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Tittel

Kolsvika Project: Report on geological, diamond drilling and metalurgical investigations.

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Geologi Boring Oppredning		Kolsvik
Råstofftype	**Emneord**	
Malm/metall	Au W As	

Sammendrag

Gull og arsenkis mineraliseringen opptrer i kvartsfylte årer og segregasjoner, men også sammen med massiv eller avsnørt arsenkis på sprekker og skjærsoner i en granitt med antatt alder på 424+/-24 my (Rb-Sr), som ligger i senprekambriske - kaledonske vulkanitter og sedimenter. Ut fra eksisterende data, strekker det seg en gullmineralisert tektonisk sone fra F-området til Seksa, over en avstand på omlag 500m. Boringer indikerer en utstrekning mot dypet på min.200m. Formen på den mineraliserte delen varierer, det samme gjør mineraliseringen selv. Et potensial på 2 mill tonn er antydet/indikert. En eksakt bestemmelse av gehalten er ikke mulig på grunnlag av dagens informasjon, men det er indikasjoner på at en mulig økonomisk gehalt er tilstede.

KOLSVIK PROJECT

Joint venture between

A/S Sulfidmalm and Superior Norge Exploration Company

REPORT ON GEOLOGICAL, DIAMOND DRILLING AND METALLURGICAL INVESTIGATIONS

SEPTEMBER 1983

RONNIE SIVERTSEN
ØYSTEIN MJELDE

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KOLSVIK PROJECT

LOCATION

The Kolsvik gold showing is located at approximately 65°40' E in Bindal community, Nordland County, Norway.

The showing lies on the western side of the Tosenfjord, some 4 kms directly south of Kolsvik Bay. The fjord is ice free year round and extends to considerable depth (up to 700 m).

From Kolsvik Bay there is a distance of 3 kms across the fjord to Lande which has road connections to Brønnøysund. To the local community center of Terråk is a distance of approx. 30 kms by boat.

At the head of Kolsvik Bay, a hydro-electric power station (Åbjøra power-station) is located. In connection with the power-station there is a small shipping quay and a good quality gravel road extending approx. 1 km south towards the gold showing.

Fig. 1. shows the general geographic location of the area. Fig. 2 shows the topographic conditions and location of the gold showings in relation to the fjord.

PREVIOUS WORK IN THE AREA

Gold has been known in the Kolsvik area since the 1920's, and investigations were carried out in the 1930's by a private Norwegian company. This work which mainly consisted of adit driving and sampling was terminated by the start of the Second World War and never recommenced.

The Swedish company Boliden were also involved during this period and were rumoured to be interested in taking over the property, but they could not accept the conditions stipulated by the Norwegian Government at that time.

Since the war the claims in the Kolsvik area have been held by the Norwegian State. Minor investigations were carried out by the Norwegian Geological Survey in 1962 and the property was optioned to A/S Sydvaranger for å short period in the early 1970's.

PRESENT OWNERSHIP

The mining claims to the Kolsvik property are owned by the Norwegian State. A/S Sulfidmalm became interested in the area in 1978 and in 1979 an agreement was signed whereby the Norwegian State optioned to Sulfidmalm

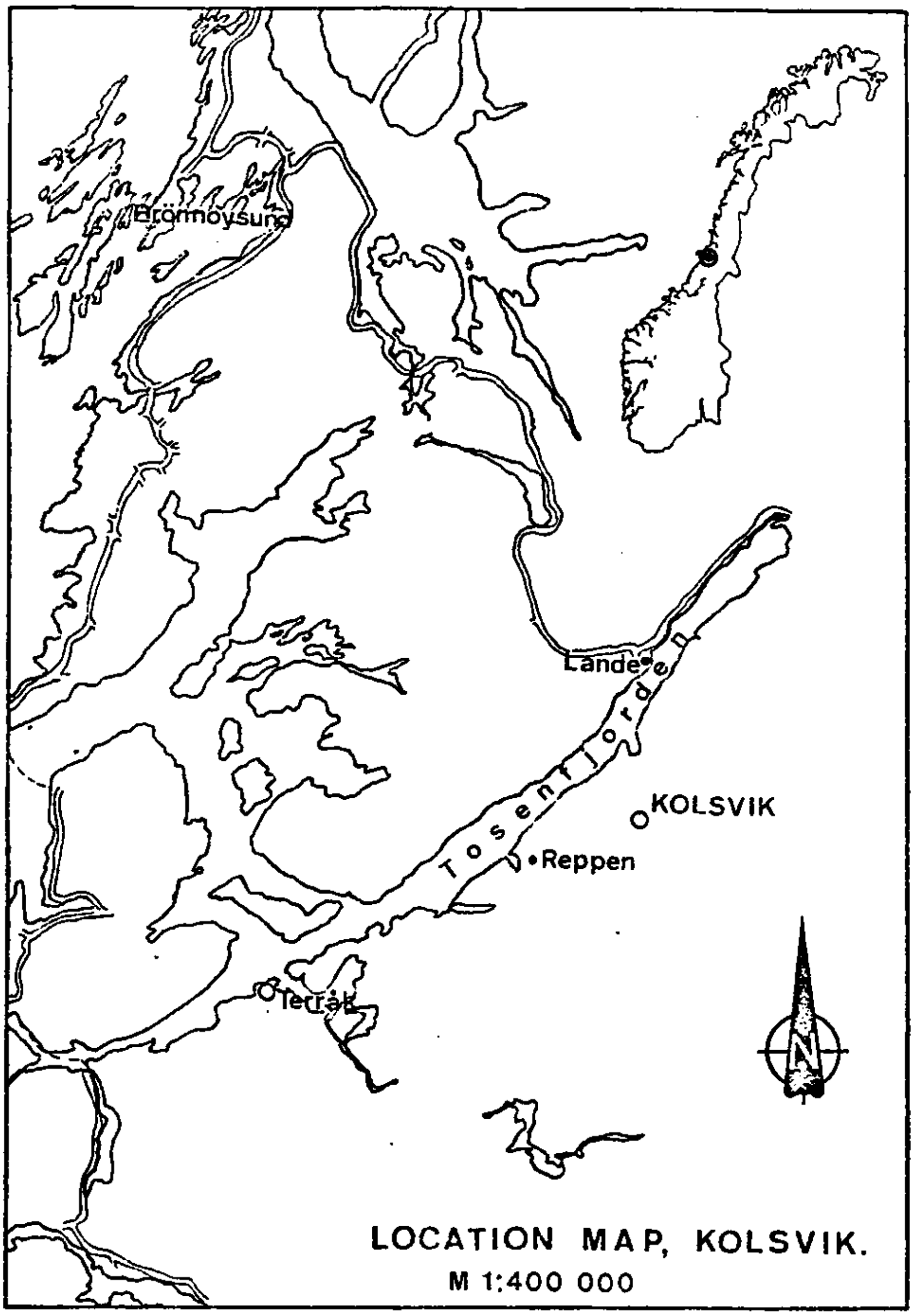

Brönnöysund
Lande
Tosenfjorden
KOLSVIK
Reppen
Terråk
N
LOCATION MAP, KOLSVIK.
M 1:400 000

FIG. 1.

Bogelva
Bogelva
KOLSVIKBOGEN
TOSENFJ
trace of mineralized zone
gravel road
quay
power station

FIG. 2 1 10,000

the Kolsvik claims for a 5 year period.

A/S Sulfidmalm then commenced with exploration activities in the area on their own.

In 1981 Superior Norge Exploration Company (SNEC) became involved in the project and an agreement between Sulfidmalm and SNEC was signed giving SNEC the option to earn up to 49 % interest in the venture.

GEOLOGICAL SETTING

The geology of north-central Norway is dominated by nappes of relatively high grade psammitic, pelitic and calcareous metamorphic rocks with subordinate metavolcanics and with intrusive masses of Caledonian age. The depositional age of the metasediments of the nappe sequence has for a long time been regarded as most probably Cambro-Silurian, but recent age determinations and stratigraphic investigations are indicating that parts of certain successions may be of late Precambrian age.

The rocks in the Bindal region belong to the Helgeland Nappe which is the highest tectono-stratigraphic unit in this part of north-central Norway (fig. 3, fig. 4.).

The area is dominated by basic intermediate and granitoid intrusives, some of which are extremely large in areal extent.

The granitic bodies show marked age differences and represent a complex batholitic development. The largest granitic body, the Bindal granite has given a Rb-Sr whole rock age of 424±26 m.y.

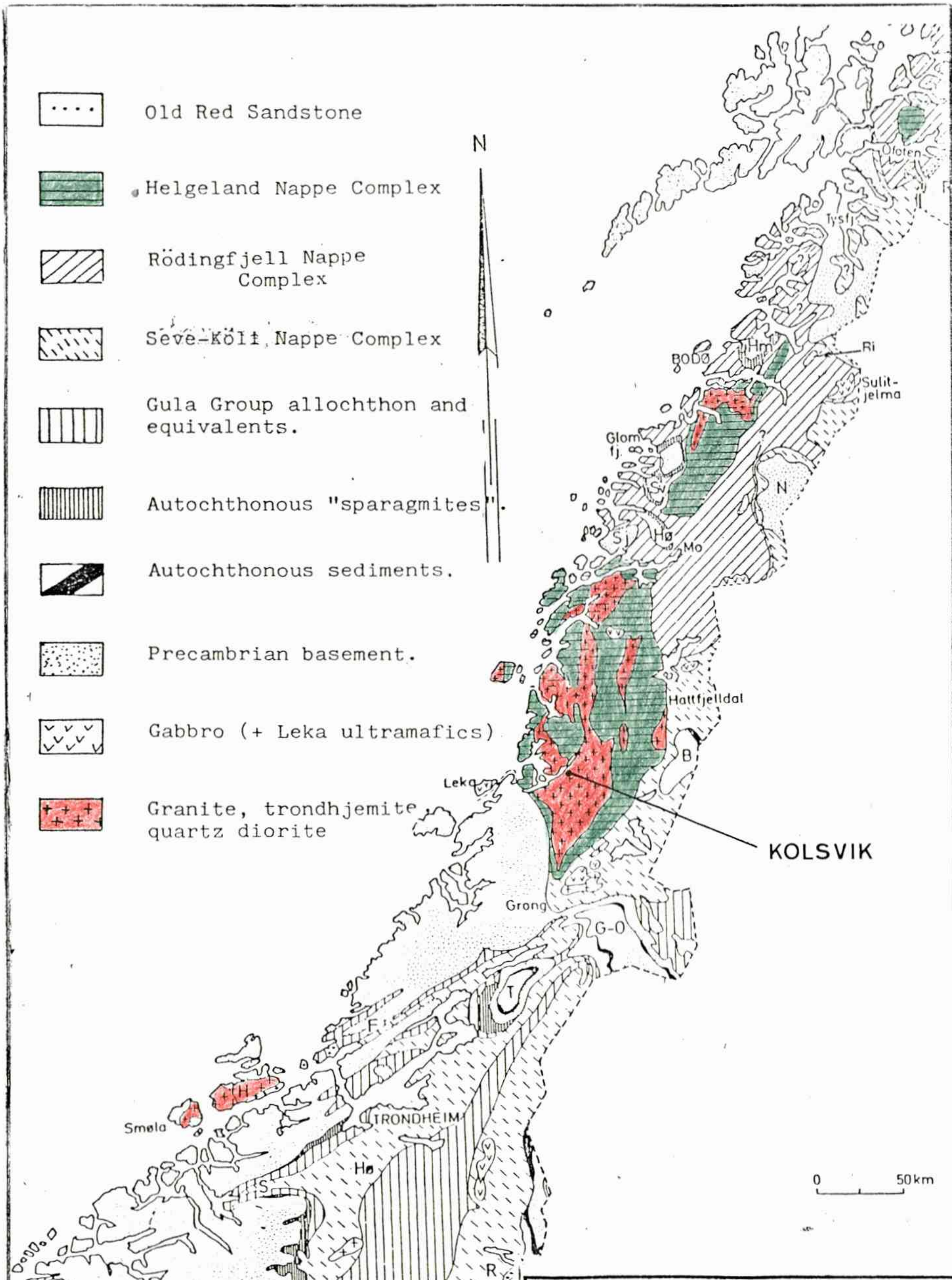
The immediate carapace to the granitic rocks of the region would appear to be of oceanic crust (ophiolite) with an unconformable or Palaeozoic cover sequence of psammitic pelitic and calcareous rocks.

The result of reconnaissance studies on the tectono-stratigraphy of these units reveal that several major thrust nappes must be present within the confines of the Helgeland Nappe itself.

Apart from Kolsvik, gold is also present at several other localities in the immediate area - one of these areas, Reppen, some 6 kms to the west of Kolsvik is at present under investigation.

The area is also notable for its scheelite mineralization which again is the object of considerable exploration interest

Fig. 5 shows the geology of the immediate area to Kolsvik.



A/S SULFIDMALM

GEOLOGY AND LOCATION

CENTRAL NORWAY

SCALE	DRAWN
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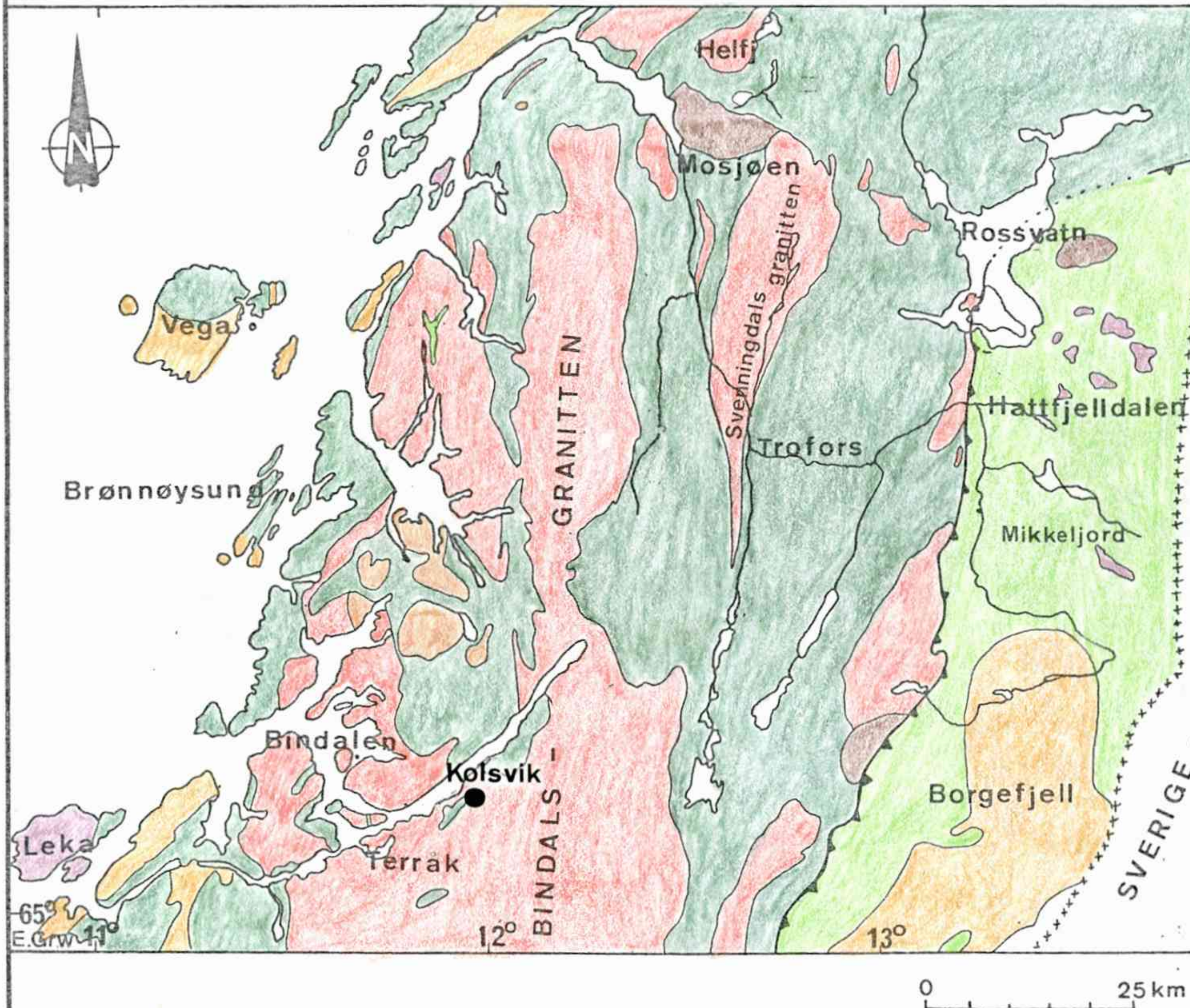
FIG. 3



High grade cambrian-silurian sediments

Low grade – " – " – " – " – " – " –

Caledonian granites, granodiorites, diorites.

Hypersthene monzodiorite

Ultrabasics

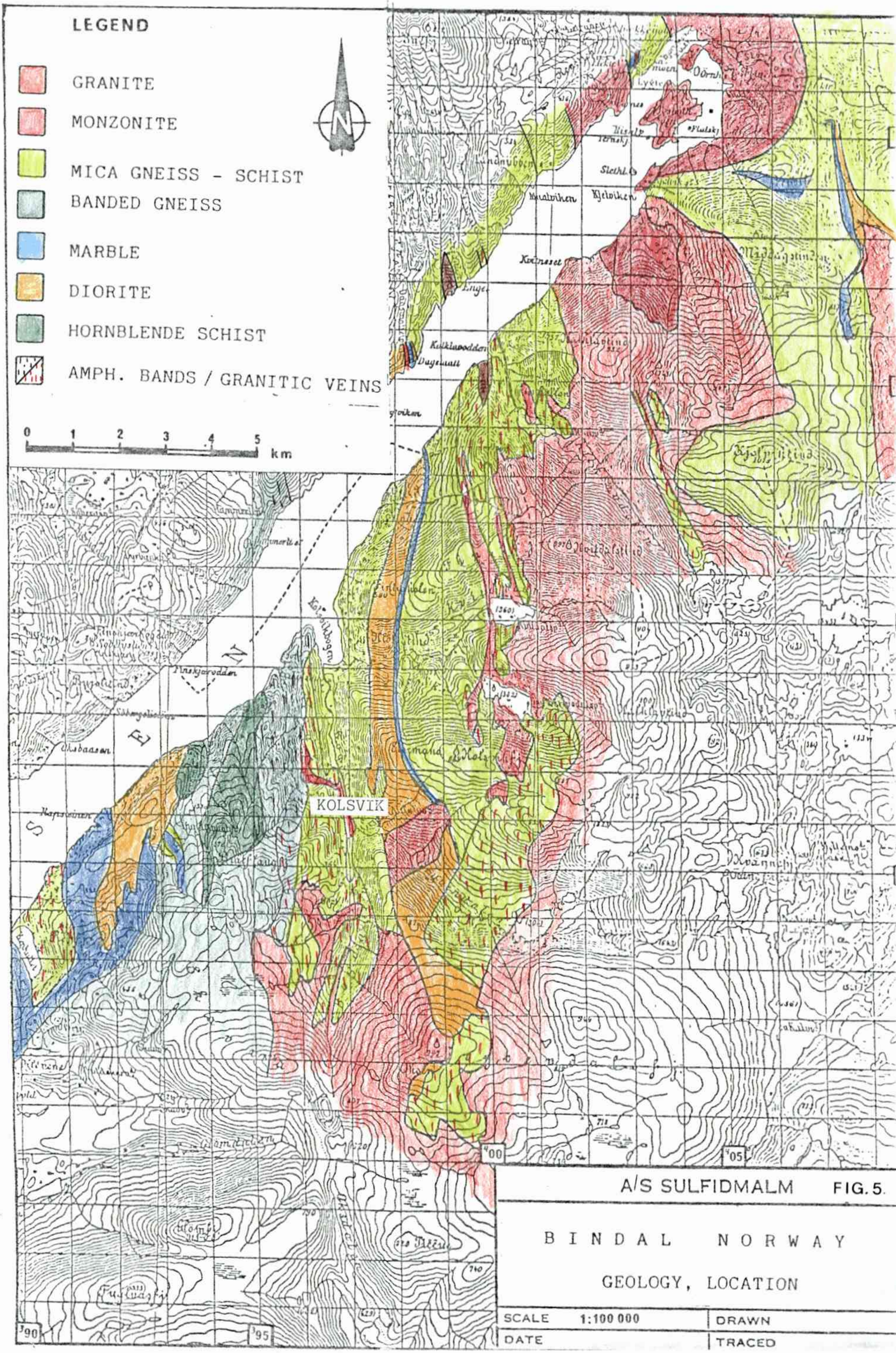
Gabbro

Precambrian rocks

Thrust

FIG. 4.

