concentrated in the Matinenda conglomerates because of their higher porosity and is unidirectional because of lithologically induced pressure gradients. Very fine grained gold is preferentially dissolved because of high surface area, (the extremely fine grained flour gold, characteristic of Tertiary fluvial placers, is a possible analogue for the Elliot Lake case), and is redistributed by fluids moving dominantly from the deeply buried interior to margins of the basin. As the fluids pass through the conglomerates, original detrital pyrite has to recrystallize to incorporate the gold. One has to envisage static recrystallization of pyrite while fluids are passing through the conglomerates.

Another possibility is that fluids, bearing trace amounts of gold, passed through the conglomerates from the southeast. On deposition of pyrite, either directly from diagenetic or epigenetic fluids or as a replacement of a pre-existing iron mineral, the fluids would become increasingly depleted in gold as they moved to the northwest, the gold being preferentially incorporated into pyrite. Solutions bearing trace amounts of gold, may have originated from lower Huronian volcanism or as brines from deep within the sedimentary basin. Bottrill (1971) has suggested that lower Huronian volcanism was the source of pyritising sulphur at Elliot Lake. Hydrogen sulphide associated with hydrocarbon may be another possible source for sulphur, though the sulphur isotope data of Roscoe (1969) tends to indicate a magmatic origin. The anomalous values in Figure 16 (points circled by the dashed line) may represent sites for preferential gold deposition or local assimilation of the rare, very fine gold grain into pyrite.

In reference to the above hypotheses, one advocates a source for gold within the Matinenda Formation, while the other advocates an extraneous source for the gold. In either case the implication is that the pyrite now observed in the Elliot Lake uranium conglomerates is not of direct detrital origin. At the very least, it indicates that the pyrite, if originally detrital, has been totally recrystallized to incorporate the trend in gold values now found. Investigation of other trace element trends in pyrite along with the areal distribution of sulphur isotopes, may throw more light on the processes that operated to form the pyrite.

A vertical profile of gold content in pyrite for the Upper Reef, Denison Mine, is illustrated in Figure 14. There is an apparent increase in gold values from hanging wall to footwall, though Martindale (1968) argues the trend may be due to experimental variation. Samples from the lower half of the reef were analysed separately from the upper half.

CHAPTER VI

Conclusions

The distribution of the majority of major and trace elements within the ore conglomerates can be related to a relatively small number of minerals : pyrite, sericite, uraninite, "brannerite", monazite and zircon. Relationships of many of the elements with pebble size probably indicates their distribution was controlled by processes operating at the time the conglomerates were deposited. Evidence exists however, which may indicate that the distribution of uranium now present in the ore conglomerates is controlled by the original distribution of carbon and titanium.

Gold is present almost entirely in the pyrite and gold concentration in the ore varies with pyrite content. Sporadic high gold values can be related to specific sedimentary structures e.g. channels, and may indicate the presence of original detrital gold. Gold is preferentially concentrated in pyrite compared to pyrrhotite. Geographic trends in pyritic gold indicate that the pyrite is not of direct detrital origin. Source for the gold may have been original flour gold present within the Matinenda Formation, or from fluids originating outside the Matinenda e.g. from lower Huronian volcanism.

REFERENCES

Abraham, E.M. (1953): Geology of Parts of Long and Spragge Townships, Blind River Uranium Area, District of Algoma; Ontario Dept. Mines, P.R.1953-2, Prelim. Rept., 10p., and attached map, scale 1 inch to ¼mile.

Arnold, R.G. (1954): A Preliminary Account of the Mineralogy and Genesis of the Uraniferous Conglomerate of Blind River, Ontario; Unpubl. M.A.Sc. Thesis, University of Toronto.

Bateman, J.D. (1955): Recent Uranium Developments in Ontario; Econ. Geol., 51, p.361-372.

Bottrill, T.J. (1971): Uraniferous Conglomerates of the Canadian Shield; p.77-83 in Report of Activities, Part A: April to October, 1970, Geol. Surv. Canada, Paper 7-1, Pt. A., 251p.

Boyle, R.W. (1979): The Geochemistry of Gold and its Deposits; Geol. Surv. Canada, Bull. 280.

Cloud, P.E. (1976): Beginnings of biospheric evolution and their biogeochemical consequences. Paleobiol., 2, no.4, p.351-387.

Cousins, C.A. (1973): Platinoids in the Witwatersrand System; J.S. Afr. Inst. Min. Metall., 73, no.6, p.184-199.

Crocket, J.H., Keays, R.R. and Hsieh, S. (1968): Determination of some precious metals by neutron activation analysis; J. Radioanal. Chem., 1, p.487-507.

Davidson, C.F. (1957): On the occurrence of uranium in ancient conglomerates; Econ. Geol., 52, p.668-693.

________, (1965): The Mode of Origin of Banket Orebodies; Trans. Inst. Min. and Metall., 74, Pt.6, p.319-338.

Derry, D.R. (1960): Evidence of the Origin of the Blind River Uranium Deposits; Econ. Geol., 55, no.5, p.635-647.

Ferris, C.S., and Ruud, C.O. (1971): Brannerite: Its Occurrences and Recognition by Microprobe; Quarterly of the Colorado School of Mines, 66, Oct. 1971, no.4.

Hallbauer, D.K. and van Warmelo, K.T. (1974): Fossilized plants in thucolite from Precambrian rocks of the Witwatersrand, South Africa; Precambrian Res., 1, p.199-212.

Hart, R.C., Harper, H.G. and Algom Field Staff. (1955): Uranium deposits of the Quirke Lake Trough, Algoma District, Ontario; Trans. Can. Inst. Min. Met., 48, no.517, p.260-265.

Heinrich, E.W. (1958): Mineralogy and Geology of Radioactive Raw Materials; McGraw-Hill Book Company, Inc. New York, 1958.

Hoefs, J. and Schidlowski, M. (1967): Carbon isotope composition of carbonaceous matter from the Precambrian of the Witwatersrand System; Science, 155, p.1096-1097.

Holmes, S.W. (1957): Pronto Mine in "Structural Geology of Canadian Ore Deposits"; CIM Spec. 2, p.324-339.

Joubin, F.R. (1954): Uranium Deposits of the Algoma District, Ontario; CIM Transactions, v. LVII, p.431-437.

______, (1960): Comments Regarding the Blind River (Algoma) Uranium Ores and Their Origin; Econ. Geol. 55, p. 1751-1756.

______, and James, D.H. (1957): Algoma Uranium District in "Structural Geology of Canadian Ore Deposits"; CIM Spec. 2, p.305-317.

Kimberly, M.M. (1978): Origin of stratiform uranium deposits in sandstone, conglomerate and pyroclastic rock; in Uranium Deposits: Their Mineralogy and Origin. Short Course, 3, Min. Assoc. Canada. Edit. by M.M. Kimberly.

Martindale, R.D. (1968): The Concentration and Distribution of

Gold in the Uraniferous Conglomerates of Elliot Lake. Unpubl. M.Sc. Thesis, McMaster University.

McDowell, J.P. (1957): The Sedimentary Petrology of the Mississagi Quartzite in the Blind River Area; Ontario Dept. Mines, GC6, 31p. Accompanied by 3 figures.

Muir, M.D. (1978): Occurrence and Potential uses of Archean Microfossils and Organic Matter; in Archean Cherty Metasediments, publ. by Univ. of Western Australia.

Patchett, J.E. (1960): A study of Radioactive Minerals of the Uraniferous Conglomerate, Blind River Area; Unpubl. Ph.D. Thesis, University of Toronto, Ontario.

Piennar, P.J. (1963): Stratigraphy, Petrology and Genesis of the Elliot Group, Blind River, Ontario, Including the Uraniferous Conglomerate; Geol. Surv. Canada, Bull. 83.

Pretorius, D.A. (1974): The Nature of the Witwatersrand Gold-Uranium Deposits; Econ. Geol. Res. Unit, Info. Circ. No. 86, Univ. of Witwatersrand.

__________, (1980): Gold Deposit Workshop; Dept. of Geology, University of Toronto. Dec.15 1980.

__________, (1981): Gold and Uranium in Quartz-Pebble Conglomerates; Econ. Geol. Res. Unit, Info. Circ. No.151, University of Witwatersrand.

Ramdohr, P. (1957): Die "Pronto-Reaction" : Neues Jahrbuch Min., p.217-221.

_______, (1958): New Observations on the Ores of the Witwatersrand in South Africa and Their Genetic Significance; Geol. Society South Africa, Annexure to vol.61.

_______, (1969): The ore minerals and their intergrowths; Pergamon Press, 1174p. English translation of 3rd ed.

Rice, R. (1958): Geology and Ore Deposits of the Elliot Lake District, Ontario; Unpubl. Ph.D. Thesis, Emmanuel College, Cambridge, England.

Robertson, D.S. (1962): Thorium and uranium variations in the Blind River ores; Econ. Geol., 57, p.1175-1184.

__________, and Steenland, N.C. (1960): On the Blind River uranium ores and their origin; Econ. Geol., 55, p.659-694.

Robertson, J.A. (1966): The Relationship of Mineralization to Stratigraphy in the Blind River Area, Ontario; Reprint from The Geol. Assoc. of Canada, Spec. Paper No.3, Precambrian Symposium, June 1966.

__________, (1969): Geology and Uranium Deposits of the Blind River Area, Ontario; CIM, Trans. 72, p.156-171.

__________, (1976): The Blind River uranium deposits: the ores and their setting; Ont. Div. Mines, MP 65, 45p.

__________, Frarey, M.J. and Card, K.D. (1969): The Federal-Provincial Committee on Huronian Stratigraphy Progress Report; Can. J. Earth Sci., 6, no.2, p. 335-336.

Roscoe, S.M. (1959): Monazite as an Ore Mineral in Elliot Lake Uranium Ores; Canadian Min. Jour., July 1959, p.65.

______, (1969): Huronian rocks and uraniferous conglomerates in the Canadian Shield; Geol. Surv. Can., Paper 68-40, 205p.