A/S SULFIDMALM			
GEOLOGICAL MAP SØR HELGELAND			
SCALE	1:750 000	DRAWN	
DATE	3.82	TRACED	AKB



LEGEND
GRANITE
MONZONITE
MICA GNEISS - SCHIST
BANDED GNEISS
MARBLE
DIORITE
HORNBLENDE SCHIST
AMPH. BANDS / GRANITIC VEINS
0 1 2 3 4 5 km
KOLSVIK
A/S SULFIDMALM
FIG. 5.
B I N D A L   N O R W A Y
GEOLOGY, LOCATION
SCALE 1:100 000
DRAWN
DATE
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WORK CARRIED OUT ON THE PROPERTY

1979 Initial location, mapping and sampling of several areas of gold/ arsenopyrite mineralization in the region.

1980 Regional mapping and regional geochemical sampling. Detailed mapping, sampling and diamond drilling at Kolsvik: - 4 holes totalling 390.35 m.

1981 Detailed geological mapping at structural interpretation in the Kolsvik area. Detailed sampling of surface showings and adits. diamond drilling 1.516.3 m in 15 holes.
Metallurgical testing of the Kolsvik mineralization.
Detailed mapping and sampling of alluvial and galciofluvial deposits north of the Kolsvik showing.

1982 Drilling 1 468.4 m in 15 holes.
Extra metallurgical testing.

DESCRIPTION OF THE PROPERTY

In describing the property various terms from the 1930 investigations have been used, and a short description of the area is given here, and is also shown on fig. 6.

The southernmost outcrops in the mountainside on the east side of the Bogdalen River are called the F-zone. The Storstein adit is driven along the F-zone. Moving north and down towards the river we find the Kaffistein adit.

Along the western side of the Bogdal River are a series of five adits comprising what is termed the C-zone. The adits from south to north are named Hartvig, Mannerheim, Boliden, South Skar and North Skar.

Immediately across the river from South Skar is a small showing termed the D-zone.

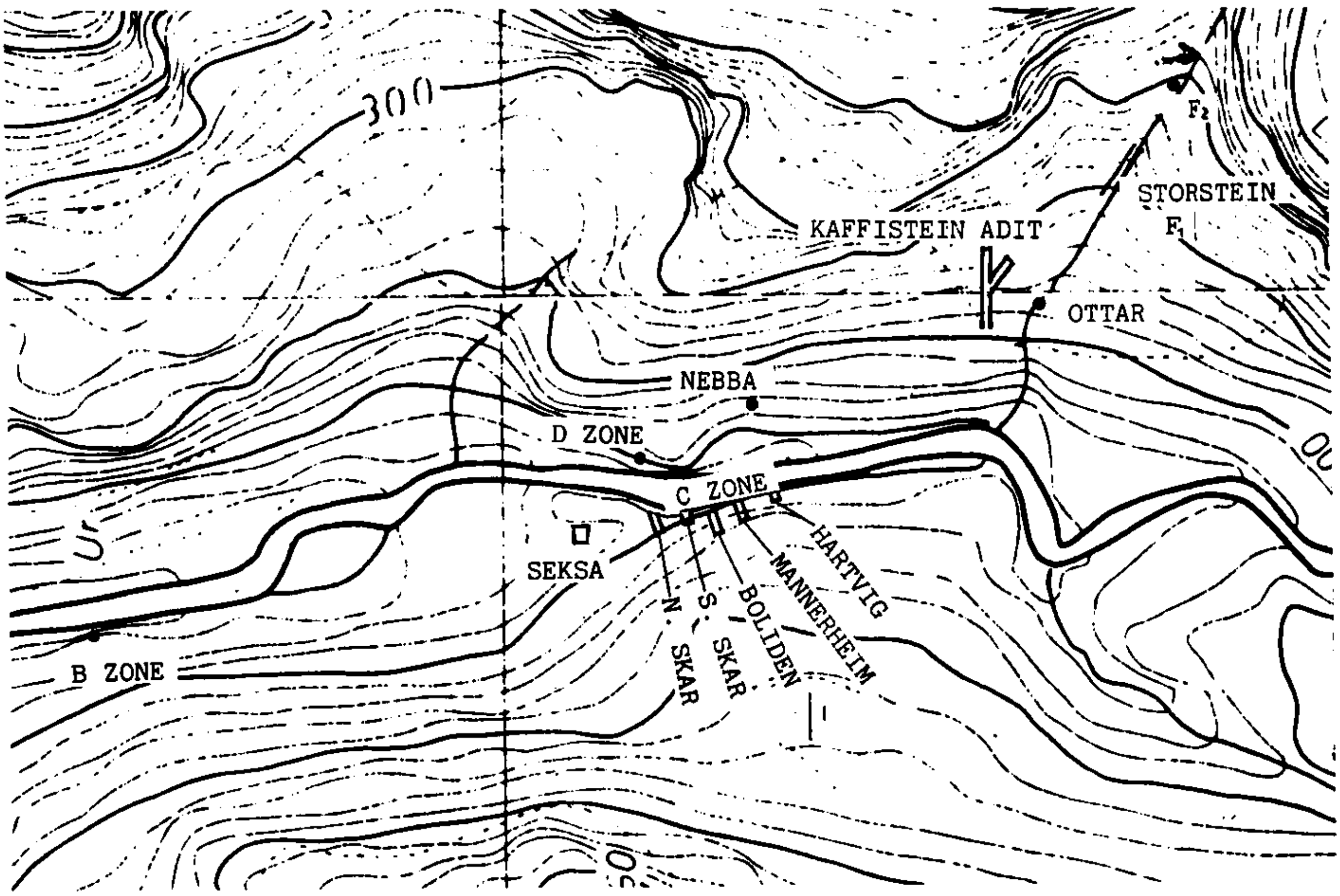
Further north from the C-zone is an old waterfilled shaft termed Seksa.

From Seksa there is a distance of some 300 m north to the B-area.

GEOLOGY AND MINERALIZATION

The major lithologies found in the Kolsvik area are:

I. Granite
II. Augen gneiss / banded gneiss (altered monzonite)
III. Marble
IV. Mica schists.



SCALE 1:5000

FIG. 7.

LOCALITY NAMES IN KOLSVIK

I.) Granite

The notable feature of the granite in the Kolsvik area is its general lack of mafic constituents. In many cases its composition is simply quartz and feldspar (orthoclase, oligoclase, microcline). More biotite rich phases are only seen locally.

The granite is usually without any planar structure, but dark variants may show a weak biotite foliation.

The granite often shows alteration in the vicinity of tectonic zones, where carbonate, sericite, muscovite and chlorite are common. A characteristic pinkish alteration is also developed along joints. These joints are often lined with secondary minerals such as desmin, lammonite, ankerite, calcite and quartz. Especially quartz and carbonate veining is common.

Disseminated arsenopyrite is frequently seen in the vicinity of tectonic structures and is usually accompanied by alteration products. The quartz-gold and arsenopyrite bearing veins and segregations are usually limited to the granite. Good Au mineralization is often seen to be related to highly altered red granite especially in the C-area.

II.) The gneisses

The gneisses in the Kolsvik area vary in composition and texture from augen-/banded gneisses and dioritic gneisses to more schistose mica variants of these.

The augen-/banded gneiss structurally overlies the other rocks and can be seen especially in the F- and Kaffistein areas. It is a biotite rich rock with augen or bands of plagioclase and quartz. A planar structure is well developed and shows a constant N-S strike and steep dip towards E.

The diorite gneiss is usually more massive, but occasionally it shows foliation in more mica rich parts. The contacts between diorite gneiss and other gneisses and schists are generally diffuse, especially in sheared areas. Definite intrusive diorite is seen at several locations (especially in drill holes) but texturally similar rocks are also seen in sequences assumed to be metasediments.

In pol-thin section several of the augen and dioritic gneisses are shown to have a quartz monzonite composition, and often the more massive varieties, although having a distinct augen texture in hand specimen, exhibit a granitic texture in section with scattered coarse flakes of biotite and muscovite occuring in a coarse mosaic of feldspar, - both sodic and potassic and quartz.

The gneisses are cut by a great number of veins and at least three phases of granitic veins are noted, the earliest veins being highly deformed. Aspy mineralization is rare, but can be seen in some quartz and granitic veins. Py is a common mineral in both dioritic- and augen/banded gneisses.

III. The marble

The marbles (dominantly calcite marble) are all highly deformed rocks. They vary in composition and texture from banded marble, containing thin bands of pelitic composition which are often folded to highly deformed fragment rich marble, now showing a breccia texture.

A rapid interchange between marble and carbonate rich mica schists is seen in drill holes from the C-area.

Skarn (diopside-garnet) zones are frequently developed in the marble, especially in contact relations to younger crosscutting granite.

IV. Mica schists

The mica schists vary from fine to medium grained, mostly strongly sheared biotitic rocks. They are mainly found in or adjacent to shear zones, especially well developed in the C-area.

The mineralogical and textural variations of the schists are thought to represent both a primary change in the sequence and a strongly variable deformation of the rocks.

V. Mineralization

The gold and arsenopyrite mineralization occurs dominantly in granite near the contact zone with gneisses and metasediments. The mineralization is typically tectonically controlled and related to such structures as

a) Quartz vein fillings in fractures, shears and joints.
b) Quartz segregations in or associated to the above structures.
c) Quartz/Asp matrix fill in breccias.
d) Massive Asp zones in fractures and shears.
e) Joint smearings of Asp.

Relationships of tectonics and mineralization and extent of mineralization will be treated later in this report.

Two typical quartz vein type mineralizations show the following in polished thin section

Sample PTS 5629 C zone vein type

		Grain size (mm) max.	avg.
Quartz	95 %		
Muscovite	tr.		
Arsenopyrite	3-4 %	0.75	0.40
Native gold	1 %	0.25	0.05

Masses of euhedral arsenopyrite grains, locally intergrown with coarse blebs of native gold occupy fracture zones within a coarse interlocking quartz mosaic. Muscovite is the sole alteration mineral associated with the mineralization. Individual quartz grains exhibit undulose, strained extinction and together with arsenopyrite are commonly criss-crossed with microfractures. The latter manifest themselves in the form of thin "tracks" of microcrystalline quartz within the coarser vein quartz and quartz filled fractures transecting arsenopyrite grains.

Sample PTS 5630 C zone vein type

		Grain size max.	avg.
Quartz	55-60 %		
Alkali feldspar	4-5 %		
Carbonate	tr.		
Chlorite. Biotite	tr.		
Arsenopyrite	35-40 %	massive	
Galena	tr.		
Native gold	tr.	0.006	0.006
Rutile	tr.		

Texturally this sample is similar to PTS 5629. From a mineralogical point of view, however, subtle yet distinct differences exist. In place of muscovite an alteration assemblage of carbonate and chlorite/biotite is found associated with the arsenopyrite in fracture zones. Minor coarse grained K feldspar joins the quartz gangue and occurs both as localized grain aggregates and as isolated single crystals.

These two samples represent typical vein type mineralization which is common through the property. Another type of mineralization in the area and common in the F zone is a "breccia type". A typical PTS shows the following

Sample PTS 5631 F zone breccia type

		Grain size (mm) max.	avg.
Quartz	15-20 %		
K Feldspar Plagioclase (Albite)	65-70 %		
Chlorite	‹1		
Apatite	tr.		
Sericite	tr.		
Arsenopyrite	5-10 %	3.00	1.50
Rutile	‹1		
Zircon	tr.		
Native gold	tr.	0.006	0.006

Here masses of arsenopyrite together with associated chlorite alteration occur within fracture zones. The granitic host rock which has been strongly shattered consists of predominantly coarse interlocking K feldspar and albite grains with lesser interstitial (=primary) and fracture-filling (=secondary) quartz.

Scheelite has been noted in several of the gold bearing veins and detrital cassiterite has been found in glaciofluvial deposits north of the area.

STRUCTURAL OBSERVATIONS

I. Summary

The Kolsvik valley to which the gold property is located is a deeply glaciated valley, the course of which is influenced by the strong shattering assocaited with a major fault zone with a north south trend extending along the valley floor. This fault zone is a dominant structural feature, can be traced for some tens of kilometers and is readily seen on ERTS satellite images.

The lithological assemblage of the area has been variably affected by late Caledonian and subsequent deformation as revealed in fault, shears and joint systems. It is these faults, shears and joints which provided the passage for mineral-bearing solutions or the redistribution and concentration of metals.

Several categories of fracture characterize the late tectonic fabric of the Kolsvik district.

1) Shear zones and faults marked by zones of crush and or shear.
2) Joints.
3) Later joints and shear zones - possibly non Caledonian.
4) Rebound joints i.e. parallel to the ground surface.

Categories 1 and 2 are Caledonian in age and relate to granite emplacement and subsequent Caledonian tectonics.

Gold mineralization appears to occur chiefly in shear fractures, faults or joints together with arsenopyrite or in association with a gangue of quartz in which arsenopyrite can occur as fine disseminations, veinlets or irregular segregations. Native gold is commonly seen in the area and is most common in association with quartz. The arsenopyrite and/or quartz arsenopyrite veins usually occur as thin discontinuous veins or less regular elliptical bodies within the fractures. Vein quartz - sometimes Asp and Au bearing also occurs in systems of tension gash veins associated to some of the minor faults.

The most conspicuous development of sulphide occurs in very brittle rocks which become more heavily broken or diced up with successive fracture systems. Massive arsenopyrite fills the fractures, frequently giving the rock the appearance of a fault breccia.

Mineralization has been found on surface over an intermittent strike length of some 800 m from the F zone in the south through the C zone to B in the north. Diamond drilling has been concentrated between and around the F and C zones. Integrating the data from zones F, C and B brings out several features which are summarized below:

1) Each zone displays a rational but somewhat different pattern, indicating they are near coherent sub areas of a large tectonic framework.
2) Two systems of fractures seem to be significant in the distribution of mineralization in the area. In chronological sequence these are
   a) Conjugate system of gentle to moderately inclined shears and joints with an average 160° strike. The hanging wall in each case moves downwards indicative of a sub horizontal extension of the rocks. Tension gash veins of quartz are associated with these fractures in the more brittle rocks. These flat shears often contain development of massive Asp or elliptical vein quartz with Asp and Au. This conjugate

system is well seen in the C zone adits and the Kaffistein adit.

b) Steep shears-faults and joints with an average SE-NW trend (strike spread 90°-170°). They are well developed in the F zone, inner Kaffistein adit and in the C zone. The fractures frequently exhibit a suite of associated tension gash veins. The relative age relationships between the fracture systems can be seen in the C zone (Boliden adit) and in the Kaffistein adit where NNW-SSE and N-S fractures postdate the flat conjugate system.
These "b" type shears are quite dominant and some can be traced for several tens of meters as in the F and C zones.

3) The conjugate system of flat shears is compatible with sub-horizontal extension of the rocks i.e. distension above a rising plutonic mass of granite.

4) Stereographic plots indicate that despite their temporal difference the "a" and "b" systems belong to the same orogenic cycle.

5) The earliest phase of mineralization was emplacement of sulphide and sulphide-metal bearing vein quartz along the conjugate system of flat to moderately inclined fractures of "normal" type i.e. hanging-wall moves downwards.

6) Later faulting has affected redistribution of sulphides, in some cases producing a conspicuous increase in porosity and potential mineral sinks. In several places such as the F zone dramatic breakage occurs and when impregnated with massive sulphide the rock mass has the appearance of a breccia.

7) The major fault zone in the valley floor is a later event. It has effected disturbance of the mineralization and its associated fractures but the fault itself seems to carry no gold and is characterized by a low temp mineral assemblage.

8) Continuity of the various tectonic units can be established in places from surface observations and sporadic continuity can be intrepreted from drill holes. Within the tectonic units the general pattern appears to be one of somewhat erratic distribution of mineralization as demonstrated by assay results and as is to be expected in this type of deposit.

MINERALIZATION AND TECTONICS

The earliest mineralization seen is related to low angle conjugate joints supposedly related to granite intrusion. The most dominating mineralized structures in the area however are several easterly dipping and NW-SE (90°-170°) striking faults and shears with related minor fractures, shears and tension cracks. Brecciated zones are often developed as in the F zone.

Mapping and drilling in 1980/82 has indicated a "structurally controlled zone" extending from the F_2 area in the south to the B area in the north, a distance of some 900 m. The northernmost 300 m between Seksa and the B zone is completely covered by scree and offers no exposure and has not been drill tested.

The elevation difference between F_2 and B is 180 m.

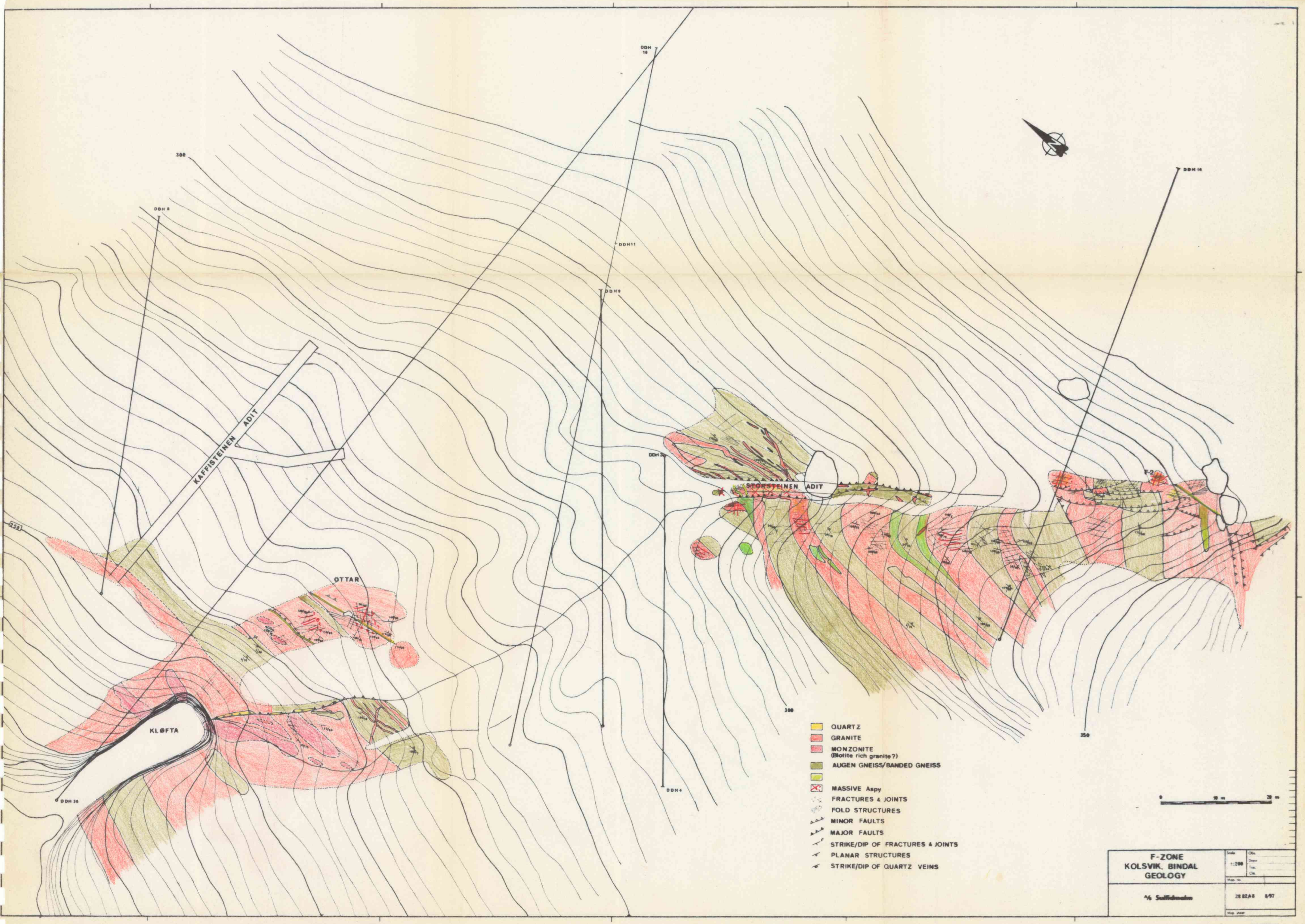
This mineralized zone is cut by the late major N/S fault system in the valley floor - the Bogdalen fault. Splays on this fault parallel earlier NW-SE trending fractures and have caused minor re-orientation (dragging) and/or displacement. No evidence of major displacement has been established.

For purposes of description the property can be divided into two areas: - the area from F to C zones and the C zone to B zone area.

a) The F - C area (fig. 7)

Mineralization in this area can be studied on surface in the Storstein adit, the Kaffistein adit and in the Ottar, Oppgangen and Nebba areas. The following drill holes are also located in this area: DDH 3, 4, 8, 9, 10, 11, 12, 13, 20 and 36.

The F-zone on which the Storstein adit is located consists of two major steep faults with an undulating trend. At the mouth of the adit the distance between the two faults is some 5 m narrowing to the south where they converge some 28 m within the adit again opening up further south. The granitic rocks between these fractures are well mineralized with arsenopyrite chlorite-quartz along steep fractures trending 120° and 180° - this gives a marked breccia appearance to the rock. Massive arsenopyrite occurs intermittently near the footwall of the easternmost fault. In the footwall to the westernmost fault related minor fractures and joints carrying arsenopyrite and quartz are present over a distance of some 20 m. Surface sampling has returned 10.63 Au g/t from bulk channel sampling over the easternmost 4.5 m of the zone at the mouth of the adit.



DDH 10
DDH 14
300
DDH 8
DDH11
DDH 9
DDH 3
DDH 4
DDH 36
KAFFISTEINEN ADIT
STORSTEINEN ADIT
OTTAR
KLØFTA
F-2
300
350
QUARTZ
GRANITE
MONZONITE
(Biotite rich granite?)
AUGEN GNEISS/BANDED GNEISS
MASSIVE Aspy
FRACTURES & JOINTS
FOLD STRUCTURES
MINOR FAULTS
MAJOR FAULTS
STRIKE/DIP OF FRACTURES & JOINTS
PLANAR STRUCTURES
STRIKE/DIP OF QUARTZ VEINS
0
10 m
20 m
F-ZONE
KOLSVIK, BINDAL
GEOLOGY
Scale
1:200
Obs.
Drawn
Trac.
Chk.
Map. no.
A/S Sulfidmalm
28 82A8 8/97
Map. sheet

The F_2 showing located some 30 m to the SE and 40 m higher elevation returned 6.22 Au g/t over 1.5 m.

The Ottar showing located some 30 m below the F zone is interpreted as the western fault observed in the F zone. Two grab samples from Ottar sampled in 1980 indicate 4.5 g/t Au and 14.9 g/t Au over 0.5 m.

In the Kaffistein adit two well mineralized (Asp, Quartz) zones are seen with related joint and fracture mineralization. Low conjugate fracture sets of the earliest generation are also seen in this area to predate the later fractures. The zone of mineralization is of the order of 15 m, but chip sampling has revealed low numbers, 2g/t Au over 2 m.

The Oppgangen and Nebba areas are extremely poorly exposed but early conjugate fractures have been recognized being cut by later NW/SE fractures. Surface sampling has given 5.1 g/t over 1 m (Oppgangen) and 3.04 g/t over 7 m (Nebba).

Small surface showings have also been located at the D zone 22.4 g/t over 1 m and below the collar of DDH 12/13 4.7 g/t over 0.3 m.

A total of 11 drillholes have been drilled in this area. The topography is extremely difficult with the trace of the zone trending across a steep rugged valley side with most of the area being covered by large masses of scree and boulders. This necessitated most of the holes being drilled from the "wrong" side i.e. footwall side of the zone.

Two holes, DDH 3 and 4 were put down on the F-zone in 1980. DDH 3 proved the depth down to at least 90 m with the best values of 9.31 g/t Au over 3.25 m. DDH 4 intersected 22.3 g/t over 0.75 m which is interpreted as footwall mineralization.

The geology and assays of the holes are shown on enclosed sections. All of the holes intersected structurally controlled arsenopyrite/quartz mineralization and visible gold was noted from DDH 8, 12 and 13.

DDH 9, 10, 11 were put down to test the northward continuation of the F-zone. DDH 9 returned only traces of gold (3.43 g/t over 0.25 m). DDH 10 gave 4.88 g/t AU over 5.0 m. DDH 11 returned 3.38 g/t Au over 5 m (5.69 g/t over 2.5 m).

DDH 8 drilled to confirm the supposed northerly extension of the Kaffistein adit mineralization gave 3.96 g/t Au over 4.75 m (5.63 g/t Au / 0.75 m - 7.82 g/t / 1.75 m.)

DDH 12 and 13 were drilled to test the northerly continuation of the DDH 8 mineralization. DDH 12 hit 10.40 g/t Au over 1.5 m (5.22 g/t over 3.5 m) whereas in DDH 13 two zones were intersected - 8.06 g/t Au / 3 m and 5.8 g/t Au.

DDH 20 intersected only two minor gold values over 0.5 m.

DDH 36 put down to intersect the F-zone at depth intersected minor mineralization between 117 and 125 m.

TABLE 1

Summary of DDH's drilled in the F - C area.

DDH	LOCATION	DIP	LENGTH	SIGNIFICANT ASSAYS FROM	TO	LENGTH	Au g/t
3	352 S - 158 E	90°	94.20 m	60.0 62.0 65.25 79.50 87.50	61.0 62.5 66.50 80.0 90.75	1.0 0.5 1.25 0.5 3.25	3.3 2.05 4.88 15.0 9.31
4	352 S - 158 E	50°	93.05	17.0 28.75	18.0 29.5	1.0 0.75	4.05 22.3
8	285 S - 83 E	40°	88.30	55.5 58.50 61.75	56.25 60.25 62.25	0.75 1.75 0.50	5.63 7.82 1.03
9	373 S - 113 E	35°	94.6	63.75 68.0 80.0 80.5	64.0 68.25 80.25 80.75	0.25 0.25 0.25 0.25	2.4 1.1 3.43 1.03
10	362 S - 101 E	36°	144.0	54.0	59.0	5.0	4.88
11	362 S - 101 E	55°	159.3	114.0 (116.5	119.0 119.0	5.0 2.5	3.38 5.69)
12	201 S - 50 E	38°	124.5	38.0 (40.0	41.5 41.5	3.5 1.5	5.22 10.4)
13	201 S - 50 E	20°	63.7	30.0 40.5	33.0 41.5	3.0 1.0	8.06 5.8
14	420 S - 168 E	42°	120.8	NOT ASSAYED			
20	130 S - 30 E	45°	89.8	17.5	18.0	0.5	1.53
36	300 S - 42 E	36°	271.5	123.0	124.0	1.0	0.83

From the available surface information and drill hole data an overall continuous "mineralized zone" extending from F- to the C- area is indicated.

DDH 3 has indicated a minimum depth of 90 m.

b) The C-area (fig. 8.)

The C-area is dominated by strong shearing/faulting with a NNW-SSE direction and a steep easterly dip. A marked fault zone follows the contact between the granite and the country rocks.

This fault zone can be traced for some 125 - 150 m along strike. Coincident and partly enclosed in the fault zone are quartz-arsenopyrite veins and irregular bodies - in places up to 1.5 m wide. These can be traced sporadically along the length of the fault zone and often are seen to carry free gold.

Several adits are driven into the footwall of the fault zone in the C-area and both detailed mapping and sampling of the adits indicate several zones of mineralization in the footwall granite.

In the Boliden adit three separate zones occur, chip samples giving 7.3 g/t Au / 3 m - this correlates with the main C-vein fault. Further 4.1 g/t Au / 6 m from 7.0 - 13.0 m and finally 3.4 g/t Au / 4 m from 30.0 - 34.0 m.

Values from the other adits on the zone were however poor.

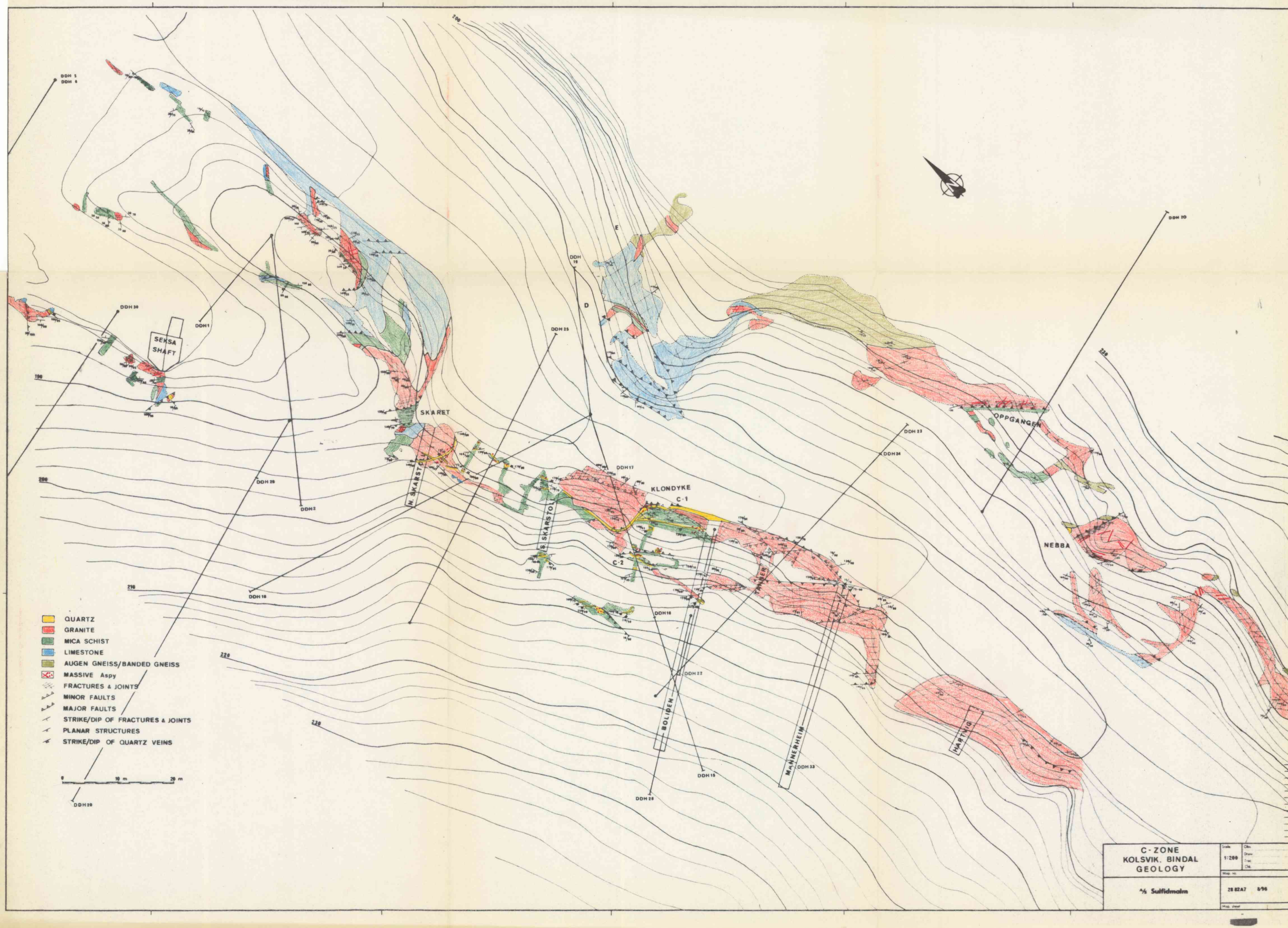
Two different joint sets carrying quartz ± Au and Asp have been mapped in the adits: - a) steep easterly dipping and b) low angle conjugate. The low angled fractures being the earliest.

Thirteen drill holes have been drilled in the C-zone area. DDH 15, 16, 17, 18, 19, 21, 22, 23, 24, 25, 27, 28 and 33.

DDH 15 which was put down to investigate the C-zone at depth intersected a well mineralized zone some 20 - 25 m below the level of the Boliden adit giving 26.1 g/t Au over 11.25 m. In core the mineralization is seen to relate to joints and shears with two sets being developed at right angles to each other.

DDH 16 and 17 put down on the same profile but lower than DDH 15 intersected mineralization over long core lengths (22.0 - 52.0 m in DDH 16; 22.0 - 68.0 m in DDH 17). These meters gave positive indications of gold but gave higher assays only in isolated areas.

DDH 16	34.0 - 36.5 m	2.25 g/t
DDH 17	45.0 - 48.0 m	2.8 g/t
	62.0 - 64.0 m	5.76 g/t



QUARTZ
GRANITE
MICA SCHIST
LIMESTONE
AUGEN GNEISS/BANDED GNEISS
MASSIVE Aspy
FRACTURES & JOINTS
MINOR FAULTS
MAJOR FAULTS
STRIKE/DIP OF FRACTURES & JOINTS
PLANAR STRUCTURES
STRIKE/DIP OF QUARTZ VEINS
0
10 m
20 m
SEKSA SHAFT
SKARET
N. SKARSTOL
S. SKARSTOL
KLONDYKE
C-1
C-2
D
E
OPPGANGEN
NEBBA
BOLIDEN
MANNERHEIM
HARTVIG
DDH 1
DDH 2
DDH 15
DDH 16
DDH 17
DDH 20
DDH 22
DDH 25
DDH 30
DDH 33
C-ZONE
KOLSVIK, BINDAL
GEOLOGY
1:200
A/S Sulfidmalm

The rest of the holes in this area all intersected significant core lengths of mineralized structures, with varying core assays. A summary of the drill holes and significant assay numbers are shown in table 2.

DDH	LOCATION	DIP	LENGTH	SIGNIFICANT ASSAYS FROM	TO	LENGTH	Au g/t
15	62 S 7.5 E	44°	93.45	27.25	38.50	11.25	26.1
16	62 S 7.5 E	65°	89.95	34.0 41.5	36.5 42.0	2.5 0.5	2.25 2.06
17	62 S 7.5 E	80°	80.60	32.0 45.0 62.0	32.5 48.0 64.0	0.5 3.0 2.0	2.06 2.80 5.76
18	62 S 7.5 E	45°	97.0	26.5 34.0	29.0 35.0	2.5 1.0	4.26 2.24
19	62 S 7.5 E	66°	56.3	9.0	9.5	0.5	2.87
20	101.5 S 27 W	90°	156.85	8.0 31.0	9.0 35.0	1.0 4.0	3.48 2.35
22	92 S 2 E	45°	38.0	0.0 22.0 26.0	2.0 24.0 27.0	2.0 2.0 1.0	1.39 1.5 1.74
23	101.5 S 27 W	60°	133.0	49.0 94.0	51.0 105.0	2.0 11.0	2.64 1.28
24	1015 S 27 W	65°	140.7	45.0 91.0 104.0	46.0 92.0 107.0	1.0 1.0 3.0	1.41 2.03 3.09
25	61.5 S 43 W	60°	116.0	29.0 37.0 70.0 (70.0 (71.0 98.0	30.0 38.0 86.0 80.0 74.0 100.0	1.0 1.0 16.0 10.0 3.0 2.0	38.93 3.49 4.86 7.32) 21.65) 2.33
27	92 S 2 E	90°	39.4	0.0 15.0 25.0 31.0	10.0 20.0 27.0 33.0	10.0 5.0 2.0 2.0	1.63 1.7 4.89 1.11
28	92 S 13 W	Core lost in helicopter transport					
33	118 S 5 E	45°	46.1				

2/36

To the north of holes 25 and 18 drilling (DDH 1, 2, 5, 6, 26, 29, 39) has not encountered significant mineralization although on surface chip samples behind the Seksa shaft have given high gold numbers.

The situation in this area is still somewhat unclear and most of the drillholes may have drilled over the continuation of the mineralization.

From Seksa to the B-area some 350 m to the north, no outcrops occur and the area is covered by large amounts of boulder and scree. No holes have been drilled in this area.

On surface in the B-area a quartz arsenopyrite vein has given up to 5 g/t Au over 2 m. Four holes were drilled in section here but gave only a little mineralization.

All drill logs, sections and assays are appendixed to this report.

MINERALOGICAL AND METALLURGICAL EXAMINATIONS

I. Mineralogical investigations

Fourteen drill core samples of various lithologies from the Kolsvik area and four surface samples of mineralization have undergone petrographic examination and qualitative spectrographic analysis. The results are shown in appendix no. 6.

Six hand samples from the "C" and "F" areas have also been examined by R. Buchan for the relationship between gold and arsenopyrite. Two polished sections from each hand sample were prepared and examined using a high magnification objective of the polarizing microscope.

Gold was observed in three of the samples in four habits: as grains completely enclosed in Aspy, as blebs and elongate grains within fractures or shatter cracks in Aspy and as isolated grains in gangue.

Distribution af 68 grains observed in the three samples indicate that over 70 % (by estimated volume) occur enclosed in massive arsenopyrite, about 10 % within fractures in arsenopyrite and 20 % within gangue. Grain sizes range from sub-micron, barely visible specks up to about 15 x 25 µm with an average grain size about 6-7 µm diameter.

The actual grain size distribution of the 68 grains is as follows

Grain size (diameter in µm)	No of grains
<1	7
1-3	27
3-5	18
5-10	9
>10	7

This distribution is in contrast to certain areas of the C zone where very coarse grains occur and average grain size is estimated at about 50 µm diameter.

TABLE 3

NATIVE GOLD DISTRIBUTION IN SAMPLES FROM BINDAL

		ASSOCIATION OF GOLD GRAINS No. of grains (Est. % by volume)			
Sample	No of grains	Enclosed Aspy	Along grain boundaries of Asp	Within cracks in Aspy	In gangue
C 1	25	16 (49 %)	4 (40%)	5 (11 %)	-
C 2	18	13 (17 %)	1 (17%)	1 (33 %)	3 (63 %)
C 3	0	-	-	-	-
F 1	0	-	-	-	-
F 2	25	8 (61 %)	6 (26%)	11 (13 %)	-
F 3	0	-	-	-	-
All samples	68	37 (43 %)	11 (28 %)	17 (9 %)	3 (20 %)

II. Metallurgical investigations

An investigation into the recovery of gold from samples from F- and C-zones has been carried out by Lakefield Research of Canada Lmt. The reports of these investigations are enclosed as appendix 7.

TONNAGE POTENTIAL

The explored part of the area covers the ground from F to Seksa, a distance of 550 m. From the pattern of showings and diamond drill core sections the main tectonized zone is indicated to have minimum depth extension of 230 m (F_1= 340 m - DDH 17 = 110 m.a.s.l.

The criteria used in outlining and limiting the area of potential gold bearing rock are

1) Minor structures such as shears, joints, brecciation, veins and quartz segregation.
2) Mineralization accompanying these minor structures, quartz, arsenopy, py.
3) Frequency of the minor structure as seen in drill core and on showings.
4) Gold assays.

The main tectonic zone thus outlined has been divided into blocks whose dimensions represent the observed mineralization potential criteria in the area. The blocks have then been reduced for topographic effects and a tonnage potential calculated for each block down to the minimum depth extension. The total tonnage of potential gold bearing area thus calculated to be associated with the main tectonic zone is in the range of 2 mill. tons. The area of potential mineralization are shown on summary sections in appendix 4.

SIGNIFICANCE OF RESULTS

From the information available it seems to be well established that a structurally controlled mineralized zone is trending from the F-area to the Seksa area - a distance of 550 m. Both on surface and in drill holes the mineralized zone is seen to have a fairly steep dip to the east and varies in width from narrow 0.5-5 m zones of cm wide veins, compact breccia zones up to 5 m in width and areas composed of several fractures and veins over substantial widths (as in the C-area). The tectonic zone from F - C gives the general impression of pinching and swelling, different minor structures related to the zone having different attitudes and occurrences along the zone.

The criteria which have been used in outlining the structurally controlled mineralized zone (the potential ore zone) are mainly geological, based on information from diamond drilling, surface and adit mapping.

The pattern and trend and frequency of minor structures and accompanying quartz and arsenopyrite within the tectonic zone are the most significant information factors.

In outlining the mineralized zone the gold values are only used as an indicator although positive gold values in most cases support and are co-incident with the geological interpretation.

Based on these criteria a tonnage potential of some 2 mill. tons is indicated.