Ruzicka, V. and Steacy, H.R. (1976): Some sedimentary features of conglomeratic uranium ore from Elliot Lake, Ontario; in Report of Activities, Pt. A, Geol. Surv. Can., Paper 76-1A, p.343-346.

Theis, N.J. (1973): Comparitive Mineralogy of the Quirke No.1 and New Quirke Mines, Rio Algom Mines Ltd., Elliot Lake, Ontario; Unpubl. M.Sc. Thesis, Queens Univ. Kingston, Ontario.

_____, (1979): Uranium-bearing and associated minerals in their geochemical and sedimentological context, Elliot Lake, Ontario; Geol. Surv. Canada, Bull. 304.

Thorpe, R.I. (1963): The radioactive Mineralogy of the Ore Conglomerate at Panel Mine, Blind River, Ontario; Unpubl. M.Sc. Thesis, Queens Univ. Kingston, Ontario.

APPENDIX I

Element Analyses and Pebble Sizes.

30E* = Iron reported as Fe_2O_3

Pebble Size* = Average of ten largest pebbles
unless otherwise stated in brackets.

(%)

Sample	SiO_2	Al_2O_3	FeS_2	MgO	CaO	Na_2O	K_2O	TiO_2	MnO	P_2O_5	TOTAL
C Reef - New Quirke Line(Quirke Zone)											
1	70.75	9.91	14.79	0.65	0.05	0.10	3.55	0.25	0.03	0.02	100.05
2	79.[illegible]	4.78	12.60	0.27	0.05	0.09	1.73	0.11	0.03	0.02	99.57
3F	65.[illegible]	4.[illegible]	24.01	0.51	0.30	0.08	1.29	1.03	0.03	0.44	101.30
3K	70.6[illegible]	5.0[illegible]	21.83	0.44	0.25	0.09	1.38	0.52	0.03	0.16	100.47
[illegible]	81.44	4.38	11.99	0.02	0.00	0.05	1.14	0.07	0.02	0.02	99.18
4	73.[illegible]	3.31	20.44	0.06	0.19	0.03	0.80	0.07	0.02	0.02	99.33
5	71.06	7.23	19.3[illegible]	0.11	0.10	0.09	1.41	0.14	0.03	0.07	99.50
6	81.01	4.79	11.64	0.05	0.06	0.11	1.26	0.17	0.03	0.03	99.45
6AP	75.76	12.98	6.69	0.19	0.09	0.11	3.78	0.65	0.03	0.04	100.11
6AL	75.71	9.33	11.15	0.03	0.12	0.09	2.82	0.93	0.03	0.06	100.31
7	64.55	4.76	28.37	0.07	0.00	0.09	1.03	0.72	0.03	0.02	99.52
8	80.71	4.27	13.11	0.07	0.08	0.10	0.91	0.16	0.03	0.05	99.36
9	82.03	4.42	11.84	0.07	0.27	0.09	1.01	0.15	0.03	0.05	99.64
10	[illegible].25	6.93	1.37	0.0	[illegible]	0.07	1.40	0.74	0.03	0.03	99.77
10A	90.6[illegible]	4.94	[illegible]	0.00	0.07	0.09	1.22	1.09	0.03	0.03	99.36
10AP	[illegible].45	6.46	11.44	0.0	0.15	0.10	1.67	0.39	0.03	0.04	99.63
10AQ	[illegible]	9.95	0.30	0.04	0.00	0.06	2.26	0.07	0.03	0.01	99.21
11	74.21	2.55	21.60	0.02	0.64	0.07	0.64	0.11	0.03	0.04	99.91
12	81.77	2.91	13.71	0.07	0.15	0.09	0.56	0.06	0.03	0.02	99.35
12A	69.03	3.23	25.96	0.00	0.00	0.04	0.73	0.12	0.07	0.07	99.22
13	59.09	5.66	32.70	0.00	0.02	0.08	1.49	0.33	0.02	0.04	99.43
14	68.59	8.75	19.32	0.06	0.07	0.11	2.05	0.87	0.02	0.13	100.26
30A	74.[illegible]	[illegible].00	17.82	0.12	0.65	0.12	1.91	0.03	0.02	0.02	99.[illegible]
30B	80.27	5.92	11.22	0.27	0.07	0.09	1.57	0.14	0.03	0.03	99.63
30C	76.99	7.31	12.34	0.29	0.71	0.06	1.96	0.91	0.03	0.34	101.16
30D	74.55	19.79	0.47	0.39	0.02	0.12	6.10	0.44	0.03	0.02	100.02
30E	[illegible]2.58	31.70	1.55	0.80	0.63	0.13	9.75	2.75	0.03	0.04	100.00
30F	65.77	25.42	1.01	0.59	0.09	0.09	6.80	0.33	0.03	0.02	100.15
30G	72.17	5.88	19.45	0.51	0.14	0.09	1.51	0.46	0.03	0.07	100.30
30H	76.[illegible]	5.05	15.91	0.75	0.19	0.07	1.43	1.10	0.03	0.09	101.23
30I	81.0[illegible]	13.00	0.77	0.0[illegible]	0.05	0.10	4.26	0.23	0.03	0.02	99.57
30J	79.[illegible]	14.17	0.69	0.21	0.00	0.09	4.55	0.03	0.03	0.01	99.42
50	53.85	3.68	40.69	0.00	0.00	0.06	0.75	0.22	0.02	0.04	99.34
54	56.69	3.29	38.43	0.00	0.01	0.07	0.59	0.15	0.02	0.03	99.28
56	84.37	3.92	10.19	0.96	0.02	0.06	0.52	0.15	0.04	0.02	100.25
5603	39.63	6.55	51.14	0.45	0.06	0.06	1.67	0.42	0.02	0.07	100.08
Lower and Upper Reefs - Denison Line(Quirke Zone) - Equivalent to C Reef.											
M44	82.03	13.46	0.71	0.17	0.03	0.10	2.80	0.19	0.03	0.03	99.55
M45	87.20	[illegible].22	2.00	0.03	0.03	0.10	1.68	0.28	0.03	0.04	99.51
M46	81.11	13.11	1.06	0.13	0.03	0.12	2.82	0.35	0.03	0.04	99.70
M47	76.91	12.77	6.29	0.2	0.08	0.15	3.09	0.32	0.03	0.05	99.92
M48	73.05	8.03	[illegible].01	0.2[illegible]	0.08	0.14	1.97	0.42	0.03	0.16	100.11
M49	84.66	9.17	3.12	0.04	0.03	0.09	1.78	0.31	0.03	0.10	99.60
M50	77.67	13.9[illegible]	4.43	0.19	0.06	0.10	3.05	0.60	0.03	0.13	100.11
M51	76.79	16.85	1.27	0.21	0.03	0.14	4.10	0.44	0.03	0.03	99.88
M52	75.74	8.23	13.07	0.32	0.10	0.15	1.89	0.40	0.03	0.13	100.13
M53	73.54	6.84	17.27	0.06	0.06	0.09	1.31	0.30	0.03	0.05	99.63
M54	59.40	5.[illegible]	3[illegible].[illegible]4	[illegible]	0.06	0.09	1.16	0.31	0.02	0.11	99.64
M55	7[illegible]	7.[illegible]	15.[illegible]	0.10	0.0	0.09	1.48	0.74	0.03	0.12	99.76
M56	7[illegible]	10.[illegible]	10.7	[illegible]	0.0[illegible]	0.11	2.22	0.37	0.03	0.17	99.80
M57	84.[illegible]	7.10	6.[illegible]	[illegible]	0.07	0.11	1.33	0.11	0.03	0.03	99.37
M58	89.50	6.[illegible]	2.00	0.00	0.07	0.11	1.25	0.09	0.03	0.02	99.72
M59	84.91	[illegible]	3.72	[illegible]	0.64	0.10	1.62	0.26	0.03	0.05	99.52
M60	84.92	[illegible]	3.10	[illegible]	0.03	0.09	2.08	0.11	0.03	0.02	99.30

(%)