Examination of the different minor structures show that the gold is irregularly distributed with nuggets and concentrations of smaller grains being common. Sampling of this type of mineralization using diamond drilling and/or chip samples will give an irregular pattern with overrepresentation of low numbers. In spite of this, averaging all the drill core samples in the main tectonic zone returns for the F-zone an average value of 2.09 g/t Au from 131 samples from 8 drill holes. For the C zone the average value of 634 samples was 1.46 g/t from 10 holes and 4 adits.

Sampling carried out by A/S Kolsvik Malmfelter in 1935 - 36 returned fairly good grades both from the C and F areas. The sample size normally brought out was in the range of 80 - 100 kg containing 6 - 12 g/t Au. The irregular and unpredictable gold values returned from samples was also noted by the early workers.

This pattern is also supported by sampling carried out by Sulfidmalm where two 100 kg samples returned 7.77 g/t Au from the F zone and 39.1 g/t Au from the C zone.

The structural/geological interpretation and tonnage potential estimation is based on surface observations and information from drill core. The significance of gold values returned from drill core is difficult to evaluate without taking into consideration the following.

1) The gold bearing minor structures vary both in orientation, attitude and width.

2) These minor structures also vary in intensity and distribution.

3) The internal gold distribution within the minor structures is irregular with the occurrence of nuggets or grain concentrations.

Given the very strong nugget effect and irregular distribution both of gold and gold bearing structures any grade evaluation based on core samples and chip samples will be highly uncertain.

The effect of nuggets on sampling and sample size are well demonstrated in the following models:

A) Using an ideal model with one m^3 of rock (2.5 t) containing an even distribution of equal sized gold grains totalling 12.5 g. This gives an average of 5 g/t Au.
The core sample used in assaying has a weight of 2.5 kg, in other words 1 m^3 consists of 1000 core samples.
We can consider 3 cases where the 12.5 g is divided among 1) 10 grains 2) 100 grains and 3) 1000 grains. In these cases the probability of getting 1 grain in core sample and the resulting ppm value in the sample is as follows:

	1	2	3
Grains Au	10	100	1000
Probability of one grain in core sample	1/100	1/10	1
ppm Au in sample	500	50	5

B) A model which tries to take into consideration the situation at Kolsvik with the nugget effect and the irregular distribution and concentration of smaller grains will be as follows.

In this case 1 m^3 contains 10.5 g/Au giving 4.2 g/t. Again one core sample is 2.5 kg giving 1000 samples/m^3.

Number of samples	5	10	10	25	50	100	100	200	500
g Au in each sample	0.5	0.25	0.1	0.05	0.025	0.01	0.005	0.001	0.0005
Probability of positive assay in core	1/200	1/100	1/100	1/40	1/20	1/10	1/10	1/5	1/2
ppm Au in sample	200	100	40	20	10	4	2	0.4	0.02

Also to be taken into consideration are mistakes introduced by core splitting and sample reducing prior to assaying.

Model B shows that the possibility for getting a low value in core sampling is statistically much higher than for getting an high or even average number.

Despite this the average value of all core samples in the "potential zone" return approx. 2 g/t Au.

Based on the models presented above one can argue that a true average grade should be at least 2 or 3 times higher than this. Attention should also be given to the two larger samples that have been taken from F and C, both of which returned high values.

CONCLUSIONS AND RECOMMENDATIONS

From the information available a tectonic mineralized gold bearing zone extends from the F-area to Seksa - a distance of some 500 m. Drilling has indicated a depth extension on the zone of 200 m.

The geometry of the mineralized sone varies and the distribution of mineralization varies. A tonnage potential of 2 million tons is indicated.

An accurate determination of the grade of the deposit is not possible based on the available information, but arguments can be presented that indicate the possibilities of an economic grade being present.

It is recommended that the results to date warrant more work and that a program of bulk sampling in the 5 - 10.000 ton range be carried out in order to evaluate an average grade that can be related to a given tonnage.

HOLE	CO-ORDINATES [1]	BEARING	DIP	LENGTH [2]	ASSAYS ppm Au							
					FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au
1.	0 – 0	274°	80°	117.80 m	13.5	16.0	4.5	<0.5				
					28.0	29.0	1.0	<0.5				
					45.0	45.75	0.75	<0.4				
					45.75	46.0	0.25	6.7				
					46.0	47.0	1.0	<0.5				
					50.0	52.0	2.0	<0.5				
					56.7	60.0	3.3	<0.5				
					60.0	60.25	0.25	0.8				
					60.25	61.25	1.0	<0.4				
					61.25	61.50	0.25	18				
					61.50	64.75	3.25	<0.4				
					66.25	69.20	3.05	<0.5				
					94.0	95.0	1.0	<0.5				
					112.0	112.5	0.5	<0.5				
2.	0 – 0	227°	55°	85.30	5.25	6.0	0.75	<0.4				
					13.0	18.0	5.0	<0.5				
					36.0	37.0	1.0	<0.5				
					37.0	37.3	0.3	1.9				
					44.0	48.0	4.0	<0.5				
3.	352 S 158 E		90°	94.20	8.0	12.0	4.0	<0.7				
					18.0	19.0	1.0	<0.6				
					20.0	22.5	2.5	<0.6				
					22.5	22.75	0.25	2.7				
					22.75	23.0	0.25	2.2				
					23.0	30.0	7.0	<0.8				
					34.0	38.25	4.25	<0.4				
					38.25	38.50	0.25	1.6				
					38.50	38.75	0.25	1.2				
					38.75	41.0	2.25	<0.6				

[1] Reference point 0|0 = Skaret

[2] All lenghts in meters

HOLE	CO-ORDINATES[1]	BEARING	DIP	LENGTH[2]	ASSAYS ppm Au							
					FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au
3	325 S 158 E		90°	94.20 m	53.0	57.0	4.0	<0.4				
					58.0	59.0	1.0	<0.4				
					60.0	60.25	0.25	5.7				
					60.25	60.50	0.25	4.7	60.0	61.0	1.0	3.3
					60.50	60.75	0.25	1.8				
					60.75	61.0	0.25	1.0				
					61.0	61.5	0.50	<0.6				
					61.5	61.75	0.25	1.4				
					61.75	62.0	0.25	<0.4				
					62.0	62.25	0.25	1.3				
					62.25	62.50	0.25	2.8	62.0	62.5	0.5	2.05
					62.50	65.25	2.75	<0.9				
					65.25	65.50	0.25	10.6				
					65.50	65.75	0.25	3.9				
					65.75	66.0	0.25	2.8	65.25	66.50	1.25	4.88
					66.0	66.25	0.25	5.5				
					66.25	66.5	0.25	1.6				
					66.5	67.25	0.75	<0.6				
					67.25	67.5	0.25	5.3				
					67.5	67.75	0.25	0.6				
					67.75	68.0	0.25	<0.4				
					68.0	68.25	0.25	1.7				
					68.25	68.5	0.25	1.0				
					68.5	68.75	0.25	3.4				
					68.75	70.0	1.25	<0.8				
					70.0	70.25	0.25	1.0				
					70.25	76.25	6.0	<0.4				
					76.25	76.50	0.25	1.0				
					76.50	77.25	0.75	<0.6				
					77.25	77.50	0.25	1.7				
					77.50	79.50	2.0	<0.8				
					79.50	79.75	0.25	15				
					79.75	80.0	0.25	15	79.50	80.0	0.5	15
					80.0	87.50	7.50	<0.4				

[1] Reference point 0|0 = Skaret

[2] All lenghts in meters

HOLE	CO-ORDINATES[1]	BEARING	DIP	LENGTH[2]	ASSAYS ppm Au FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au	
3	352 S 158 E				87.50	87.75	0.25	3.3					
					87.75	88.0	0.25	2.0					
					88.0	88.25	0.25	37					
					88.25	88.50	0.25	1.9					
					88.50	88.75	0.25	3.5					
					88.75	89.00	0.25	6.7	87.50	90.75	3.25	9.31	
					89.0	89.25	0.25	9.6					
					89.50	90.0	0.50	<0.5					
					90.0	90.25	0.25	9.0					
					90.25	90.5	0.25	13.9					
					90.5	90.75	0.25	14.8					
					90.75	93.0	2.25	<0.6					
4	352 S 158 E	226°	50°	93.05 m	13.75	14.75	1.0	<0.6					
					14.75	15.0	0.25	1.0					
					15.0	17.0	2.0	<0.6					
					17.0	17.25	0.25	1.0					
					17.25	17.50	0.25	7.0	17.0	18.0	1.0	4.05	2.42.
					17.5	17.75	0.25	7.4					
					17.75	18.0	0.25	0.8					
					18.0	18.25	0.25	<0.3					
					18.25	18.50	0.25	1.9					
					18.50	19.75	1.25	<0.5					
					21.25	28.75	7.5	<0.5					
					28.75	29.0	0.25	2.9					
					29.0	29.25	0.25	63	28.75	29.5	0.75	22.3	[illegible]
					29.25	29.5	0.25	1.2					
					29.5	33.0	3.50	<0.5					
5.	48 N 1 E	082°	45°	122.0 m									
6.	48 N 1 E	082°	65°	92.0 m									
7.	ABANDONED IN OVERBURDEN AT 22 m.												

[1] Reference point 0|0 = Skaret

[2] All lengths in meters

HOLE	CO-ORDINATES[1]	BEARING	DIP	LENGTH[2]	ASSAYS ppm Au FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au	
8.	285 S 83 E	060°	40°	88.30 m	40.0	50.0	10.0	Nil					
					51.0	51.25	0.25	<0.01					
					54.0	55.5	1.5	<0.4					
					55.5	55.75	0.25	2.5					
					55.75	56.0	0.25	1.4	55.5	56.25	0.75	5.63	
					56.0	56.25	0.25	13					
					56.25	58.50	2.25	<0.6					
					58.50	58.75	0.25	3.6					
					58.75	59.0	0.25	28					441/4.75m
					59.0	59.25	0.25	4.7					
					59.25	59.50	0.25	8.8	58.50	60.25	1.75	7.82	
					59.50	59.75	0.25	6.1					
					59.75	60.0	0.25	1.4					
					60.0	60.25	0.25	2.2					
					60.25	61.75	0.50	<0.4					
					61.75	62.25	0.50	1.03					
					62.25	79.0	17.75	<0.3					
9.	373 S 113 E	052°	34.9°	94.6 m	40.25	43.0	2.75	<0.6					
					45.0	45.75	0.75	<0.1					
					48.0	63.75	15.75	<0.5					
					63.75	64.0	0.25	2.4					
					64.0	68.0	4.0	<0.2					
					68.0	68.25	0.25	1.1					Tom
					68.25	80.0	11.75	<0.2					
					80.0	80.25	0.25	3.43					
					80.25	80.50	0.25	–					
					80.5	80.75	0.25	1.03					
					80.75	84.0	3.25	<0.2					
10.	362 S 101 E	062°	36°	144. m	40.0	43.0	3.0	0.1					
					43.0	50.0	7.0	0.03					
					50.0	54.0	4.0	<0.1					
					54.0	55.0	1.0	9.29					
					55.0	56.0	1.0	9.29					
					56.0	57.0	1.0	2.75	54.0	59.0	5.0	4.88	
					57.0	58.0	1.0	2.06					
					58.0	59.0	1.0	1.03					
					59.0	70.0	11.0	<0.3					

[1] Reference point 0|0 = Skaret

[2] All lenghts in meters

KOLSVIK, BINDALEN, DIAMOND DRILL RECORD

HOLE	CO-ORDINATES[1]	BEARING	DIP	LENGTH[2]	ASSAYS ppm Au FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au				
10.	362 S 101 E	062°	36°	144.0 m	70.0	72.0	2.0	0.06								
					72.0	74.0	2.0	Nil								
					74.0	79.0	5.0	0.05								
					79.0	80.0	1.0	Nil								
					80.0	88.0	8.0	0.2								
11.	362 S 101 E	062°	55°	159.3 m	83.0	94.0	11.0	Nil								
					94.0	103.0	9.0	0.7								
					103.0	103.5	0.5	1.03								
					103.5	114.0	10.5	0.7								
					114.0	114.5	0.5	1.38								
					114.5	115.0	0.5	1.72								
					115.0	115.5	0.5	1.72								
					115.5	116.0	0.5	0.17								
					116.0	116.5	0.5	0.34								
					116.5	117.0	0.5	4.47					114.0	119.0	5.0	3.38
					117.0	117.5	0.5	16.86	116.5	119.0	2.5	5.69				
					117.5	118.0	0.5	3.10								
					118.0	119.0	1.0	2.0								
					119.0	120.0	1.0	<0.5								
					120.0	121.5	1.5	0.1								
					144.0	149.0	5.0	<0.3								
12.	201 S 50 E	072°	38°	124.50	30.0	38.0	8.0	Nil								
					38.0	39.0	1.0	2.56								
					39.0	40.0	1.0	0.13								
					40.0	40.5	0.5	28.8					38.0	41.5	3.5	5.22
					40.5	41.0	0.5	1.4	40.0	41.5	1.5	10.4				
					41.0	41.5	0.5	1.0								
					41.5	45.0	3.5	0.4								
					45.0	65.0	20.0	Nil								
					66.0	75.0	9.0	<0.3								
					84.0	103.0	19.0	0.4								

[1] Reference point 0|0 = Skaret

[2] All lengthsin meters

HOLE	CO-ORDINATES[1]	BEARING	DIP	LENGTH[2]	ASSAYS ppm Au FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au	
13.	201 S 50 E	072°	20°	63.7 m	10.0	27.0	17.0	Nil					
					27.0	30.0	3.0	<0.4					
					30.0	30.5	0.5	19					
					30.5	31.0	0.5	<0.4					
					31.0	31.5	0.5	25	30.0	33.0	3.0	8.06	
					31.5	32.0	0.5	1.2					
					32.0	32.5	0.5	<0.3					
					32.5	33.0	0.5	3.2					
					33.0	40.5	7.5	<0.4					
					40.5	41.0	0.5	2.7					
					41.0	41.5	0.5	8.92	40.5	41.5	1.0	5.8	
					41.5	50.0	8.5	Nil					
14.	420 S 168 E	075°	42°	120.8 m									NOT ASSAYED
15.	62 S 75 E	215°	44°	93.45	18.5	20.0	1.5	Nil					
					20.0	22.0	2.0	<0.3					
					22.0	24.0	2.0	<0.5					
					24.0	24.25	0.25	4.6					
					24.25	24.50	0.25	0.5					
					24.50	25.50	1.0	<0.5					
					25.50	25.75	0.25	1.1					
					25.75	26.0	0.25	0.3					
					26.0	26.25	0.25	1.7					
					26.25	26.50	0.25	<0.3					
					26.50	26.75	0.25	0.6					
					26.75	27.0	0.25	<0.3					
					27.0	27.25	0.25	<0.3					
					27.25	27.50	0.25	7.8					
					27.50	27.75	0.25	55					
					27.75	28.0	0.25	5.9					
					28.0	28.25	0.25	2.1					
					28.25	28.50	0.25	35					

[1] Reference point 0|0 = Skaret

[2] All lengths in meters

HOLE	CO-ORDINATES	BEARING	DIP	LENGTH[1]	ASSAYS ppm Au FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au	
15.	62 S 7,5 E				28.50	28.75	0.25	40					
					28.75	29.0	0.25	28					
					29.0	29.25	0.25	5.6					
					29.25	29.50	0.25	17					
					29.50	29.75	0.25	11					
					29.75	30.0	0.25	8.2					
					30.0	30,25	0.25	0.7					
					30.25	30.5	0.25	6.4					
					30.50	30.75	0.25	1.6					
					30.75	31.0	0.25	2.5	27.25	38.50	11.25	26.1	10
					31.0	31.25	0.25	0.5					
					31.25	31.50	0.25	0.3					
					31.30	31.75	0.25	‹ 0.4					
					31.75	32.0	0.25	2.5					
					32.0	32.25	0.25	5.7					
					32.25	32.50	0.25	‹ 0.3					
					32.50	32.75	0.25	‹ 0.2					
					32.75	33.0	0.25	21					
					33.0	33.25	0.25	‹ 0.3					
					33.25	33.50	0.25	·0.3					
					33.50	33.75	0.25	‹ 0.3					
					33.75	34.0	0.25	5.9					
					34.0	34.25	0.25	3.3					
					34.25	34.50	0.25	0.2					
					34.50	34.75	0.25	5.5					
					34.75	35.0	0.25	11					
					35.0	35.25	0.25	777 - 30					
					35.25	35.50	0.25	2.0					
					35.50	35.75	0.25	0.9					
					35.75	36.0	0.25	6.0					
					36.0	36.25	0.25	0.5					
					36.25	36.50	0.25	0.4					
					36.50	36.75	0.25	0.6					
					36.75	37.0	0.25	33					
					37.0	37.25	0.25	0.8					
					37.25	37.50	0.25	2.7					
					37.50	37.75	0.25	0.4					
					37.75	38.0	0.25	1.2					
					38.0	38.25	0.25	6.4					
					38.25	38.50	0.25	4.9					
					38.50	39.0	0.5	‹0.3					

[1] All lenghtsin meters

HOLE	CO-ORDINATES[1]	BEARING	DIP	LENGTH[2]	ASSAYS ppm Au							
					FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au
15.	62 S 75 E				39.0	39.5	0.5	1.0				
					39.5	40.0	0.5	2.2				
					40.0	43.0	3.0	<0.3				
					43.0	60.0	17.0	Nil				
16.	62 S 75 E	215°	65°	89.95 m	19.5	22.0	2.5	Nil				
					22.0	23.0	1.0	0.13				
					23.0	23.25	0.25	<0.3				
					23.25	23.50	0.25	1.2				
					23.50	32.0	8.5	<0.3				
					32.0	32.50	0.5	1.4				
					32.50	34.0	1.5	<0.3				
					34.0	34.5	0.5	5.1				
					34.5	35.0	0.5	2.9				
					35.0	35.5	0.5	1.72	34.0	36.5	2.5	2.25
					35.5	36.0	0.5	0.17				
					36.0	36.5	0.5	1.37				
					36.5	37.0	0.5	0.17				
					37.0	37.50	0.5	0.17				
					37.50	38.0	0.5	0.07				
					38.0	38.5	0.5	1.37				
					38.5	41.5	3.0	<0.4				
					41.5	42.0	0.5	2.06				
					42.0	43.5	1.5	<0.2				
					43.5	44.0	0.5	1.72				
					44.0	51.0	7.0	<0.7				
					51.0	52.0	1.0	0.06				
					52.0	53.0	1.0	1.63				
					53.0	55.0	2.0	Nil				
					55.0	56.0	1.0	0.40				
					56.0	63.0	7.0	Nil				
					63.0	64.0	1.0	1.26				
					64.0	65.0	1.0	0.09				
					65.0	70.0	5.0	Nil				

[1] Reference point 0|0 = Skaret

[2] All lengths in meters

HOLE	CO-ORDINATES[1]	BEARING	DIP	LENGTH	ASSAYS ppm Au							
					FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au
17.	62 S 75 E	215°	80°	80.60	20.0	23.0	3.0	Nil				
					23.0	24.0	1.0	0.08				
					24.0	24.5	0.5	0.16				
					24.5	32.0	7.5	0.2				
					32.0	32.5	0.5	2.06				
					32.5	34.5	2.0	0.2				
					34.5	35.0	0.5	0.01				
					35.0	36.0	1.0	1.15				
					36.0	45.0	9.0	0.09				
					45.0	46.0	1.0	0.83				
					46.0	47.0	1.0	0.32	45.0	48.0	3.0	2.80
					47.0	48.0	1.0	7.26				
					48.0	50.0	2.0	0.5				
					50.0	51.0	1.0	0.14				
					51.0	52.0	1.0	0.03				
					52.0	53.0	1.0	0.45				
					53.0	54.0	1.0	Nil				
					54.0	55.0	1.0	0.37				
					55.0	56.0	1.0	0.04				
					56.0	57.0	1.0	0.06				
					57.0	58.0	1.0	0.11				
					58.0	59.0	1.0	0.32				
					59.0	60.0	1.0	0.24				
					60.0	61.0	1.0	0.06				
					61.0	62.0	1.0	Nil				
					62.0	63.0	1.0	2.88	62.0	64.0	2.0	5.76
					63.0	64.0	1.0	8.64				
					64.0	65.0	1.0	0.01				
					65.0	66.0	1.0	Nil				
					66.0	67.0	1.0	0.01				
					67.0	68.0	1.0	0.04				
					68.0	69.0	1.0	0.21				
					69.0	75.0	6.0	Nil				

[1] Reference point 0|0 = Skaret
All lengths in meters

HOLE	CO-ORDINATES[1]	BEARING	DIP	LENGTH[2]	ASSAYS ppm Au							
					FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au
18.	62 S 7,5 E	295°	45°	97.0 m	16.6	22.0	5.4	<0.6				
					26.5	27.0	0.5	1.26				
					27.0	27.50	0.5	6.84				
					27.5	28.0	0.5	4.14	26.5	29.0	2.5	4.26
					28.0	28.5	0.5	7.86				
					28.5	29.0	0.5	1.20				
					29.0	32.5	3.5	<0.9				
					32.5	33.0	0.5	3.25				
					33.0	34.0	1.0	<0.2				
					34.0	34.5	0.5	1.96	34.0	35.0	1.0	2.24
					34.5	35.0	0.5	2.53				
					35.0	38.0	3.0	<0.1				
19.	62 S 7,5 E	048°	66°	56.3 m	7.5	9.0	1.5	<0.2				
					9.0	9.5	0.5	2.87				
					9.5	15.0	5.5	<0.8				
20.	130 S 30 E	087°	45°	89.8 m	15.0	17.5	2.5	<0.2				
					17.5	18.0	0.5	1.53				
					18.0	22.0	4.0	<0.1				
					24.0	26.0	2.0	<0.1				
					34.0	38.0	4.0	<0.1				
					47.0	67.0	20.0	<0.4				
21.	101,5 S 27 W		90°	156.85 m	2.0	8.0	6.0	Nil				
					8.0	9.0	1.0	3.48				
					9.0	10.0	1.0	0.02				
					10.0	11.0	1.0	0.29				
					11.0	12.0	1.0	0.01				
					12.0	13.0	1.0	0.05				
					13.0	17.0	4.0	Nil				
					17.0	18.0	1.0	0.21				
					18.0	19.0	1.0	0.07				
					19.0	20.0	1.0	0.03				

[1] Reference point 0|0 = Skaret

[2] All lengths in meters

HOLE	CO-ORDINATES[1]	BEARING	DIP	LENGTH[2]	ASSAYS ppm Au							
					FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au
21.	1015 S 27 W		90°		26.0	27.0	1.0	Nil				
					27.0	28.0	1.0	0.63				
					28.0	29.0	1.0	0.01				
					29.0	30.0	1.0	Nil				
					30.0	31.0	1.0	0.07				
					31.0	32.0	1.0	0.14				
					32.0	33.0	1.0	8.66	31.0	35.0	4.0	2.35
					33.0	34.0	1.0	0.11				
					34.0	35.0	1.0	0.51				
					35.0	36.0	1.0	0.01				
					36.0	37.0	1.0	0.03				
					37.0	38.0	1.0	0.10				
					38.0	39.0	1.0	0.01				
					39.0	40.0	1.0	0.02				
					40.0	41.0	1.0	0.27				
					41.0	45.0	4.0	Nil				
					45.0	46.0	1.0	0.47				
					46.0	48.0	2.0	Nil				
					61.0	62.0	1.0	Nil				
					69.0	70.0	1.0	Nil				
					72.0	73.0	1.0	Nil				
					78.0	80.0	2.0	0.03				
					81.0	82.0	1.0	Nil				
					88.0	89.0	1.0	Nil				
					105.0	110.0	5.0	Nil				
					135.0	145.0	10.0	Nil				
22.	92 S 2 E	247°	45°	38 m	0.0	1.0	1.0	1.04				
					1.0	2.0	1.0	1.74				
					2.0	3.0	1.0	0.03				
					3.0	4.0	1.0	0.41				
					4.0	5.0	1.0	0.04				
					5.0	6.0	1.0	0.05				
					6.0	7.0	1.0	0.01				
					8.0	9.0	1.0	0.48				
					9.0	10.0	1.0	0.08				
					10.0	11.0	1.0	Nil				

[1] Reference point 0|0 = Skaret

[2] All lengthsin meters

HOLE	CO-ORDINATES[1]	BEARING	DIP	LENGTH[2]	ASSAYS ppm Au							
					FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au
22.	92 S 2 E	247°	45°	38 m	11.0	12.0	1.0	0.76				
					12.0	13.0	1.0	0.19				
					13.0	14.0	1.0	Nil				
					14.0	15.0	1.0	0.05				
					15.0	16.0	1.0	1.91				
					16.0	17.0	1.0	0.02				
					17.0	18.0	1.0	0.40				
					18.0	19.0	1.0	Nil				
					19.0	20.0	1.0	1.86				
					20.0	21.0	1.0	0.02				
					21.0	22.0	1.0	0.27				
					22.0	23.0	1.0	2.33				
					23.0	24.0	1.0	0.67				
					24.0	26.0	2.0	Nil				
					26.0	27.0	1.0	1.74				
					27.0	28.0	1.0	0.06				
					28.0	31.0	3.0	Nil				
					31.0	32.0	1.0	0.02				
					32.0	33.0	1.0	0.11				
					33.0	34.0	1.0	0.15				
					34.0	35.0	1.0	0.14				
					35.0	36.0	1.0	0.72				
					36.0	37.0	1.0	0.16				
					37.0	38.0	1.0	0.01				
23.	1015 S 27 W	095°	60°	133 m	40.0	41.0	1.0	0.49				
					41.0	42.0	1.0	0.01				
					42.0	43.0	1.0	0.08				
					43.0	44.0	1.0	Nil				
					44.0	45.0	1.0	0.05				
					45.0	46.0	1.0	0.37				
					46.0	47.0	1.0	0.01				
					47.0	48.0	1.0	Nil				
					48.0	49.0	1.0	0.03				
					49.0	50.0	1.0	1.19	49.0	51.0	2.0	2.64
					50.0	51.0	1.0	4.09				

[1] Reference point 0|0 = Skaret

[2] All lengths in meters

HOLE	CO-ORDINATES[1]	BEARING	DIP	LENGTH[2]	ASSAYS ppm Au FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au
23.	1015 S 27 W	095°	60°	133 m	51.0	52.0	1.0	0.04				
					52.0	53.0	1.0	0.95				
					53.0	54.0	1.0	0.05				
					54.0	55.0	1.0	1.45				
					55.0	56.0	1.0	0.42				
					56.0	57.0	1.0	0.03				
					57.0	58.0	1.0	0.07				
					58.0	59.0	1.0	0.05				
					59.0	70.0	11.0	0.02				
					70.0	71.0	1.0	0.96				
					71.0	75.0	4.0	0.02				
					75.0	76.0	2.0	0.76				
					76.0	83.0	7.0	0.06				
					83.0	85.0	2.0	0.16				
					85.0	92.0	7.0	0.06				
					92.0	94.0	2.0	0.16				
					94.0	95.0	1.0	0.51				
					95.0	96.0	1.0	2.70				
					96.0	97.0	1.0	1.30				
					97.0	98.0	1.0	1.98	94.0	105.0	11.0	1.28
					98.0	99.0	1.0	0.24				
					99.0	100.0	1.0	2.12				
					100.0	101.0	1.0	0.97				
					101.0	102.0	1.0	0.40				
					102.0	103.0	1.0	1.67				
					103.0	104.0	1.0	0.36				
					104.0	105.0	1.0	0.75				
					105.0	110.0	5.0	0.08				
24.	1015 S 27 W	095°	65°	140.7	35.0	45.0	10.0	Nil				
					45.0	46.0	1.0	1.41				
					46.0	50.0	4.0	0.04				
					90.0	91.0	1.0	0.14				
					91.0	92.0	1.0	2.03				
					92.0	100.0	8.0	Nil				
					100.0	101.0	1.0	0.06				

[1] Reference point 0|0 = Skaret

[2] All lengths in meters

HOLE	CO-ORDINATES[1]	BEARING	DIP	LENGTH[2]	ASSAYS ppm Au FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au	
24.	1015 S 27 W				101.0	102.0	1.0	0.34					
					102.0	103.0	1.0	0.03					
					103.0	104.0	1.0	0.21					
					104.0	105.0	1.0	3.23					
					105.0	106.0	1.0	3.16	104.0	107.0	3.0	3.09	
					106.0	107.0	1.0	2.89					
					107.0	110.0	3.0	0.06					
					120.0	121.0	1.0	0.43					
					121.0	128.0	7.0	0.06					
25.	615 S 43 W		60°	116.0 m	25.0	29.0	4.0	Nil					
					29.0	30.0	1.0	38.93					
					30.0	31.0	1.0	0.09					
					31.0	32.0	1.0	0.24					
					32.0	37.0	5.0	0.05					
					37.0	38.0	1.0	3.49					
					38.0	39.0	1.0	0.66					
					39.0	44.0	5.0	0.06					
					44.0	45.0	1.0	0.36					
					50.0	66.0	16.0	Nil					
					66.0	70.0	4.0	0.04					
					70.0	71.0	1.0	0.60					
					71.0	72.0	1.0	4.09					
					72.0	73.0	1.0	41.64					
					73.0	74.0	1.0	19.21					
					74.0	75.0	1.0	0.57					
					75.0	76.0	1.0	2.0					
					76.0	77.0	1.0	0.52	70.0	86.0	16.0	4.86	70.0 - 80.0 10.0 7.32
					77.0	78.0	1.0	0.22					
					78.0	79.0	1.0	0.75					
					79.0	80.0	1.0	3.61					
					80.0	81.0	1.0	0.24					
					81.0	82.0	1.0	0.72					
					82.0	83.0	1.0	0.33					
					83.0	84.0	1.0	0.32					
					84.0	85.0	1.0	0.62					
					85.0	86.0	1.0	2.40					

Mde galt

[1] Reference point 0|0 = Skaret

[2] All lengthsin meters

HOLE	CO-ORDINATES	BEARING	DIP	LENGTH	ASSAYS ppm Au							
					FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au
25.	615 S 43 W				86.0	90.0	4.0	0.10				
					90.0	93.0	3.0	0.41				
					93.0	98.0	5.0	0.18				
					98.0	99.0	1.0	3.57				
					99.0	100.0	1.0	1.10	98.0	100.0	2.0	2.33
26.	23 S 25 W		31°	91.10 m	13.0	14.0	1.0	0.18				
					14.0	16.0	2.0	0.02				
					24.0	40.0	16.0	Nil				
					75.0	91.0	16.0	Nil				
27.	92 S 2 E		90°	39.4	0.0	1.0	1.0	4.6				
					1.0	2.0	1.0	1.8				
					2.0	3.0	1.0	1.0				
					3.0	4.0	1.0	0.96				
					4.0	5.0	1.0	0.35				
					5.0	6.0	1.0	0.44	0.0	10.0	10.0	1.63
					6.0	7.0	1.0	3.55				
					7.0	8.0	1.0	0.47				
					8.0	9.0	1.0	0.39				
					9.0	10.0	1.0	2.78				
					10.0	11.0	1.0	0.16				
					11.0	12.0	1.0	0.03				
					12.0	13.0	1.0	0.10				
					13.0	14.0	1.0	0.16				
					14.0	15.0	1.0	0.23				
					15.0	16.0	1.0	0.90				
					16.0	17.0	1.0	2.02				
					17.0	18.0	1.0	2.29	15.0	20.0	5.0	1.7
					18.0	19.0	1.0	0.35				
					19.0	20.0	1.0	2.94				
					20.0	21.0	1.0	Nil				
					21.0	22.0	1.0	0.03				
					22.0	23.0	1.0	0.64				
					23.0	24.0	1.0	0.22				
					24.0	25.0	1.0	0.03				
					25.0	26.0	1.0	9.26				
					26.0	27.0	1.0	0.52	25.0	27.0	2.0	4.89
					27.0	28.0	1.0	0.18				

Reference point 0|0 = Skaret
All lengths in meters

HOLE	CO-ORDINATES[1]	BEARING	DIP	LENGTH[2]	ASSAYS ppm Au FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au	
27.	92 S 2 E				28.0	31.0	3.0	0.03					
					31.0	32.0	1.0	1.55					
					32.0	33.0	1.0	0.68					
					33.0	34.0	1.0	0.04					
					34.0	35.0	1.0	0.10					
					35.0	36.0	1.0	0.31					
					36.0	37.0	1.0	0.66					
					37.0	38.0	1.0	0.40					
					38.0	39.0	1.0	0.27					
					39.0	40.0	1.0	0.04					
28	92 S 13 W	Hole lost in helicopter transport											
29.	23 S 25 W		75°	45.20 m	9.0	11.0	2.0	0.06					
					11.0	12.0	1.0	0.68					
					12.0	24.0	2.0	Nil					
30	165 N 28 W	260°	45°	77.1									NOT ASSAYED
31.	341 N 53 W	270°	80°	49.15	1.5	7.0	5.5	0.02					
					7.0	8.0	1.0	1.17					
					8.0	9.0	1.0	0.14					
					9.0	10.0	1.0	0.23					
					10.0	12.0	2.0	Nil					
					12.0	14.0	2.0	0.05					
					14.0	15.0	1.0	Nil					
					15.0	18.0	3.0	0.04					
					18.0	20.0	2.0	Nil					
32.	340 N 48 W	90°	65°	95.6									NOT ASSAYED

[1] Reference point 0|0 = Skaret

[2] All lengths in meters

KOLSVIK, BINDALEN. DIAMOND DRILL RECORD.

HOLE	CO-ORDINATES[1]	BEARING	DIP	LENGTH[2]	ASSAYS ppm Au							
					FROM	TO	LENGTH	Au	FROM	TO	LENGTH	Au
33.	118 S 5 E	256°	45°	46.1	1.0	2.0	1.0	Nil				
					2.0	7.0	5.0	0.18				
					7.0	9.0	2.0	Nil				
					9.0	15.0	6.0	0.27				
					15.0	17.0	2.0	Nil				
					17.0	25.0	8.0	0.08				
					25.0	27.0	2.0	Nil				
					27.0	37.0	10.0	0.03				
					37.0	44.0	7.0	Nil				
					44.0	46.0	2.0	0.02				
34.	340 N 48 W		90°	73.1	15.0	21.0	6.0	Nil				
					21.0	30.0	9.0	0.16				
35.	341 N 53 W	90°	65°	95.6								
36.	300 N 42 E		36°	271.5	52.0	53.0	1.0	Nil				
					53.0	56.0	3.0	0.12				
					56.0	58.0	2.0	Nil				
					115.0	117.0	2.0	Nil				
					117.0	122.0	5.0	0.02				
					122.0	123.0	1.0	0.28				
					123.0	124.0	1.0	0.83				
					124.0	125.0	1.0	0.04				
					210.0	218.0	8.0	0.02				
					219.0	240.0	21.0	Nil				

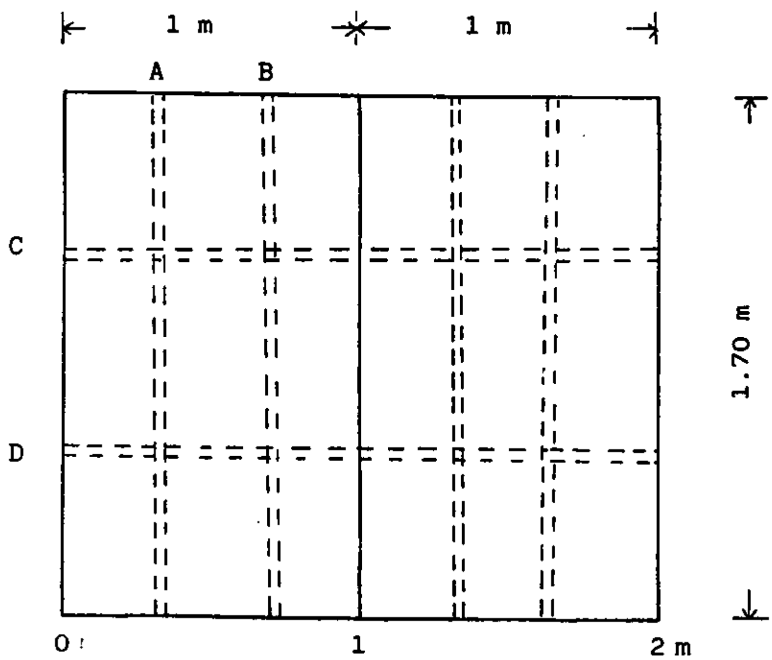
[1] Reference point 0|0 = Skaret

[2] All lengths in meters

APPENDIX

Channel samples from adits in the Kolsvik area.

The adits were sampled on one wall over 1 m intervals. 4 channel samples being taken over 1.70 m^2 as shown below.



BOLIDEN ADIT Sampled on South Wall

Location	Sample no.		Gold g/t	Average gold g/t
÷1 – 0 m	3242	A	1.64	
		B	22.94	7.23
		C	0.58	
		D	3.78	
0 – 1 m	3201	A	16.29	
		B	2.62	10.89
		C	16.49	
		D	8.18	
1 – 2 m	3202	A	3.09	
		B	1.56	3.87
		C	7.41	
		D	3.43	
2 – 3 m	3203	A	0.03	
		B	0.42	0.27
		C	0.14	
		D	0.49	
3 – 4 m	3204	A	1.04	
		B	0.08	0.37
		C	0.25	
		D	0.10	
4 – 5 m	3205	A	1.25	
		B	2.68	1.48
		C	0.51	
		D	Sample missing	
5 – 6 m	3206	A	1.27	
		B	0.98	0.65
		C	0.21	
		D	0.16	
6 – 7 m	3207	A	0.39	
		B	0.65	0.82
		C	1.67	
		D	0.56	

Boliden adit, sampled on south wall.

Location	Sample no.		Gold g/t	Average gold g/t
7 - 8 m	3208	A	0.45	
		B	1.18	1.16
		C	0.28	
		D	2.74	
8 - 9 m	3209	A	0.52	
		B	2.65	4.72
		C	0.62	
		D	15.09	
9 - 10 m	3210	A	1.14	
		B	0.18	1.56
		C	4.62	
		D	0.30	
10 - 11 m	3211	A	1.06	
		B	5.37	2.42
		C	2.84	
		D	0.41	
11 - 12 m	3212	A	1.06	
		B	7.58	11.15
		C	35.22	
		D	0.76	
12 - 13 m	3213	A	4.39	
		B	1.63	3.41
		C	7.43	
		D	0.21	
13 - 14 m	3214	A	0.99	
		B	0.41	0.74
		C	1.51	
		D	0.07	

Boliden adit, sampled on south wall.