Sample	SiO$_2$	Al$_2$O$_3$	FeO$_2$	MgO	CaO	Na$_2$O	K$_2$O	TiO$_2$	MnO	P$_2$O$_5$	TOTAL
M61	80.15	14.87	0.79	0.09	0.00	0.11	3.19	0.15	0.03	0.02	99.40
M67	72.00	20.87	0.79	0.11	0.02	0.12	5.34	0.35	0.03	0.02	99.65
M68	81.84	6.32	9.57	0.09	0.03	0.10	1.27	0.12	0.03	0.02	99.39
M69	84.65	10.45	1.89	0.00	0.01	0.10	2.01	0.10	0.03	0.02	99.26
M70	74.57	8.55	14.14	0.03	0.04	0.09	1.75	0.19	0.03	0.03	99.41
M11	76.71	7.57	13.18	0.12	0.06	0.13	1.53	0.14	0.03	0.04	99.52
M26	49.88	5.21	42.54	0.06	0.05	0.10	1.27	0.20	0.03	0.06	99.50
ME1	46.81	28.51	13.33	1.73	0.36	0.14	10.35	0.99	0.03	0.13	102.78
ME2	78.84	15.28	1.66	0.08	0.03	0.09	3.22	0.43	0.03	0.04	99.70
ME3	52.31	4.03	41.64	0.00	0.07	0.09	1.01	0.34	0.02	0.11	99.63
A Reef - New Quirke Mine(Quirke Zone)											
15A	84.95	8.74	2.73	0.00	0.03	0.08	2.58	0.42	0.02	0.04	99.59
15B	82.57	5.49	9.11	0.24	0.11	0.09	1.83	0.48	0.02	0.09	100.83
15C	85.33	9.65	1.18	0.06	0.01	0.09	2.85	0.24	0.03	0.02	99.45
16	82.75	8.25	5.46	0.12	0.83	0.11	2.54	0.96	0.03	0.48	101.53
17	79.97	6.44	10.81	0.02	0.02	0.11	1.79	0.37	0.02	0.04	99.58
18	82.51	6.44	8.06	0.07	0.04	0.11	1.99	0.36	0.03	0.05	99.66
19	78.39	4.91	14.21	0.04	0.00	0.11	1.48	0.47	0.03	0.09	99.74
19A	77.32	8.99	9.09	0.16	0.14	0.11	3.60	0.38	0.02	0.05	99.86
19B	76.37	13.82	4.10	0.29	0.00	0.11	4.71	0.29	0.03	0.02	99.74
21A	82.44	[illegible]	[illegible]	0.16	0.06	0.09	1.54	0.43	0.03	0.07	99.54
21B	86.26	8.39	1.33	0.17	0.09	0.10	2.49	0.31	0.03	0.04	99.74
21C	82.25	6.33	8.77	0.09	0.08	0.09	1.65	0.43	0.03	0.06	99.78
22A	86.29	3.34	8.98	0.03	0.32	0.07	0.39	0.32	0.03	0.07	99.84
23	89.05	4.33	4.60	0.40	6.27	0.07	1.02	0.43	0.07	0.11	106.35
23A	77.36	5.62	14.53	0.36	0.29	0.12	1.49	0.43	0.03	0.17	100.40
24A	79.91	7.49	9.59	0.03	0.05	0.11	2.20	0.33	0.03	0.04	99.59
25	84.45	8.37	3.72	0.16	0.03	0.09	2.46	0.69	0.03	0.03	100.03
31	77.46	8.68	10.40	0.19	0.28	0.10	2.37	0.34	0.03	0.17	100.61
32	79.60	8.78	8.13	0.28	0.04	0.09	2.48	0.57	0.03	0.08	100.09
Upper Reef - New Quirke Mine(Quirke Zone)											
20	75.28	10.97	8.82	0.26	0.29	0.10	3.93	0.71	0.03	0.22	100.61
Nordic Zone											
N1	77.41	9.02	9.09	0.15	0.03	0.17	3.48	0.20	0.03	0.02	99.60
N2	40.52	2.78	54.53	0.00	0.03	0.06	1.17	0.42	0.02	0.15	99.68
N3	67.13	13.51	13.39	0.33	0.03	0.14	4.97	0.53	0.02	0.06	100.11
N4	78.37	10.02	7.41	0.12	0.02	0.08	3.20	0.31	0.03	0.03	99.59
N5	74.31	10.15	9.67	0.20	0.03	0.10	4.88	0.24	0.02	0.01	99.61
N6	73.83	10.31	10.43	0.19	0.04	0.17	4.43	0.15	0.02	0.02	99.62
N7	79.97	7.97	8.19	0.12	0.03	0.06	2.87	0.19	0.02	0.03	99.47
N8	78.22	8.98	7.70	0.14	0.03	0.17	4.09	0.17	0.03	0.03	99.57
Reworked C Reef(Ramsay Lake Conglomerate) - New Quirke Mine(Quirke Zone)											
41	[illegible]	5.16	14.49	0.50	0.20	0.13	1.32	0.31	0.03	0.04	100.21
45	69.21	20.98	3.74	0.89	0.08	0.13	5.57	0.31	0.03	0.06	101.00
51	85.27	5.98	6.49	0.17	0.02	0.10	1.26	0.23	0.03	0.02	99.57
51A	[illegible]	4.56	[illegible]	0.06	0.10	0.12	0.96	0.17	0.02	0.05	99.53
52	[illegible]	[illegible]	24.22	[illegible]	0.06	0.10	1.03	0.15	0.03	0.04	99.47

(ppm)

Sample	U	Th	Zr	Ce	La	Y	Ba	Pb	Cr	As	V
C Reef - New Quirke Mine(Quirke Zone)											
1	2623	358	174	351	241	145	1019	1353	218	41	21
2	726	139	54	175	102	65	512	592	262	20	10
3F	767	1513	649	4595	2778	290	269	593	192	49	55
3M	2676	1034	365	2130	1292	394	199	1122	190	47	27
3C	1769	248	58	137	86	113	173	1055	105	40	5
4	1578	246	51	45	20	82	85	1352	212	32	6
5	3773	539	108	295	185	189	147	2313	243	56	25
6	929	259	75	422	265	85	186	754	359	31	10
6AP	29	269	296	526	338	35	488	32	223	14	35
6AL	117	392	429	805	516	49	400	71	234	22	43
7	4735	859	145	1389	738	319	46	3353	276	60	14
8	3185	690	93	990	587	209	65	1859	194	52	8
9	1833	458	126	710	396	96	180	1402	226	32	7
10	829	472	147	604	256	65	169	594	199	12	10
10A	150	119	47	290	176	17	142	195	106	12	2
10AP	957	740	122	1362	1003	98	156	1013	74	49	11
10AQ	20	59	51	44	28	9	254	13	39	11	1
11	3218	594	82	596	344	209	122	2160	320	40	8
12	2053	271	52	331	162	117	104	1150	329	43	5
12A	1640	299	69	444	252	116	103	1172	99	65	7
13	596	22[illegible]	80	459	304	38	131	625	236	95	24
14	983	754	199	1814	1130	152	179	658	237	48	52
30A	1446	204	55	126	98	107	276	806	240	37	8
30B	1702	369	96	447	289	145	199	1292	266	37	8
30C	657	1236	585	2530	1722	211	307	269	268	35	46
30D	23	77	120	243	152	17	761	10	260	2	40
30E	80	26	403	412	281	62	1317	57	242	-	303
30F	17	34	174	316	239	11	793	11	200	1	41
30G	425	419	128	1043	633	114	191	781	240	30	24
30H	489	547	300	1308	826	145	179	1732	238	38	49
30I	13	48	90	138	119	10	495	-	330	8	15
30J	3	16	43	33	28	3	673	-	99	2	5
50	3715	562	111	312	184	268	85	3060	263	47	18
54	1804	307	88	219	126	153	120	1775	313	34	19
56	1886	315	70	254	127	117	217	1412	69	45	6
56CB	783	307	95	738	530	93	499	1975	272	31	31
Lower and Upper Reefs - Denison Mine(Quirke Zone) - Equivalent to C Reef.											
M44	147	230	126	340	239	46	177	8	494	-	9
M45	578	447	127	731	484	62	124	145	258	11	12
M46	623	460	180	412	246	86	174	157	703	-	21
M47	876	464	119	635	454	77	167	643	679	9	20
M48	1521	1105	165	2317	1487	136	120	1003	117	28	20
M49	1387	926	182	1640	1071	167	121	681	404	19	17
M50	1049	1095	392	2076	1382	124	196	724	194	2	39
M51	566	328	243	414	333	61	267	1166	395	-	25
M52	3357	1103	184	1849	1146	165	148	2282	102	32	24
M53	3293	752	117	904	653	134	30	1880	561	26	21
M54	2044	625	83	1705	879	76	65	1291	572	46	20
M55	4954	1077	117	1468	[illegible]	271	79	3205	639	45	15
M56	1624	1024	210	1[illegible]	[illegible]	111	146	1176	[illegible]	35	[illegible]
M57	3151	547	66	767	220	142	90	1587	497	26	[illegible]
M58	2577	270	62	[illegible]	[illegible]	65	76	569	394	15	5
M59	779	541	214	[illegible]	570	[illegible]	131	314	566	3	14
M60	[illegible]	232	[illegible]	252	171	51	116	215	223	9	[illegible]

(ppm)

Sample	U	Th	Zr	Ce	La	Y	Ba	Pb	Cr	As	V
M61	267	118	38	161	97	39	202	18	64	4	6
M67	28	43	145	109	41	31	247	-	445	-	27
M68	1245	210	57	137	59	47	48	283	76	39	7
M69	564	73	59	61	40	38	78	141	83	15	5
M70	1647	321	60	159	113	55	72	1203	607	39	11
M11	2931	437	74	172	132	88	115	1956	636	40	12
M16	2961	975	55	479	352	105	49	2434	715	66	21
ME1	1116	637	235	1363	854	86	1932	210	243	13	95
ME2	200	279	220	712	463	42	175	71	439	4	27
ME3	1112	422	64	1100	711	59	73	870	767	40	22
A Reef - New Quirke Mine(Quirke Zone)											
15A	664	511	165	725	462	73	563	319	206	13	23
15B	2237	1075	252	1714	1045	211	453	1310	300	48	27
15C	398	275	112	237	139	44	580	92	268	11	15
16	2758	2233	518	2413	1263	283	528	1937	292	33	57
17	365	267	109	694	447	58	270	277	277	63	24
18	1355	671	125	1447	820	110	335	691	239	21	15
19	3466	369	146	1511	850	245	266	1956	227	44	23
19A	1073	432	125	725	441	101	508	637	260	22	21
19B	921	420	100	567	335	76	708	496	224	23	16
21A	1681	703	133	1141	714	147	199	1743	378	25	23
21B	940	443	144	567	360	73	352	521	227	5	20
21C	1909	791	142	1090	661	163	261	2350	317	19	23
22A	2387	779	111	1501	852	145	209	1367	208	30	15
23	1141	779	172	1245	727	110	214	658	273	20	22
23A	2714	1187	150	2760	1576	222	204	2696	225	43	25
24A	1432	435	99	590	432	126	336	1005	384	31	21
25	143	154	112	452	353	20	372	66	99	22	34
31	435	1252	329	2523	1580	132	318	353	281	20	46
32	2413	1061	235	1392	839	248	344	1882	305	29	34
Upper Reef - New Quirke Mine(Quirke Zone)											
20	2965	1387	417	2353	964	272	536	1703	90	17	37
Nordic Zone											
N1	2416	407	76	274	158	77	310	1364	72	37	16
N2	4960	899	83	1529	971	117	69	2740	261	139	30
N3	506	454	125	971	647	50	422	479	220	37	50
N4	642	97	78	326	215	36	312	670	591	31	24
N5	1773	362	79	267	170	53	579	1115	114	47	18
N6	1460	239	68	173	98	62	475	919	72	42	17
N7	3686	618	83	369	223	110	278	2363	327	42	15
N8	979	271	71	371	265	50	452	672	181	24	15
Reworked C Reef(Ramsay Lake Conglomerate) - New Quirke Mine(Quirke Zone)											
41	1159	274	100	211	122	116	169	562	327	7	30
45	255	199	156	397	235	74	768	81	249	-	77
51	960	195	84	321	184	45	159	1071	234	15	13
51A	6456	943	170	614	321	505	145	4539	250	71	25
52	7637	1042	16[illegible]	[illegible]	214	468	121	5440	263	64	20

Sample	Co	Cu	Zn	Ni (ppm)	Nd	Nb	Rb	Sr	Pebble Size(cm)	Au(ppb)	U/Th
1	[illegible]	83	14	43	110	68	142	21	4.2	157	7.3
2	143	141	19	ND	50	31	67	8	3.3	74	5.2
3F	[illegible]	96	53	ND	1157	80	43	36	1.5	144	0.5
3A	278	97	47	ND	587	87	73	19	2.4	188	2.6
3C	160	128	46	ND	52	52	65	2	4.3	116	7.1
4	114	157	18	ND	22	59	52	-	3.3	113	6.4
5	339	17	21	53	102	100	72	11	4.5	148	7.0
6	210	31	18	ND	123	48	50	2	4.5	55	3.6
6AP	48	6	20	ND	129	58	121	13	-	10	0.1
6AL	83	13	18	ND	193	80	94	14	-	33	0.3
7	336	28	20	73	385	173	74	16	3.4	300	5.5
8	510	12	19	51	262	85	52	11	[illegible]	123	4.6
9	428	9	19	45	197	61	41	13	4.2	489	4.0
10	108	11	15	34	297	40	47	15	3.5	16	1.8
10A	201	10	19	24	88	31	39	15	3.3	27	1.3
10AP	661	10	94	61	474	75	52	14	-	100	1.3
10AQ	64	10	17	28	30	26	77	15	-	23	0.3
11	116	22	18	ND	175	135	51	5	[illegible]	106	5.4
12	457	16	19	52	153	67	34	10	4.9	115	7.6
12A	348	14	18	56	127	66	41	12	3.4	125	5.5
13	175	13	20	ND	110	74	59	-	3.4	159	2.7
14	149	23	19	ND	452	135	73	3	3.6	226	1.3
30A	225	165	19	ND	37	52	74	4	4.3	94	7.1
30B	176	208	79	ND	127	43	68	5	4.3	30	4.6
30C	125	66	23	ND	753	77	69	27	[illegible]	59	0.5
30D	17	3	21	ND	71	17	194	11	-	24	0.3
30E	16	35	23	ND	110	33	367	25	-	19	3.1
30F	16	9	21	ND	83	12	230	10	-	29	0.5
30G	102	38	263	ND	268	58	57	11	3.7	51	1.0
30H	169	56	592	ND	334	87	53	10	2.6	58	0.9
30I	22	19	17	ND	41	27	133	16	-	22	0.4
30J	31	7	19	ND	18	11	161	16	-	16	0.2
50	139	4	30	ND	30	184	68	-	4.4	287	6.6
54	179	62	18	ND	63	86	50	-	2.9	376	5.9
56	130	259	67	ND	93	71	43	-	5.2	87	6.0
560B	177	34	19	ND	153	122	75	-	-	510	2.6
Lower and Upper Reefs - Denison Mine(Quirke Zone) - Equivalent to C Reef.											
M44	10	31	17	24	74	22	103	9	-	9	0.6
M45	55	27	37	41	167	40	80	8		20	1.3
M46	16	4	15	39	147	43	105	8	1.9 (8)	26	1.4
M47	131	6	15	61	163	52	106	8	1.8 (4)	14	1.9
M48	171	214	13	83	554	77	78	8	2.5 (8)	40	1.4
M49	36	20	12	39	395	61	81	8	1.6 (5)	28	1.5
M50	36	8	15	45	482	61	102	8	1.6 (4)	17	1.0
M51	21	4	14	29	92	42	139	9	2.7 (7)	73	1.8
M52	197	32	12	75	474	98	107	7	2.9 (7)	262	3.0
M53	67	5	15	48	246	83	93	7	3.0 (4)	155	4.4
M54	125	-	13	40	716	91	82	6	2.6 (5)	341	7.3
M55	[illegible]	[illegible]	13	40	370	116	123	6	2.5 (5)	72	4.7
M[illegible]	[illegible]	[illegible]	15	51	513	57	[illegible]	7	2.3 (5)	86	[illegible]
M57	[illegible]	24	14	[illegible]	[illegible]	60	[illegible]	6	2.9	62	[illegible]
M[illegible]	[illegible]	10	14	[illegible]	50	33	71	[illegible]	[illegible]	30	[illegible]
M5[illegible]	[illegible]	[illegible]	14	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	2.7 (9)	24	[illegible]
M[illegible]	[illegible]	[illegible]	17	30	63	[illegible]	94	8	2.2 (7)	[illegible]	[illegible]

					(ppm)				Pebble		
Sample	Co	Cu	Zn	Ni	Nd	Nb	Rb	Sr	Size(cm)	Au(ppb)	U/Th
M61	15	6	13	26	52	23	112	8	-	16	2.3
M67	12	-	14	22	29	26	163	9	-	5	0.7
M68	117	45	235	43	37	41	73	8	5.2 (4)	22	5.9
M69	33	13	17	36	21	26	77	8	2.9 (7)	27	7.7
M70	79	22	19	55	39	65	84	7	3.4 (6)	38	5.1
M11	136	23	15	81	56	78	105	6	3.3 (5)	382	6.0
M26	307	21	17	56	119	91	99	4	3.0 (3)	500	5.0
ME1	35	-	24	15	333	104	410	17	-	50	1.8
ME2	20	15	14	29	172	39	94	9	2.7 (9)	23	0.7
ME3	220	91	18	74	285	79	56	5	3.1 (3)	160	2.6
A Reef - New Quirke Mine(Quirke Zone)											
15A	70	71	24	ND	178	51	107	10	1.9	30	1.3
15B	152	97	26	ND	458	98	77	5	3.1	112	2.1
15C	18	110	32	ND	86	31	110	10	2.8	22	1.4
16	75	21	20	ND	932	129	84	67	3.1	89	1.2
17	97	63	19	ND	167	61	66	2	3.9	102	1.4
18	241	12	19	45	368	69	76	22	3.7	71	2.0
19	374	242	17	55	437	129	74	22	3.6	130	4.0
19A	63	19	17	ND	199	72	118	15	2.9	108	2.2
19B	178	9	16	26	161	52	154	30	2.1	44	2.9
21A	102	63	26	ND	306	99	68	4	2.9	73	2.4
21B	38	68	26	ND	154	51	112	9	2.1	16	2.1
21C	68	76	42	ND	288	100	70	5	2.7	84	2.4
22A	122	20	20	31	413	90	50	23	3.8	93	3.1
23	50	190	18	ND	337	93	63	70	2.6	29	1.5
23A	260	213	30	57	718	108	66	20	2.7	124	2.3
24A	229	130	22	ND	180	81	94	11	4.1	78	3.0
25	142	12	15	27	98	56	98	21	2.9	44	1.0
31	82	47	20	ND	608	81	78	14	1.8	94	0.3
32	66	54	44	ND	359	104	94	8	3.4	49	2.3
Upper Reef - New Quirke Mine(Quirke Zone)											
20	128	27	17	46	883	85	118	28	1.5	46	1.6
Nordic Zone											
N1	199	21	14	58	76	59	134	12	4.0 (6)	70	5.9
N2	286	37	13	109	377	136	141	5	2.8 (4)	388	5.5
N3	104	-	16	57	220	48	136	12	2.3 (7)	77	1.1
N4	37	29	14	38	75	35	105	11	3.5 (7)	80	6.6
N5	41	164	17	52	76	47	140	16	-	123	4.9
N6	94	1	14	53	54	44	147	15	-	74	6.1
N7	52	14	15	58	103	69	141	10	3.6	107	6.0
N8	172	49	15	48	95	35	126	14	3.0	16	3.0
Reworked C Reef(Ramsay Lake Conglomerate) - New Quirke Mine(Quirke Zone)											
41	115	228	20	ND	71	55	64	5	6.6	29	5.0
45	43	129	19	44	123	42	204	24	1.8	19	1.4
51	133	30	19	27	86	45	56	15	4.6	101	4.9
51A	427	114	21	ND	253	220	90	-	7.4	997	6.8
52	244	117	104	91	208	167	90	13	3.5	364	7.4

APPENDIX II

Pyrite Grain Sizes.

L = >1mm in diameter

S = $\leqslant$1mm in diameter

Sample	Pyrite Grain Size	Sample	Pyrite Grain Size	Sample	Pyrite Grain Size
1	L	50	L	15B	S
2	L	54	L	15C	S
3F	S	56	L	16	L
3M	L + S	56CB	L	17	S
3C	L	M44	S	18	S
4	L	M45	S	19	S
5	L	M46	S	19A	S
6	L	M47	S	19B	S
6AP	S	M48	S	21A	S
6AL	S	M49	S	21B	S
7	L	M50	S	21C	S
8	L	M51	S	22A	S
9	L	M52	L + S	23	-
10	S	M53	L	23A	S
10A	S	M54	L	24A	S
10AP	S	M55	L	25	S
10AQ	-	M56	L	31	S
11	L	M57	L	32	S
12	L	M58	L	20	S
12A	L	M59	L + S	N1	L + S
13	L	M60	L + S	N2	L + S
14	S	M61	S	N3	S
30A	L	M67	S	N4	S
30B	L + S	M68	L	N5	S
30C	S	M69	L	N6	S
30D	-	M70	L	N7	L + S
30E	-	M11	L	N8	L + S
30F	-	M26	L	41	-
30G	S	ME1	L	45	-
30H	S	ME2	S	51	L
30I	S	ME3	L + S	51A	L
30J	-	15A	S	52	L

APPENDIX III

Analyses of Duplicate Samples.