Location	Sample no.		Gold g/t	Average gold g/t
14 - 15 m	3215	A	0.65	
		B	0.47	0.35
		C	0.07	
		D	0.21	
15 - 16 m	3216	A	Nil	
		B	0.03	0.04
		C	Nil	
		D	0.14	
16 - 17 m	3217	A	0.05	
		B	0.07	0.06
		C	0.01	
		D	0.12	
17 - 18 m	3218	A	0.03	
		B	0.10	0.05
		C	0.02	
		D	0.07	
18 - 19 m	3219	A	0.07	
		B	0.08	0.56
		C	2.06	
		D	0.10	
19 - 20 m	3220	A	0.06	
		B	0.07	0.38
		C	1.13	
		D	0.28	
20 - 21 m	3221	A	0.04	
		B	0.06	0.04
		C	0.03	
		D	0.28	

Boliden adit, sampled on south wall

Location	Sample no.		gold g/t	average gold g/t
21 - 22 m	3222	A	0.04	
		B	0.03	0.10
		C	0.14	
		D	0.20	
22 - 23 m	3223	A	0.03	
		B	0.03	0.12
		C	0.22	
		D	0.21	
23 - 24 m	3224	A	0.23	
		B	0.86	0.44
		C	0.04	
		D	0.62	
24 - 25 m	3225	A	0.88	
		B	0.99	0.76
		C	0.08	
		D	1.10	
25 - 26 m	3226	A	0.03	
		B	0.07	0.15
		C	0.36	
		D	0.14	
26 - 27 m	3227	A	0.05	
		B	0.27	0.15
		C	0.02	
		D	0.25	
27 - 28 m	3228	A	0.03	
		B	0.05	0.45
		C	0.21	
		D	0.61	
28 - 29 m	3229	A	0.03	
		B	Nil	0.03
		C	0.01	
		D	0.07	

Boliden adit, sampled on south wall

Location	Sample no.		gold g/t	average gold g/t
29 - 30 m	3230	A	0.01	
		B	0.14	0.05
		C	0.03	
		D	0.03	
30 - 31 m	3231	A	0.04	
		B	1.86	6.40
		C	23.04	
		D	0.67	
31 - 32 m	3232	A	1.02	
		B	Sample missing	0.88
		C	0.85	
		D	0.78	
32 - 33 m	3233	A	1.79	
		B	0.22	0.67
		C	0.27	
		D	0.42	
33 - 34 m	3234	A	10.12	
		B	0.70	5.68
		C	0.16	
		D	11.73	
34 - 35 m	3235	A	0.46	
		B	0.18	0.22
		C	0.07	
		D	0.18	
35 - 36 m	3236	A	Nil	
		B	0.01	Nil
		C	0.01	
		D	Nil	
37 - 38	3237	A	Nil	
		B	Nil	Nil
		C	Nil	
		D	Nil	

Boliden adit, sampled on south wall

Location	Sample no.		gold g/t	average gold g/t
37 - 38 m	3238	A	0.03	
		B	Nil	0.28
		C	0.02	
		D	1.08	
38 - 39 m	3239	A	0.01	
		B	Nil	Nil
		C	Nil	
		D	Nil	
39 - 40 m	3240	A	Nil	
		B	Nil	Nil
		C	Nil	
		D	Nil	
40 - 41 m	3241	A	Nil	
		B	Nil	Nil
		C	Nil	
		D	Nil	

MANNERHEIM ADIT Sampled on north wall

Location	Sample no.		gold g/t	average gold g/t
0 - 1 m	3331	A	6.61	
		B	0.13	2.12
		C	0.17	
		D	1.57	
1 - 2 m	3332	A	0.20	
		B	0.24	0.24
		C	0.17	
		D	0.37	
2 - 3 m	3333	A	0.29	
		B	0.34	0.23
		C	0.31	
		D	Nil	
3 - 4 m	3334	A	0.01	
		B	0.19	
		C	0.25	0.13
		D	0.07	
4 - 5 m	3335	A	0.17	
		B	0.14	0.48
		C	0.87	
		D	0.74	
5 - 6 m	3336	A	0.24	
		B	3.41	1.13
		C	0.25	
		D	0.62	
6 - 7 m	3337	A	0.71	
		B	0.66	1.13
		C	2.56	
		D	0.59	

Mannerheim adit, sampled on north wall.

Location	Sample no.		gold g/t	average gold g/t
7 - 8 m	3338	A	0.14	
		B	0.03	0.09
		C	0.03	
		D	0.15	
8 - 9 m	3339	A	0.03	
		B	Nil	0.04
		C	Nil	
		D	0.15	
9 - 10 m	3340	A	0.02	
		B	0.14	0.05
		C	0.03	
		D	Nil	
10 - 11 m	3341	A	0.39	
		B	0.50	0.42
		C	0.24	
		D	0.55	
11 - 12 m	3342	A	0.04	
		B	0.10	0.06
		C	0.07	
		D	0.05	
12 - 13 m	3343	A	0.03	
		B	0.06	0.04
		C	0.03	
		D	0.04	
13 - 14 m	3344	A	0.07	
		B	0.17	0.10
		C	0.13	
		D	0.04	
14 - 15 m	3345	A	0.07	
		B	0.05	0.03
		C	Nil	
		D	Nil	

Mannerheim adit, sampled on north wall.

Location	Sample no.		gold g/t	average gold g/t
15 - 16 m	3346	A	Nil	
		B	0.17	0.05
		C	0.03	
		D	Nil	
16 - 17 m	3347	A	0.86	
		B	0.03	0.22
		C	Nil	
		D	Nil	
17 - 18 m	3348	A	Nil	
		B	0.10	0.12
		C	0.24	
		D	0.09	
18 - 19 m	3349	A	0.16	
		B	Nil	0.05
		C	Nil	
		D	0.03	
19 - 20 m	3350	A	0.01	
		B	0.03	
		C	0.01	0.01
		D	Nil	
20 - 21 m	3351	A	Nil	
		B	Nil	Nil
		C	0.05	
		D	NIl	
21 - 22 m	3352	A	0.26	
		B	0.51	
		C	0.72	0.37
		D	Nil	

SOUTH SKAR ADIT Sampled on south wall

Location	Sample no.		gold g/t	average gold g/t
0 - 1 m	3251	A	0.05	
		B	0.20	0.08
		C	Nil	
		D	0.09	
1 - 2 m	3252	A	0.03	
		B	0.05	0.05
		C	0.03	
		D	0.10	
2 - 3 m	3253	A	0.08	
		B	0.07	0.05
		C	0.03	
		D	0.03	
3 - 4 m	3254	A	0.13	
		B	0.16	0.30
		C	0.81	
		D	0.10	
4 - 5 m	3255	A	0.22	
		B	0.10	0.15
		C	0.12	
		D	0.17	
5 - 6 m	3256	A	0.13	
		B	2.04	0.62
		C	0.16	
		D	0.15	
6 - 7 m	3257	A	0.98	
		B	1.17	0.65
		C	0.19	
		D	0.28	

South Skar adit, sampled on south wall.

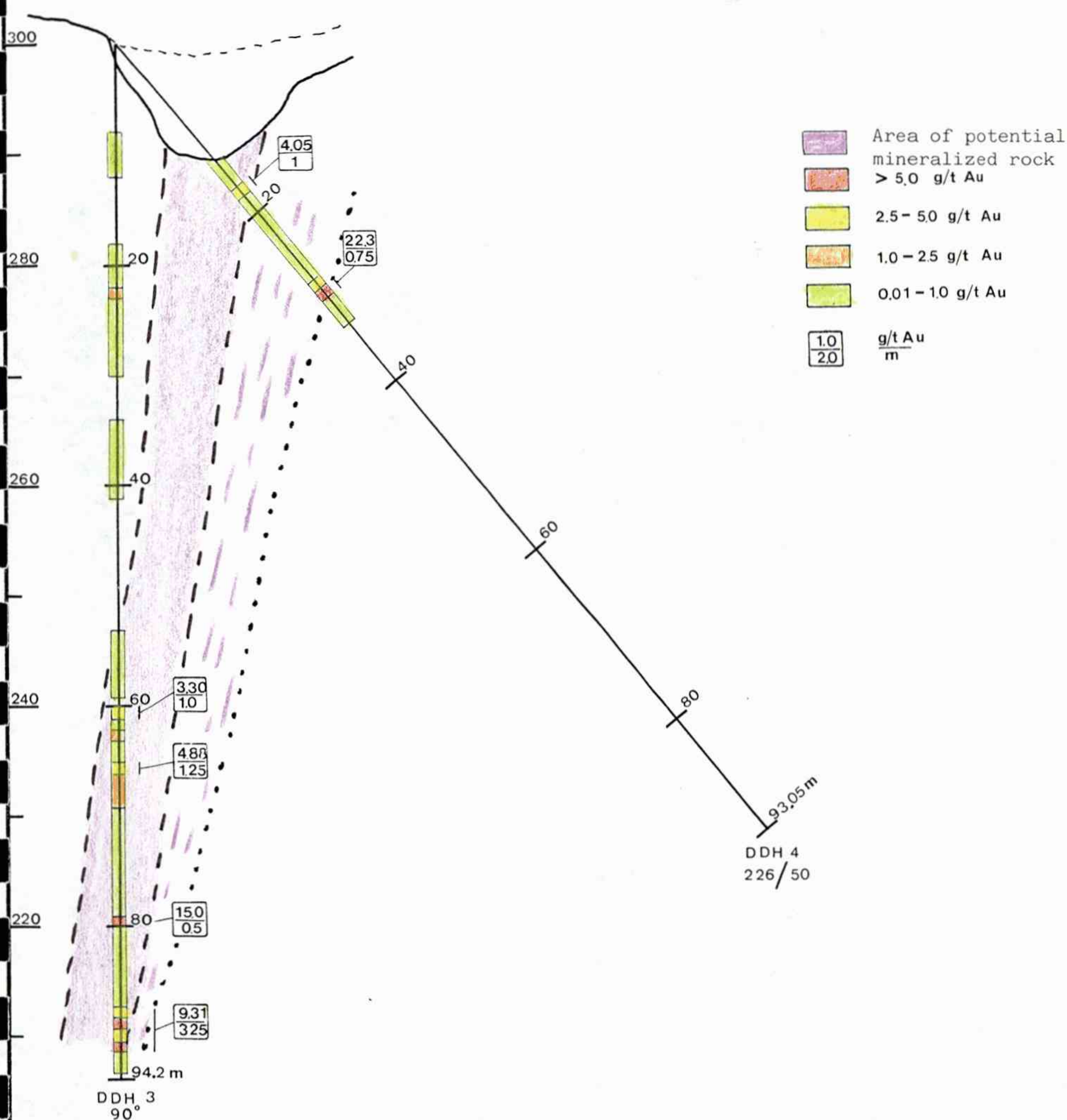
Location	Sample no.		gold g/t	average gold g/t
7 - 8 m	3258	A	0.49	
		B	0.01	0.32
		C	0.55	
		D	0.24	
8 - 9 m	3259	A	0.02	
		B	Nil	0.01
		C	0.03	
		D	0.01	
9 - 10 m	3260	A	Nil	
		B	Nil	Nil
		C	0.01	
		D	Nil	
10 - 11 m	3261	A	0.16	
		B	8.34	3.90
		C	7.17	
		D	0.08	
11 - 12 m	3262	A	0.65	
		B	0.95	0.48
		C	0.21	
		D	0.11	
12 - 13 m	3263	A	0.14	
		B	3.05	0.98
		C	0.55	
		D	0.15	

NORTH SKAR ADIT   Sampled on north wall

Location	Sample no.		gold g/t	average gold g/t
1 - 2 m	3314	A	Nil	
		B	Nil	0.01
		C	Nil	
		D	0.06	
2 - 3 m	3301	A	0.03	
		B	0.31	0.16
		C	0.22	
		D	0.07	
3 - 4 m	3302	A	0.02	
		B	0.07	0.06
		C	Nil	
		D	0.16	
4 - 5 m	3303	A	0.96	
		B	0.29	0.81
		C	0.44	
		D	1.56	
5 - 6 m	3304	A	0.21	
		B	1.44	0.53
		C	0.31	
		D	0.16	
6 - 7 m	3305	A	1.22	
		B	0.19	0.68
		C	0.76	
		D	0.55	
7 - 8 m	3306	A	0.12	
		B	0.03	0.17
		C	0.14	
		D	0.38	

North Skar adit, sampled on south wall.

Location	Sample no.		gold g/t	average gold g/t
8 - 9 m	3307	A	1.22	
		B	0.68	0.30
		C	0.22	
		D	0.20	
9 - 10 m	3308	A	2.45	
		B	0.05	0.68
		C	0.06	
		D	0.16	
10 - 11 m	3309	A	Nil	
		B	Nil	Nil
		C	0.02	
		D	Nil	
11 - 12 m	3310	A	Nil	
		B	0.10	
		C	0.02	0.03
		D	0.10	
12 - 13 m	3311	A	Nil	
		B	Nil	0.01
		C	0.06	
		D	Nil	
13 - 14 m	3312	A	0.46	
		B	0.04	0.17
		C	0.17	
		D	0.03	
14 - 15 m	3313	A	Nil	
		B	Nil	Nil
		C	0.01	
		D	Nil	



KOLSVIK, BINDAL, NORWAY.
DDH 3,4.

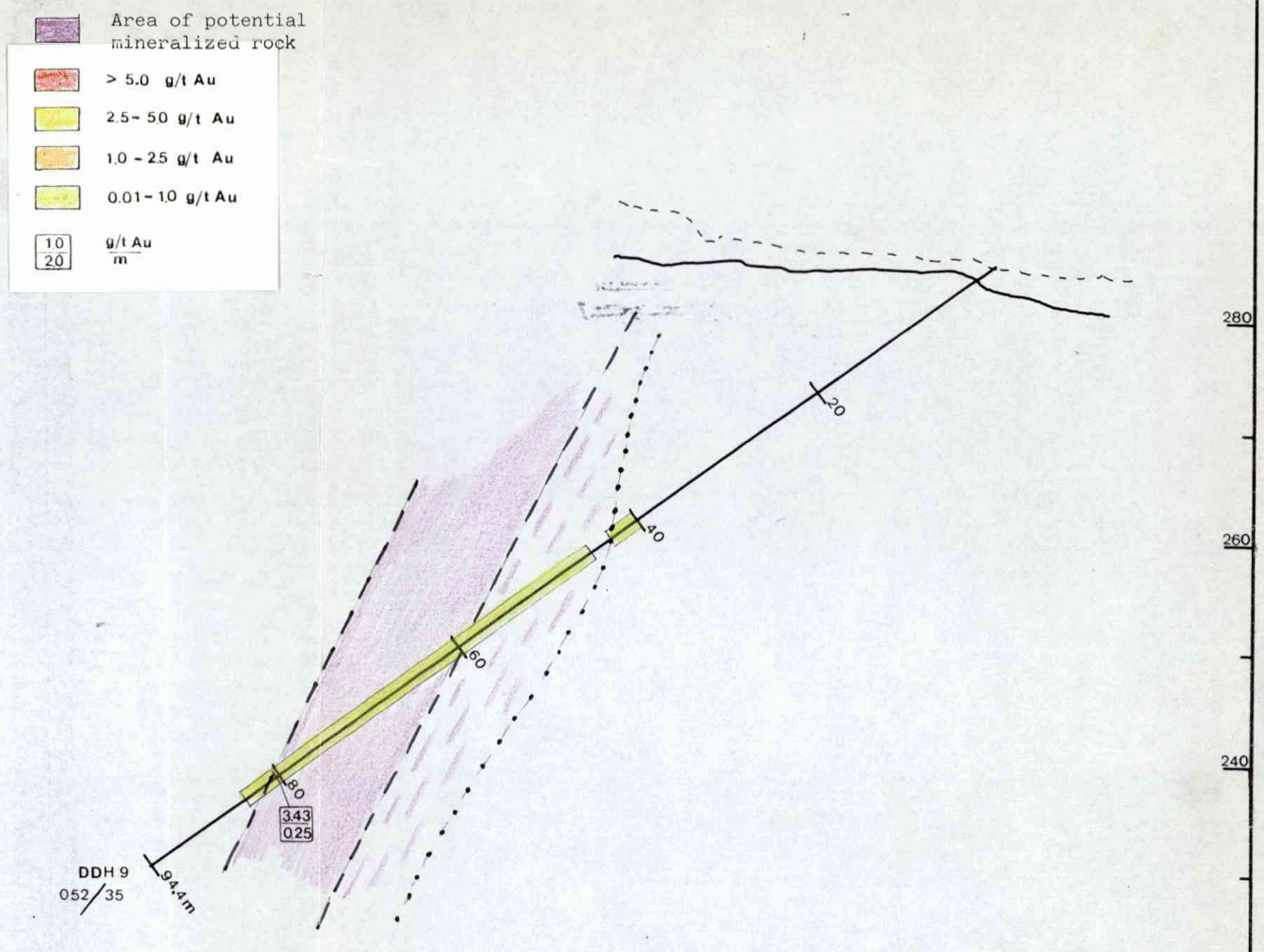
SCALE 1: 500

OBS.
DRAW.
TRAC.
CHK.

MAP NO.

A/S SULFIDMALM

MAP SHEET



KOLSVIK BINDAL NORWAY
DDH-9

SCALE 1:500

OBS.	
DRAW. LN	3-83
TRAC.	
CHK.	

A/S SULFIDMALM

MAP NO. 28 83 D7 7C54

MAP SHEET

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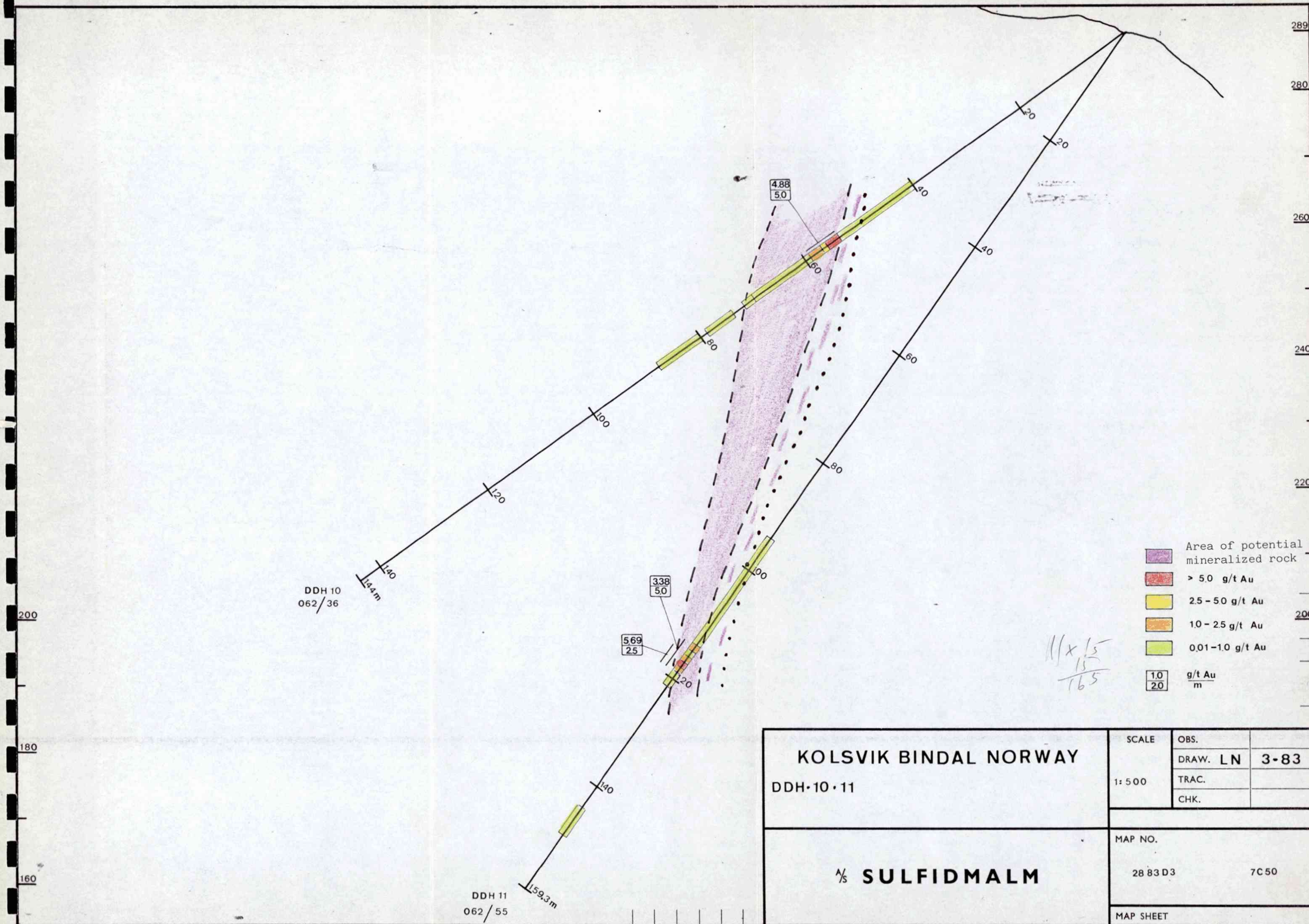