Sample	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O	TiO_2	MnO	P_2O_5	U	Th	Zr	Ce
After Coning and Splitting														
[illegible]	85.33	9.65	1.18	0.06	0.01	0.09	2.85	0.24	0.03	0.02	398	275	112	237
	86.46	8.40	1.29	0.01	0.00	0.10	2.85	0.24	0.02	0.02	397	288	111	364
24A	79.91	7.49	9.59	0.03	0.05	0.11	2.20	0.33	0.03	0.04	1432	485	99	690
	81.52	6.30	9.03	0.10	0.02	0.09	2.15	0.33	0.03	0.04	1414	499	90	695
31	77.46	8.68	10.49	0.19	0.22	0.10	2.37	0.84	0.03	0.17	435	1252	329	2523
	77.72	8.42	10.55	0.16	0.25	0.09	2.32	0.87	0.03	0.19	433	1303	330	2714
	78.90	8.00	9.72	0.21	0.26	0.07	2.33	0.82	0.03	0.16	434	1225	313	2438
54	56.69	3.29	38.43	0.00	0.01	0.07	0.59	0.15	0.02	0.03	1804	307	88	219
	55.79	3.30	39.29	0.00	0.01	0.06	0.62	0.13	0.02	0.03	1793	308	80	221
After Crushing in Shatterbox														
5	71.06	7.23	19.30	0.11	0.10	0.09	1.41	0.14	0.03	0.03	3773	539	108	295
	69.79	6.56	21.21	0.05	0.43	0.11	1.44	0.13	0.03	0.03	3903	543	102	258
N2	78.22	8.98	7.70	0.14	0.03	0.17	4.09	0.17	0.03	0.03	979	271	71	371
	78.11	8.80	7.93	0.15	0.04	0.18	4.16	0.17	0.03	0.03	969	270	71	392
24A	79.91	7.49	9.59	0.03	0.05	0.11	2.20	0.33	0.03	0.04	1432	485	99	690
	79.93	7.68	9.01	0.19	0.08	0.10	2.38	0.34	0.03	0.04	1446	514	88	696
	79.04	7.72	9.85	0.18	0.07	0.14	2.39	0.33	0.03	0.04	1457	510	97	707
154	59.47	5.32	33.04	0.05	0.06	0.09	1.16	0.31	0.02	0.11	2044	625	83	1308
	59.66	5.36	32.84	0.02	0.06	0.03	1.14	0.32	0.02	0.12	2051	622	93	1304
	59.32	5.32	33.23	0.04	0.06	0.07	1.13	0.32	0.02	0.12	2069	615	110	1338

1 OF 1

P_2O_5	U	Th	Zr	Ce	La	Y	Ba	Pb	Sr	As	V	Co	Cu	Zn	Ni	Nd	Nb	Rb	Cr
0.02	398	275	112	237	139	44	580	92	268	11	15	18	110	32	ND	66	31	110	10
0.02	397	238	111	364	[illegible]	45	565	91	247	11	32	30	141	32	ND	[illegible]	32	108	11
0.04	1432	435	99	690	432	126	336	1005	384	31	21	229	130	22	ND	180	81	94	11
.04	1414	499	90	695	431	125	340	1037	442	26	22	190	139	23	ND	180	83	36	10
0.17	435	1252	329	2523	1520	132	318	353	231	20	46	32	47	20	ND	359	104	94	3
0.19	433	1303	330	2714	1629	129	318	371	325	7	46	33	29	19	ND	661	53	72	13
0.16	434	1225	313	2438	1510	127	327	362	313	10	48	72	50	20	ND	594	81	70	12
0.03	1804	307	35	219	136	153	120	1775	313	34	19	179	62	18	ND	63	86	50	-
0.03	1793	308	30	221	137	156	82	1802	396	38	20	187	60	20	ND	69	85	53	-
0.03	3773	539	108	295	155	169	147	2313	243	56	25	339	17	21	53	102	100	72	11
0.03	3903	543	102	258	164	267	145	2408	331	68	13	321	16	21	ND	106	112	95	-
0.03	979	271	71	371	265	50	452	672	181	24	15	172	49	15	48	95	35	126	14
0.03	969	270	71	392	261	51	452	658	200	19	16	181	48	15	49	96	37	131	15
0.04	1432	435	99	690	432	126	336	1005	384	31	21	229	130	22	ND	180	81	94	11
0.04	1446	514	88	696	439	129	327	1035	545	22	20	115	131	23	ND	179	81	91	8
0.04	1457	510	97	707	432	128	327	1057	448	26	16	240	135	22	ND	184	82	93	10
0.11	2044	625	83	1308	379	76	65	1291	572	46	20	126	-	13	40	316	91	82	6
0.12	2051	622	93	1304	875	74	66	1319	445	41	20	119	-	13	47	311	95	31	7
0.12	2069	615	110	1338	904	74	66	1310	612	36	21	119	-	13	46	328	95	82	7

After coning and splitting

Sample		Au value(ppb)	Sample		Au value(ppb)
15C	-	22	31	-	94
		17			34
					48
23A	-	114			
		60	54	-	376
					569
24A	-	78			
		86			

(Average % difference= 33%)

After crushing in shatterbox

Sample		Au value(ppb)	Sample		Au value(ppb)
3C	-	116	30G	-	51
		84			43
5	-	148	ME1	-	50
		168			51
7	-	300	M60	-	22
		196			17
10A	-	27	M67	-	5
		20			10
25	-	44	N3	-	77
		37			58
30A	-	94	54	-	376
		73			389

(Average % difference= 21%)

APPENDIX IV

X.R.F. Analysis

Major and trace elements were analysed on pressed powder pellets using standard procedures.

Known standards were run for each element. The following tables list the standard and its known value(K), and the value determined by these analyses(D).

Element	NIM-G		GSP-1		W-1		MRG-1	
	D	K	D	K	D	K	D	K
SiO_2	74.81	75.72	67.77	67.31	52.24	52.72	41.77	39.24
Al_2O_3	13.34	12.09	15.20	15.19	15.07	14.87	7.54	8.56
Fe_2O_3	2.08	1.88	3.76	4.08	11.19	10.13	17.60	16.83
MgO	0.00	0.04	1.66	0.96	6.53	6.63	13.48	13.51
CaO	0.92	0.78	2.17	2.02	10.80	10.98	14.75	14.72
Na_2O	3.49	3.30	2.87	2.80	2.17	2.15	0.74	0.71
K_2O	5.19	5.00	5.66	5.53	0.71	0.64	0.26	0.18
TiO_2	0.12	0.09	0.64	0.66	1.02	1.07	3.67	3.75
MnO	0.03	0.02	0.04	0.04	0.17	0.17	0.15	0.17
P_2O_5	0.01	0.42	0.21	0.28	0.11	0.14	0.04	0.07

Element	GSP-1		W-1		NIM-S		NIM-L	
	D	K	D	K	D	K	D	K
Cr	18	13	122	120	–	–	38	14
Co	8	7	44	50	–	–	24	6?
Pb	62	53	15	8	–	–	40	42
Cu	45	35	111	110	–	–	31	14
Zn	110	98	88	86	–	–	355	360
V	49	49	224	240	–	–	62	78
Rb	245	250	26	21	497	550	180	195
Sr	246	230	190	190	60	64	4600	4600
Y	33	32	24	25	0	3?	29	23
Zr	515	500	102	105	22	45?	–	–
Nb	25	29	10	9.5?	3	3.5?	612	920
Ce	382	390?	25	23?	12	11?	–	–
Nd	139	190	21	15	15	6?	–	–
La	200	200	20	12?	10	4?	–	–
Ba	1436	1300	267	160	2629	2600	–	–

	AGV-1		GSP-1		SY-3		G-2	
Element	D	K	D	K	D	K	D	K
U	4	2	3	2	702	640	2	2
Th	3	6.4?	110	105	928	980?	21	24
	NIM-N		W-1		BCR-1		NIM-D	
	D	K	D	K	D	K	D	K
Ni	110	120	76	78	21	13	2103	2100
	As-100		As-150		As-300		As-3000	
	D	K	D	K	D	K	D	K
As	103	100	151	150	296	300	3000	3000

APPENDIX V

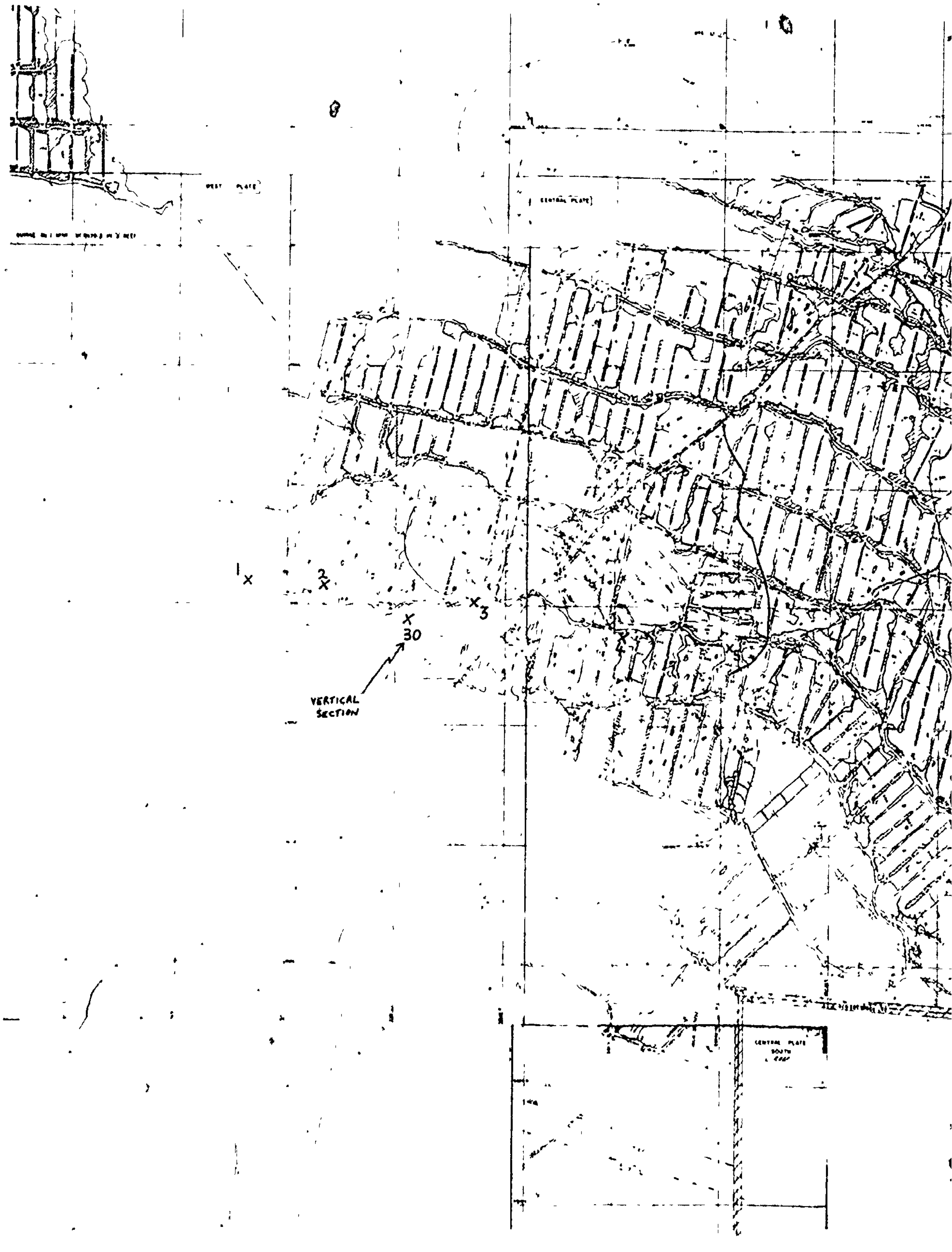
Sample Location Maps

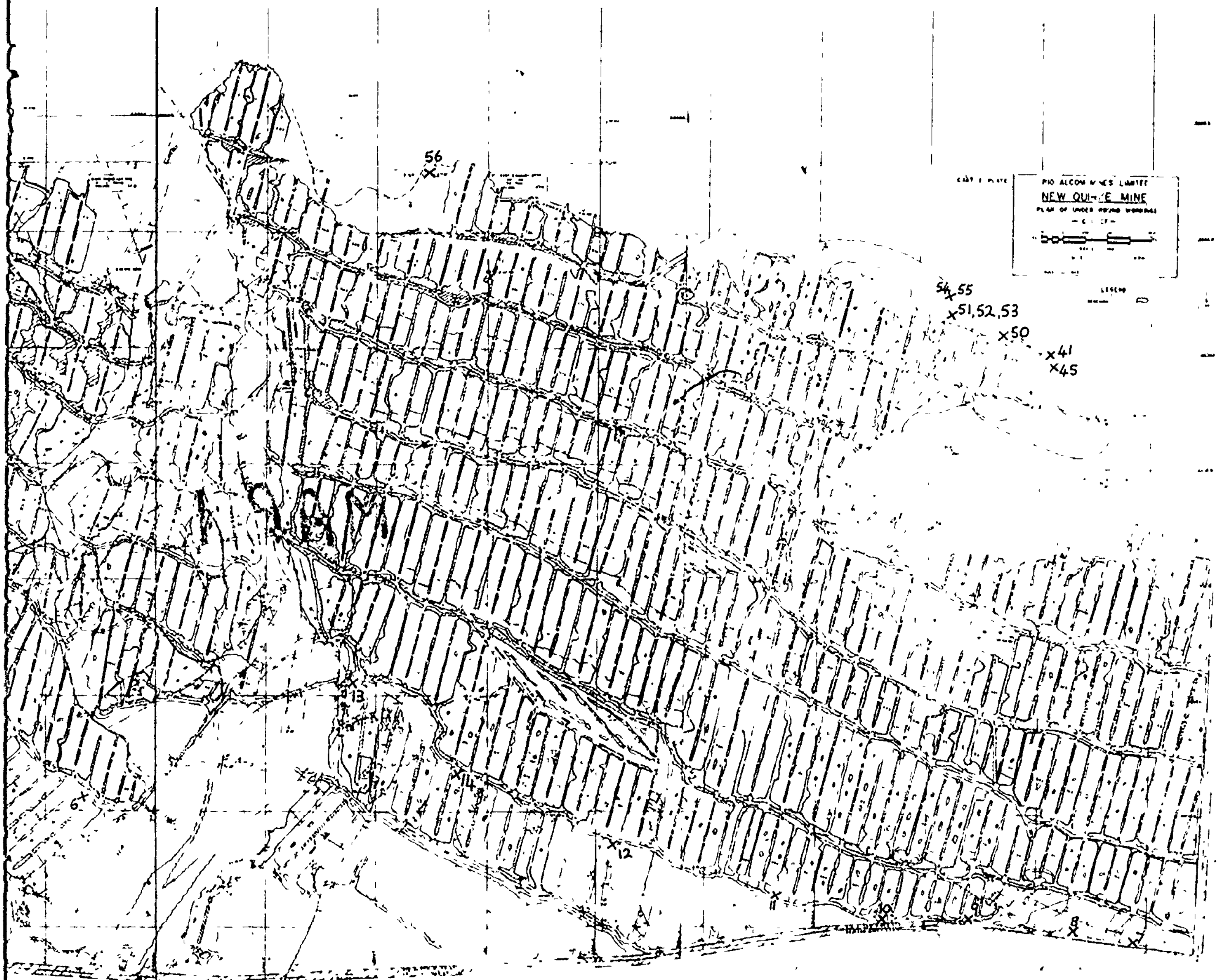
Map 1 : C Reef, New Quirke Mine.

Map 2 : A Reef, New Quirke Mine.

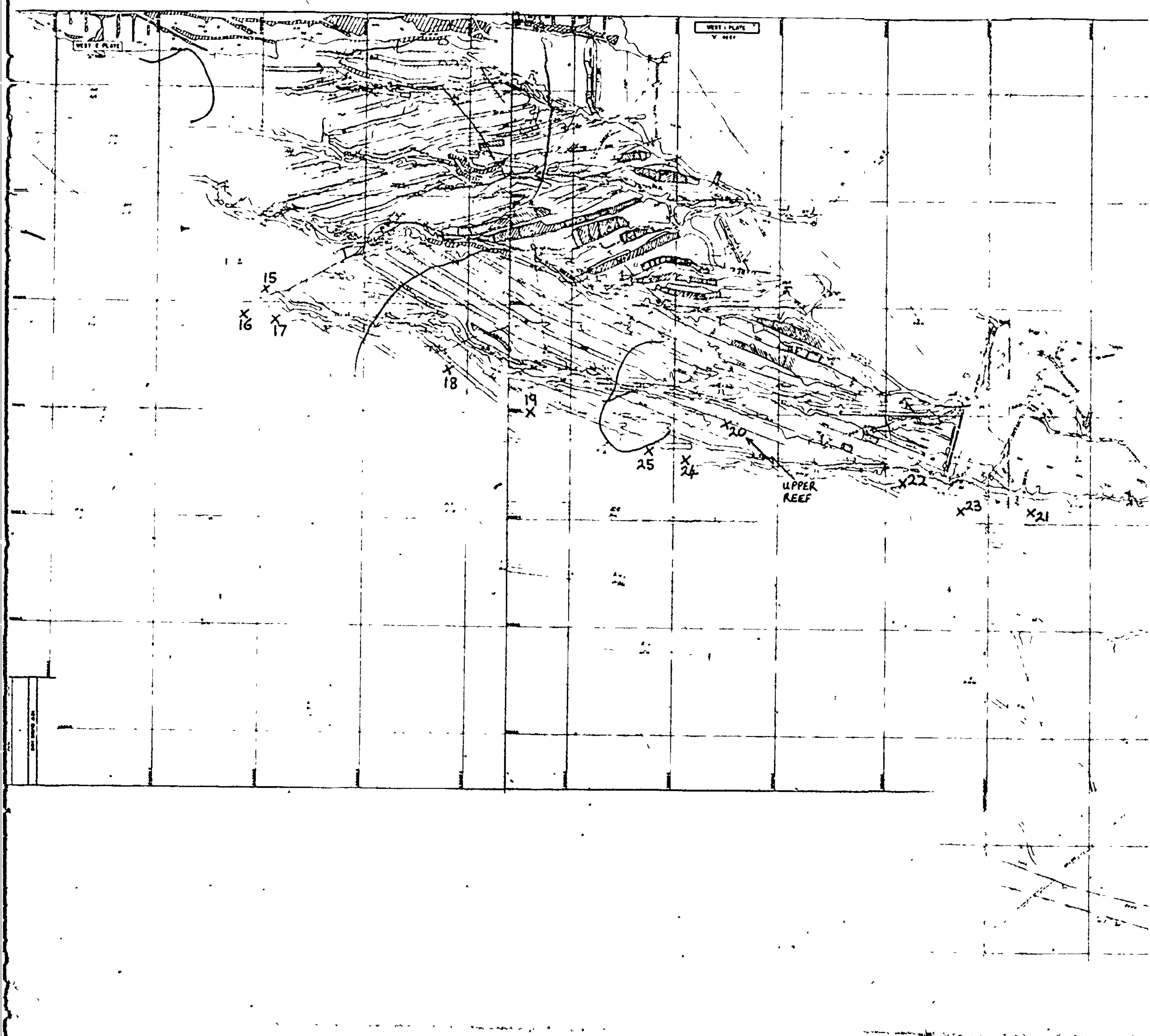
(The above maps were xeroxed from blueprint reductions supplied by Rio Algom Ltd.)