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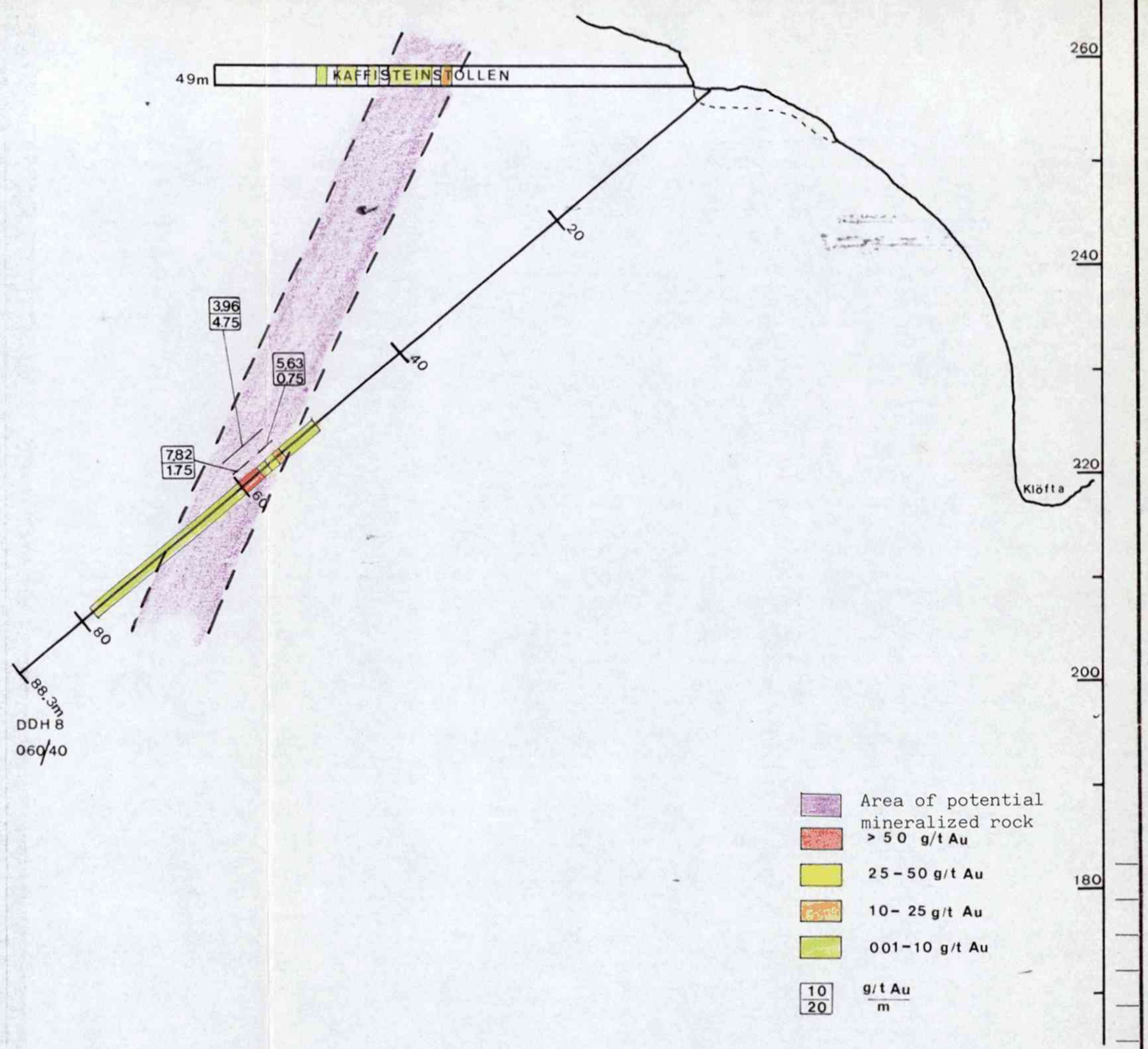
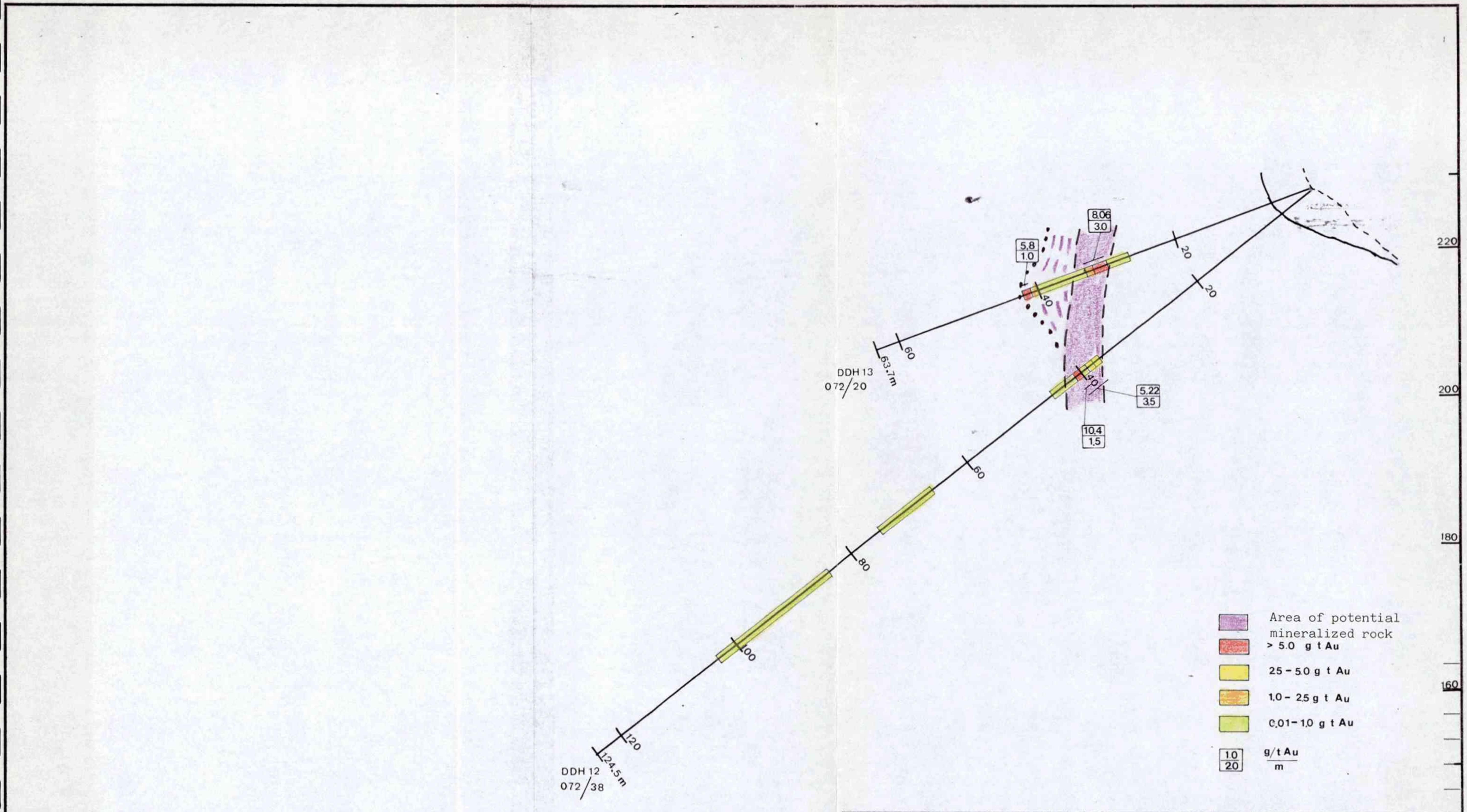


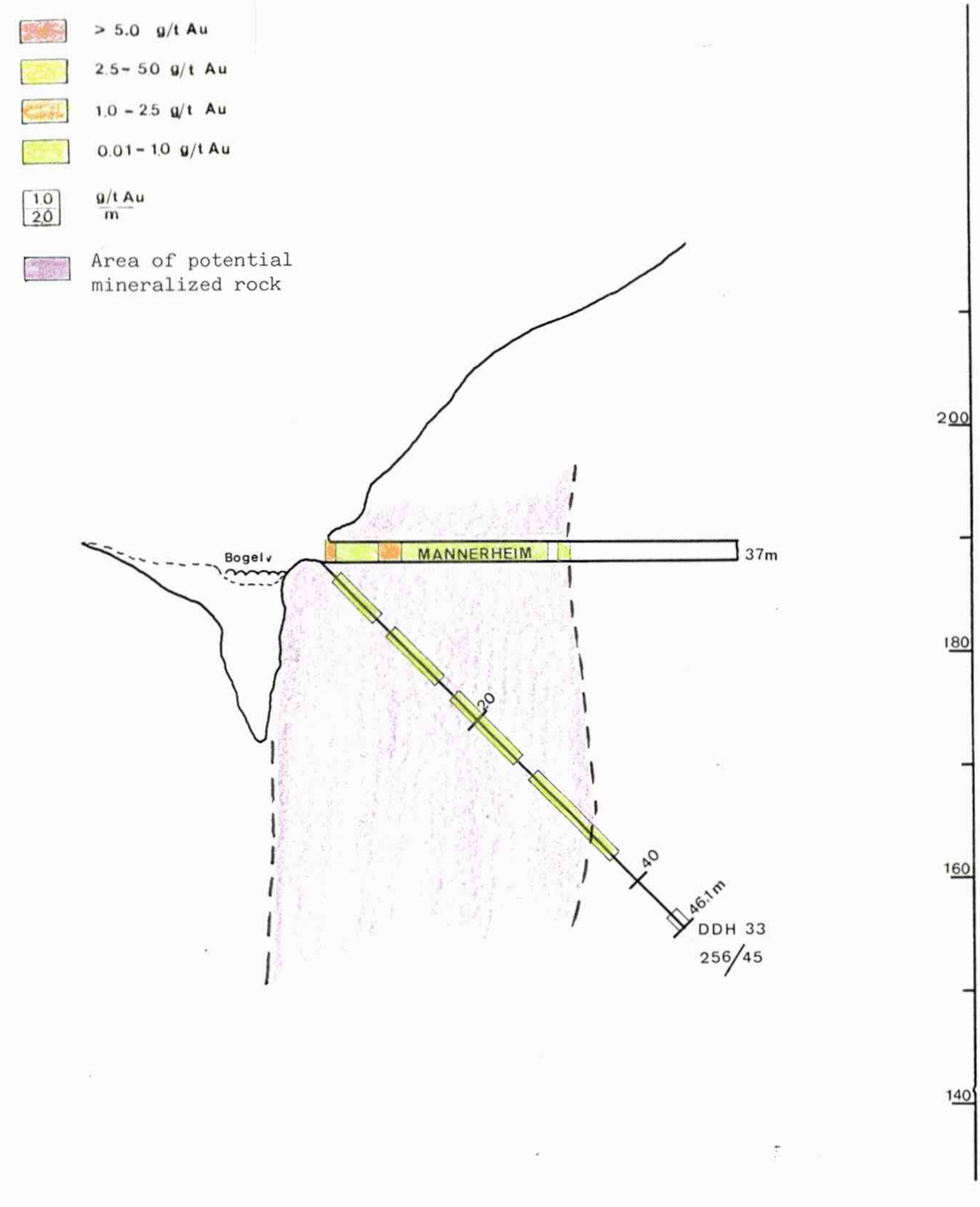
KOLSVIK BINDAL NORWAY DDH 8	SCALE 1:500	OBS.	
		DRAW. LN	3 · 83
		TRAC.	
		CHK.	
A/S SULFIDMALM	MAP NO. 28 83 D2	7C 49	
	MAP SHEET		

A.s Tørrkopi. 1000. A 3. E 9 gl. 10-71.-Sentrum Trykkeri.



KOLSVIK BINDAL NORWAY DDH 12, 13.	SCALE 1:500	OBS.	
		DRAW. LN	3·83
		TRAC.	
		CHK.	
A/S SULFIDMALM	MAP NO. 28 83 D1 7C48		
	MAP SHEET		

A.s Tørrkopi. 1000. A 3. E 9 gl. 10-71. Sentrum Trykkeri.



KOLSVIK BINDAL NORWAY DDH 33	SCALE 1:500	OBS.	
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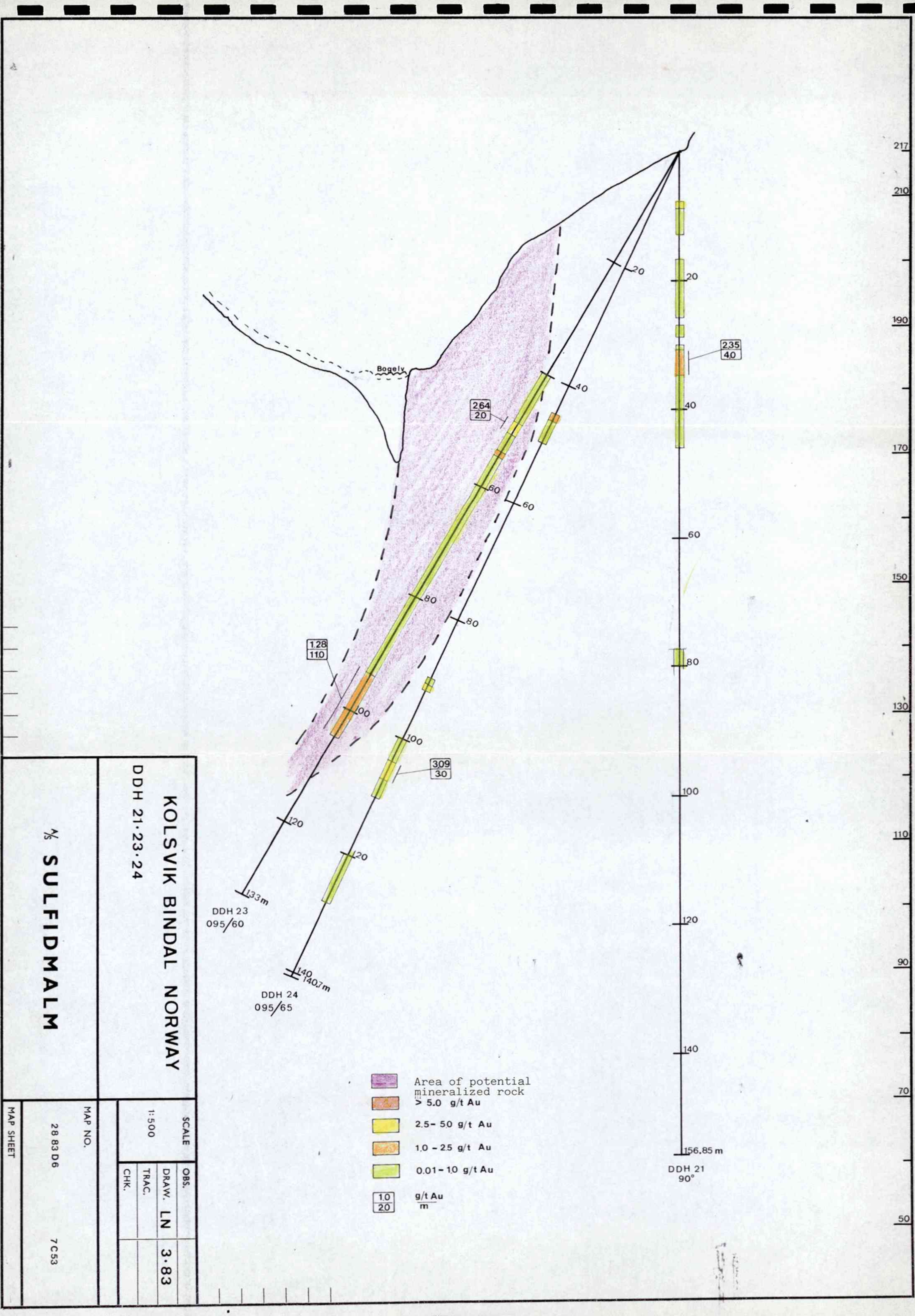
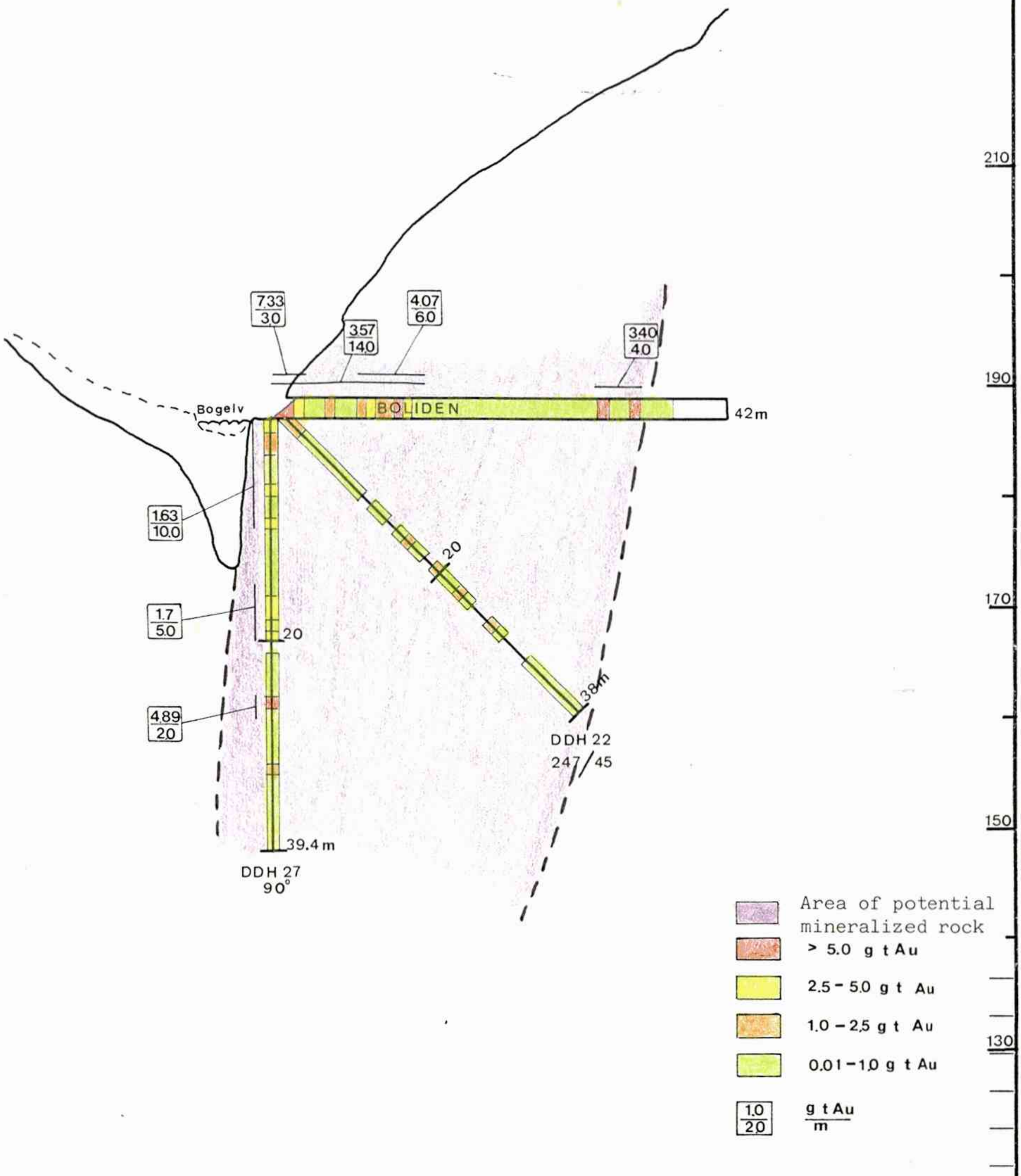
		TRAC.	
		CHK.	
A/S SULFIDMALM	MAP NO.		
	MAP SHEET		