Map 3 : Lower and Upper Reefs, Denison Mine.
(From Martindale 1968)

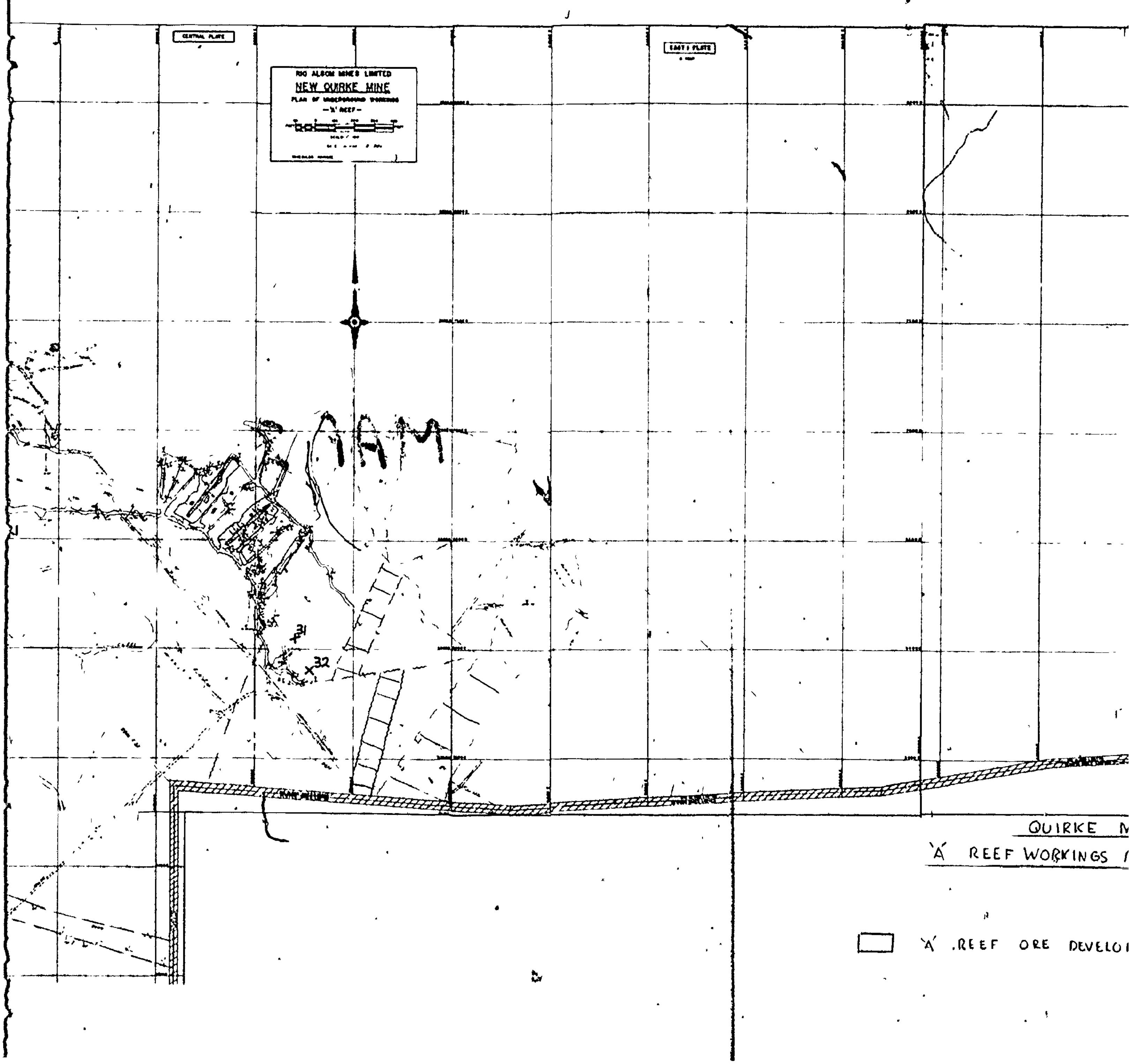

WEST PLATE
CENTRAL PLATE
1 x
2 x
x 3
x 30
4
5
VERTICAL SECTION
CENTRE PLATE SOUTH


56
RIO ALGOM MINES LIMITED
NEW QUIRKE MINE
PLAN OF UNDERGROUND WORKINGS
LEGEND
54
55
51,52,53
50
41
45
3
13
14
6
12
11
10
9
8
7

2OF2


WEST 1 PLATE
WEST 1 PLATE
15
16
17
18
19
20
25
24
UPPER REEF
22
23
21

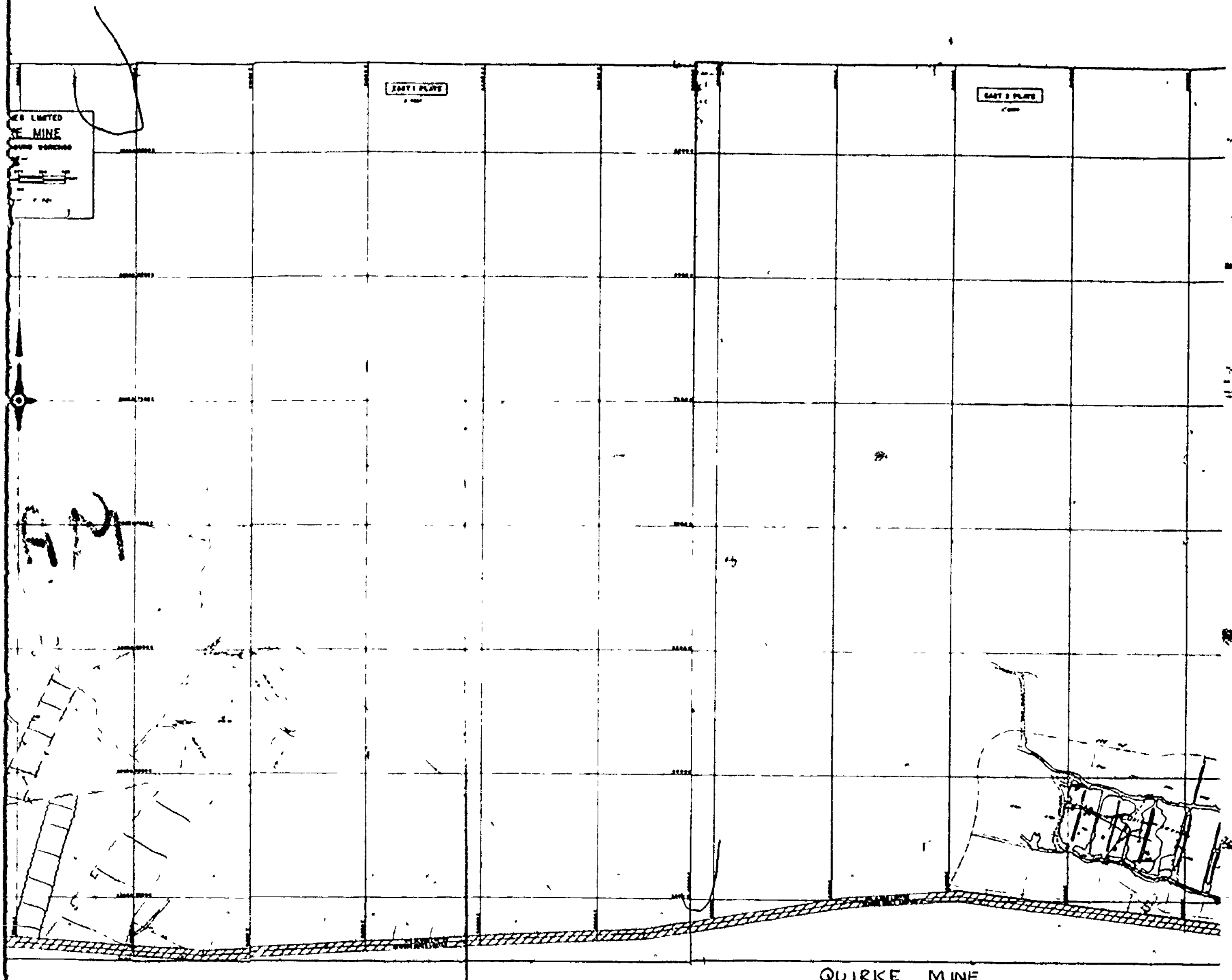
1 /OF 1



QUIRKE M

'A' REEF WORKINGS (

▭ 'A' .REEF ORE DEVELO

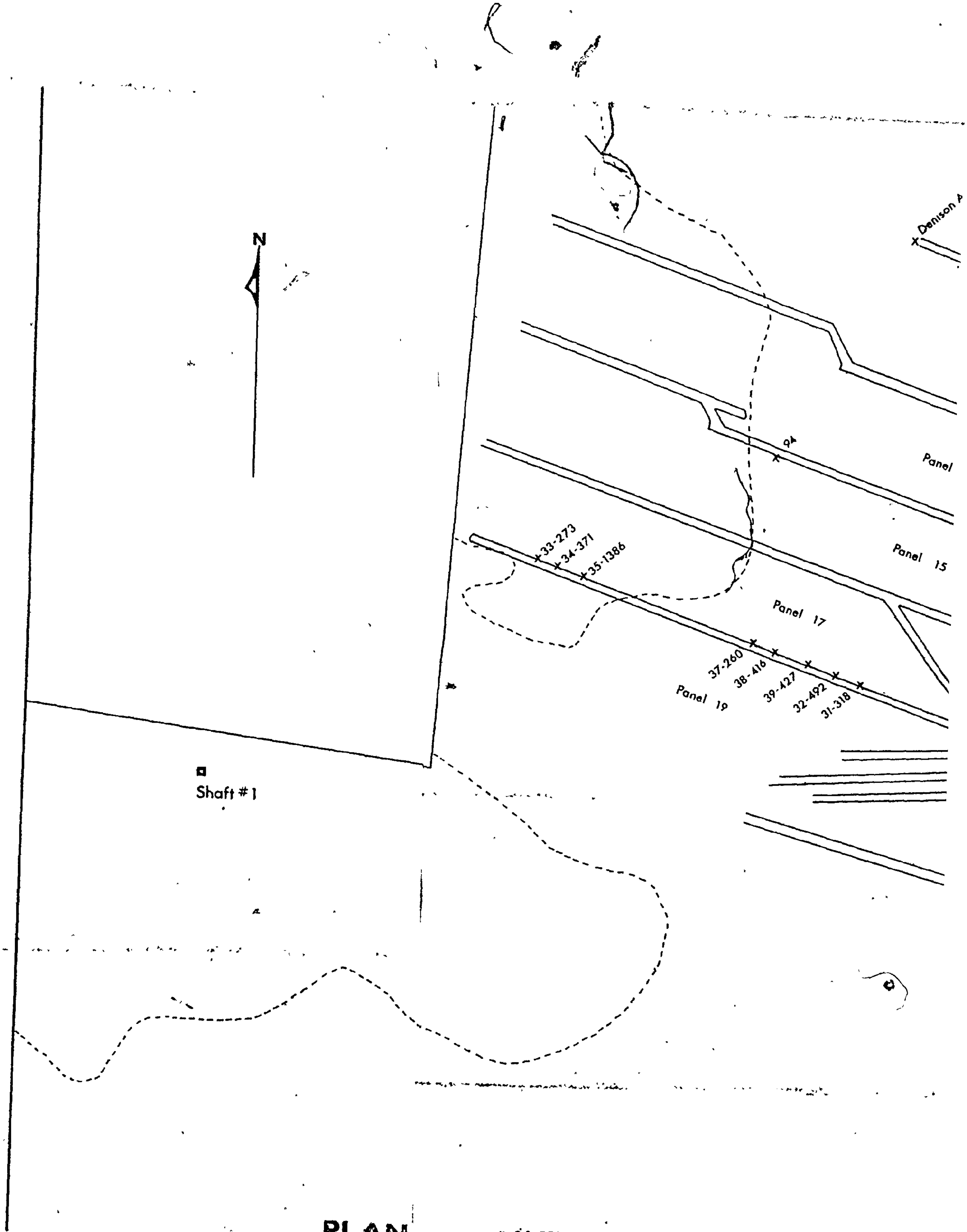


QUIRKE MINE

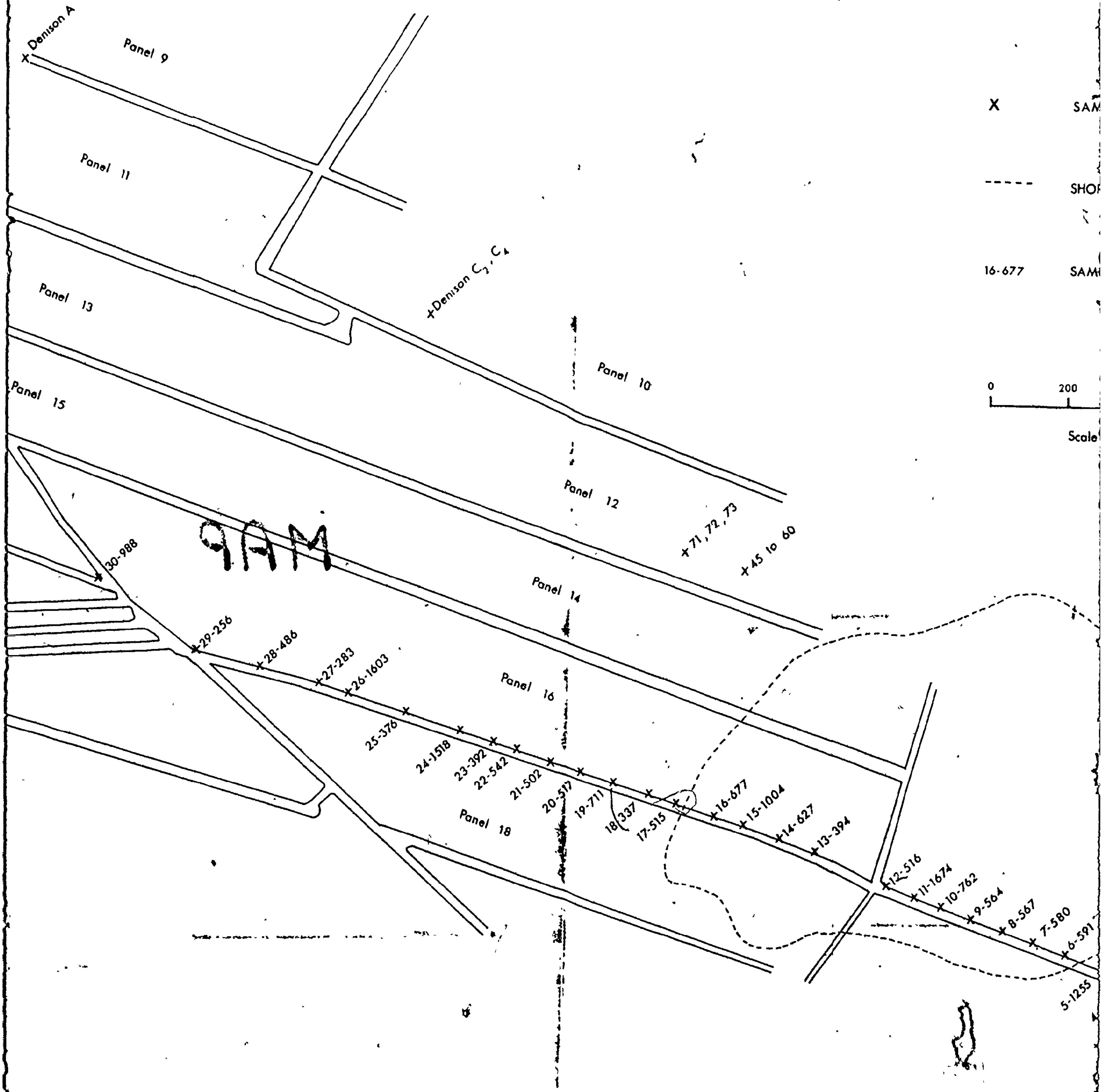
'A' REEF WORKINGS AS OF DEC 31 1979

☐ 'A' REEF ORE DEVELOPMENT IN 1979

scale . 1" = 400'



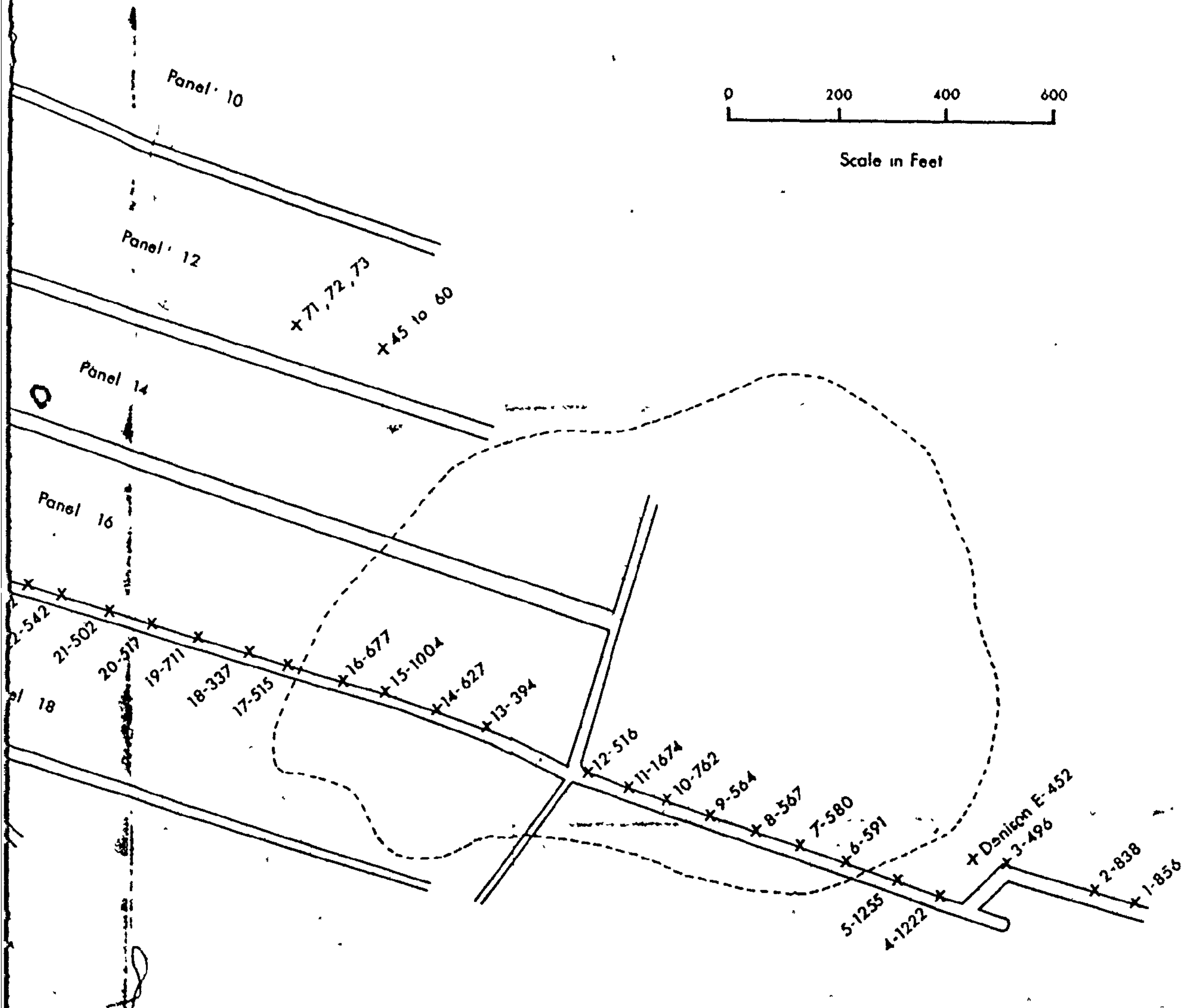
PLAN OF THE UNDERGROUND WORKINGS



IGS OF THE NC RTHEAST PORTION OF

LEGEND

X	SAMPLE LOCATION
-----	SHORELINE of QUIRKE LAKE
16-677	SAMPLE NUMBER and GOLD CONTENT in PPB



N OF DENISON MINE SHOWING SAMPLE LOCATIONS

3 OF 3