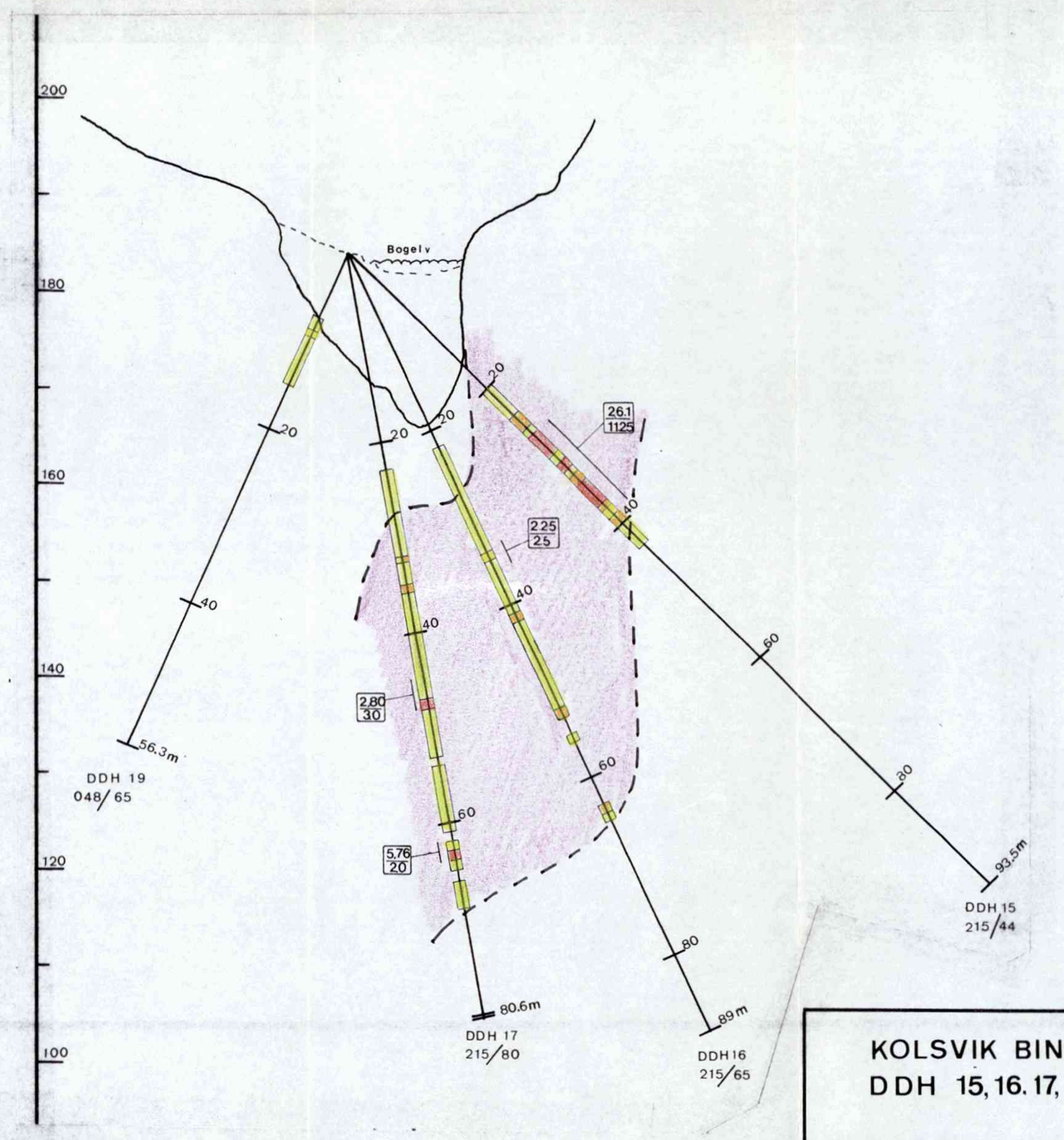


KOLSVIK BINDAL NORWAY
DDH 21·23·24
A/S SULFIDMALM
SCALE 1:500
OBS.
DRAW. LN 3·83
TRAC.
CHK.
MAP NO. 28 83 D6
MAP SHEET 7C53
Bogelv
DDH 23 095/60
DDH 24 095/65
DDH 21 90°
133 m
140.7 m
156.85 m
2.64 / 2.0
2.35 / 4.0
1.28 / 11.0
3.09 / 3.0
Area of potential mineralized rock
> 5.0 g/t Au
2.5- 5.0 g/t Au
1.0 - 2.5 g/t Au
0.01- 1.0 g/t Au
1.0 / 2.0 g/t Au / m
217
210
190
170
150
130
110
90
70
50
A.s Terrkopi. 1000. A 3. E 9 gl. 10-71. Sentrum Trykkeri.



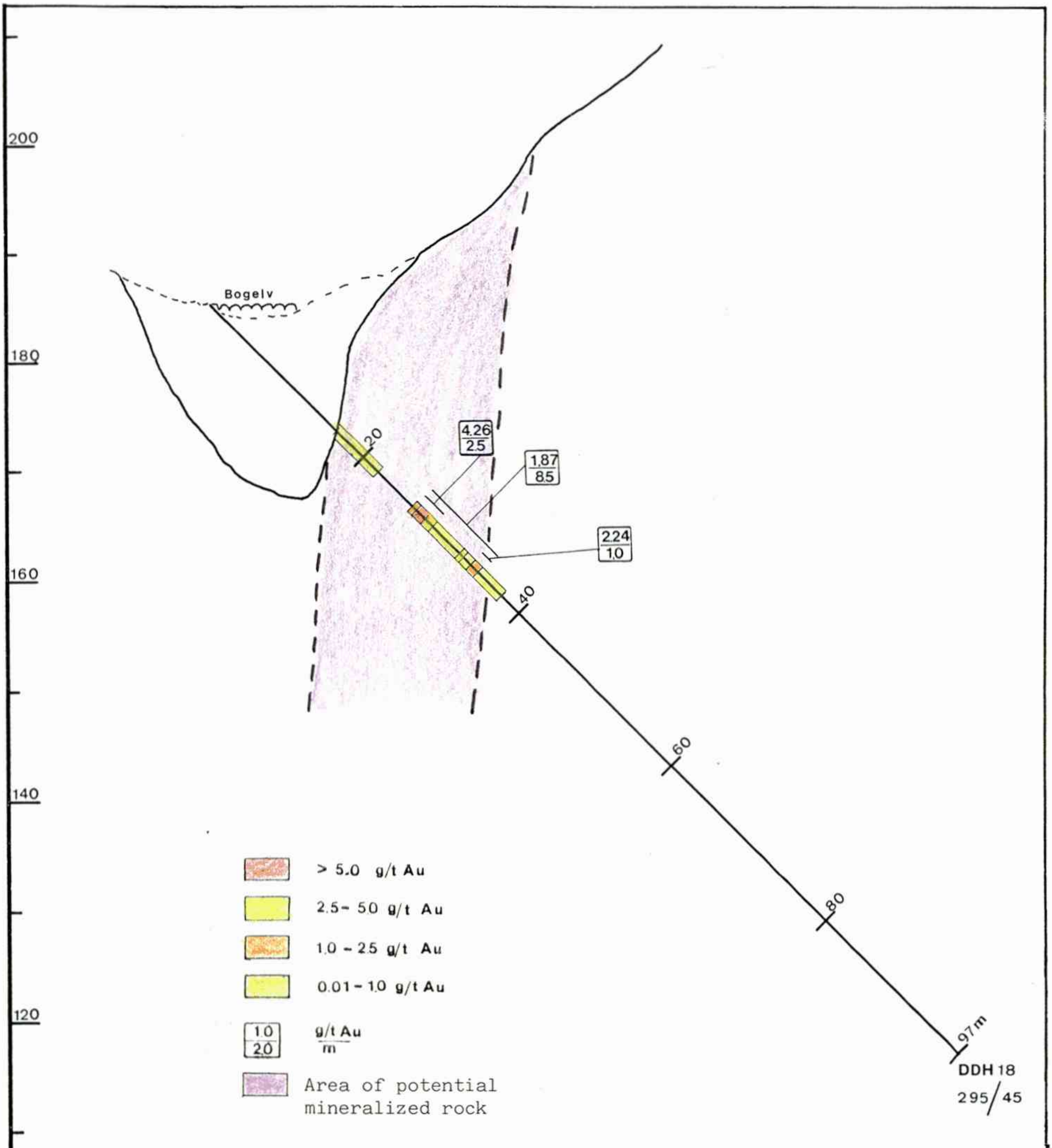
KOLSVIK BINDAL NORWAY DDH 22·27	SCALE 1:500	OBS.		
		DRAW.	LN	3 · 83
		TRAC.		
		CHK.		
A/S SULFIDMALM	MAP NO. 28 83 D5 7C52			
	MAP SHEET			

A.s T.

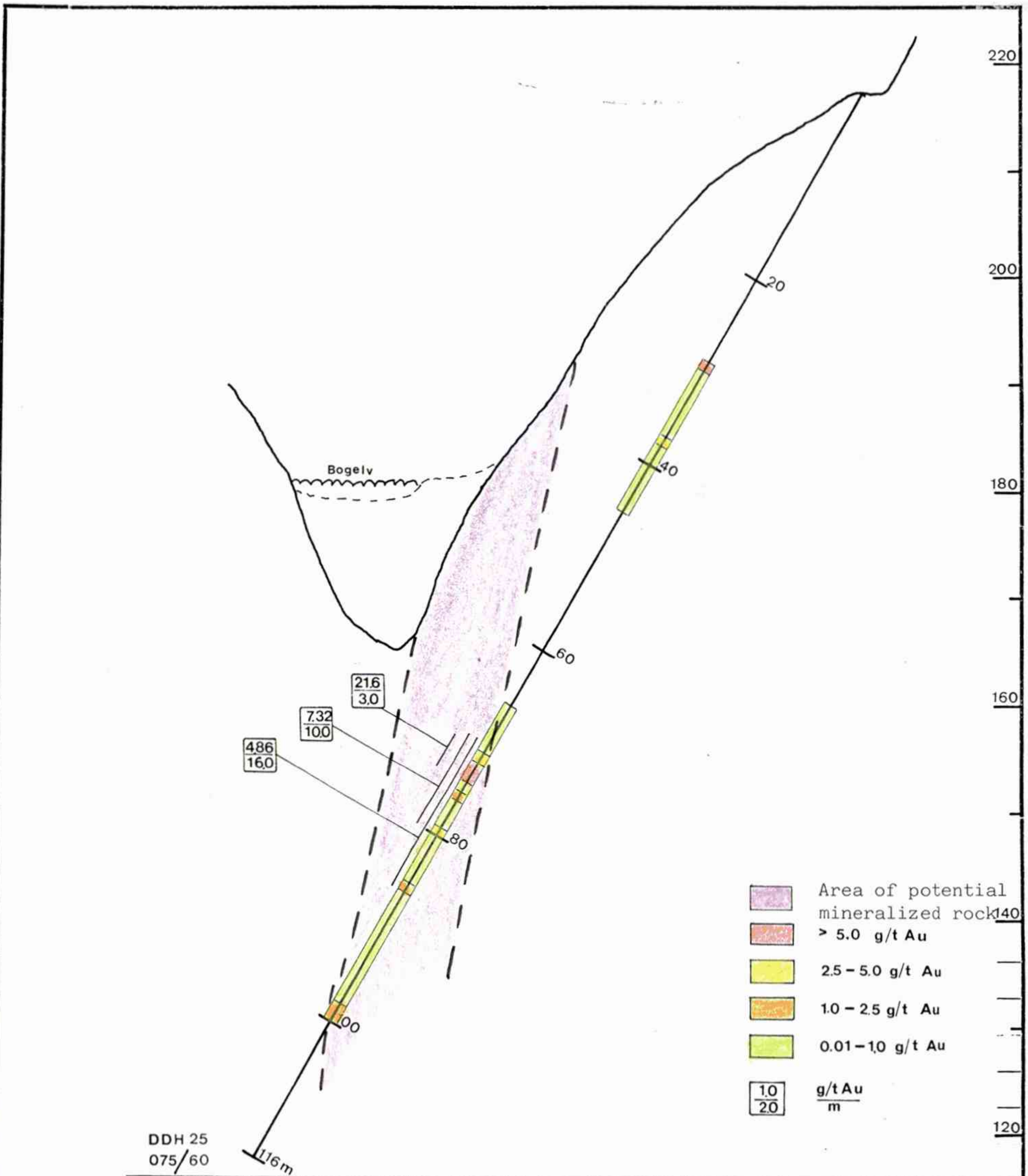


KOLSVIK BINDAL NORWAY DDH 15, 16. 17, 19.	SCALE 1 500	OBS.	
		DRAW.	
		TRAC.	
		CHK.	
A/S SULFIDMALM	MAP NO.		
	MAP SHEET		

A.s Tørrkopi. 1000. A 3. E 9 gl. 10-71. Sentrum Trykkeri.

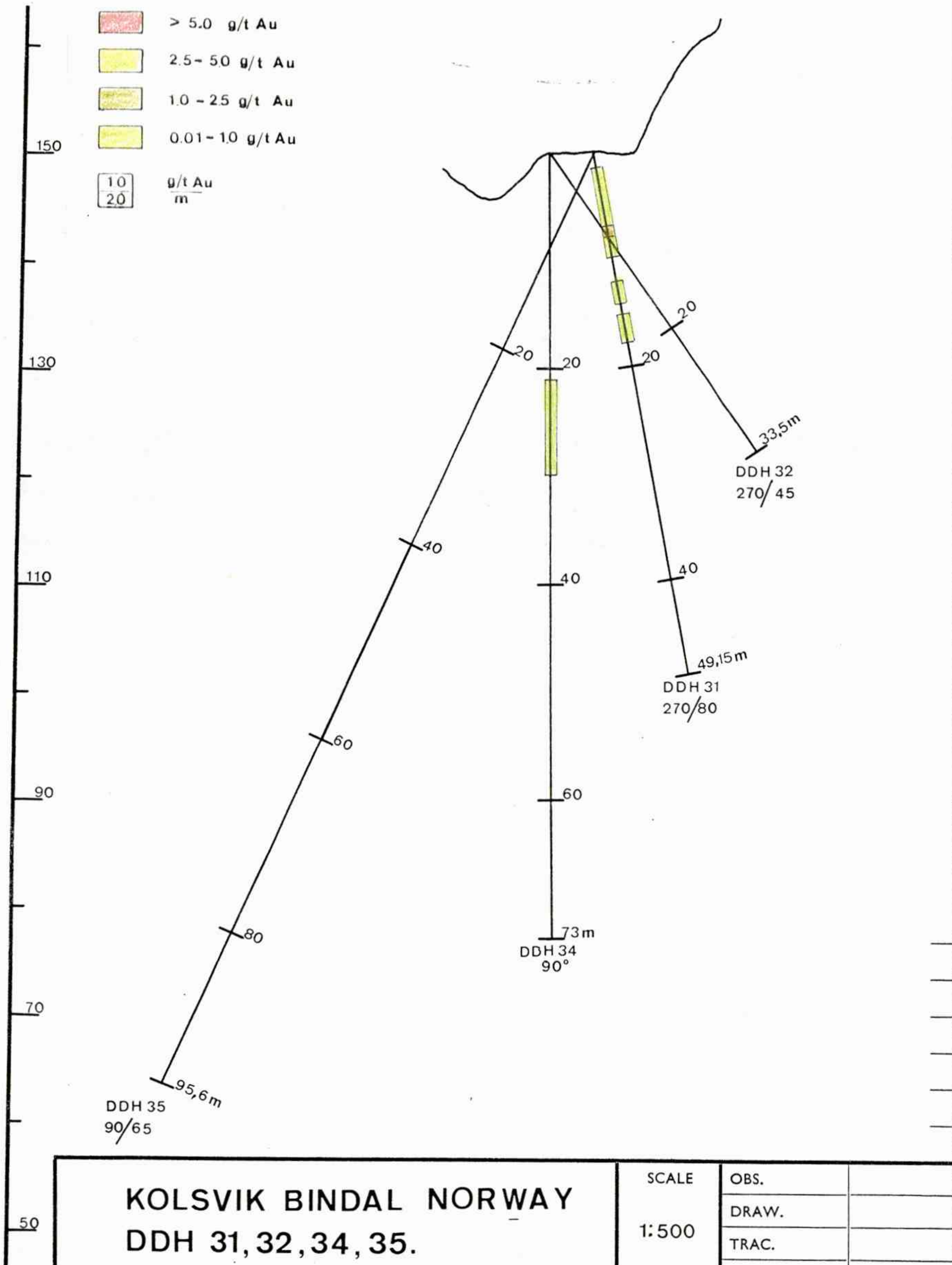


KOLSVIK BINDAL NORWAY DDH 18	SCALE 1500	OBS.	
		DRAW.	
		TRAC.	
		CHK.	
A/S SULFIDMALM	MAP NO.		
	MAP SHEET		



KOLSVIK BINDAL NORWAY DDH 25	SCALE 1: 500	OBS.	
		DRAW. LN	3 · 83
		TRAC.	
		CHK.	
A/S SULFIDMALM	MAP NO. 28 83 D4		7C51
	MAP SHEET -		

A.s Tørrkopi. 10



KOLSVIK BINDAL NORWAY DDH 31, 32, 34, 35.	SCALE 1:500	OBS.	
		DRAW.	
		TRAC.	
		CHK.	
A/S **SULFIDMALM**	MAP NO.		
	MAP SHEET		

Location	Kolsvik, Norway	Lab. No. 82-141
Sample Description	DDH 14 @ 61.95 m	PTS No. 6827

MINERALS	Est. % by Vol.	Grain Size (m.m.) Max.	Avg.
Feldspar - Oligoclase $\pm$ An_{22}	30 - 35		
Orthoclase	15 - 20		
Quartz	30 - 35		
Biotite	6 - 8		
Muscovite/Sericite	3 - 4		
Garnet	tr		
Zircon	tr		
Apatite	tr		
Magnetite, Ilmenite	tr		
Pyrite, Marcasite, Chalcopyrite	tr		

DESCRIPTION

Augen textures are evident in hand sample but the textures in pol-thin section are granitic. Scattered coarse flakes of biotite and muscovite occur in a coarse mosaic of feldspar, both sodic and potassic, and quartz. The latter shows evidence of deformation by the presence of strain shadows and slight granulation. One grain of garnet was observed in the section.

Augen gneiss of quartz monzonite composition

Location	Kolsvik, Norway	Lab. No. 82-141
Sample Description	DDH 14 @ 74.70 m	PTS No. 6828

MINERALS	Est. % by Vol.	Grain Size Max.	(m.m.) Avg.
Feldspar - Oligoclase	30 - 35		
Feldspar - Orthoclase	15 - 20		
Quartz	30 - 35		
Biotite	6 - 8		
Muscovite/Sericite	2 - 3		
Garnet	∿1		
Chlorite	2 - 3		
Apatite	tr		
Pyrrhotite, Marcasite	tr		
Magnetite, Ilmenite	tr		

DESCRIPTION

Very similar in composition to the previous sample. However, the pol-thin section displays strong orientation of biotite/muscovite/chlorite flakes to produce a gneissic texture.

Augen gneiss of quartz monzonite composition

Location	Kolsvik, Norway	Lab. No. 82-141
Sample Description	DDH 14 @ 97.40 m	PTS No. 6829

MINERALS		Est. % by Vol.	Grain Size Max.	(m.m.) Avg.
Feldspar -	Oligoclase $\pm An_{27}$	30 - 35		
	Orthoclase	12 - 15		
Quartz		15 - 20		
Amphibole		15 - 20		
Biotite		10 - 12		
Zircon, Apatite		tr		
Sphene		1 - 2		
Carbonate		tr		
Wolframite(?)		tr		
Pyrite		tr		

DESCRIPTION

This sample is finer grained and more mafic than the samples at 61.95 m and 74.70 m. It contains about 25% dark minerals but from the total mineral assemblage it is likely a mafic member of the same quartz monzonite unit.

A brown translucent mineral was picked out from the section and subjected to X-ray powder diffraction. Its pattern fits closely that of wolframite but a search of the spectrographic film revealed no lines diagnostic of tungsten.

Quartz Monzonite

Location	Kolsvik, Norway	Lab. No. 82-141
Sample Description	DDH 14 @ 100.70 m	PTS No. 6830

MINERALS		Est. % by Vol.	Grain Size (m.m.) Max.	Avg.
Feldspar -	Oligoclase $\pm$ An_{25}	40 - 45		
	Orthoclase	8 - 10		
Quartz		6 - 8		
Amphibole		25 - 30		
Biotite		4 - 6		
Epidote		1 - 2		
Sphene		3 - 4		
Carbonate		1 - 2		
Sericite		tr		
Pyrite		tr		

DESCRIPTION

This sample is less siliceous than the previous one and it is classified as a monzonite rather than a quartz monzonite. Dark green hornblende, green biotite and relatively abundant sphene are similar to the assemblage in PTS-6829.

Monzonite

Location	Kolsvik, Norway	Lab. No. 82-141
Sample Description	DDH 14 @ 125.40 m	PTS No. 6831

MINERALS		Est. % by Vol.	Grain Size Max.	(m.m.) Avg.
Feldspar -	Oligoclase	45 - 50		
	Orthoclase	10 - 12		
Quartz		6 - 8		
Amphibole		25 - 30		
Biotite		3 - 5		
Sphene		2 - 3		
Pyrite		tr		

DESCRIPTION

Almost identical to the previous sample at 100.70 m.

Monzonite

1008

Location	Kolsvik, Norway	Lab. No. 82-141
Sample Description	DDH 15 @ 30.60 m	PTS No. 6832

MINERALS		Est. % by Vol.	Grain Size Max.	(m.m.) Avg.
Quartz		70 - 75		
Feldspar -	Orthoclase	6 - 8		
	Oligoclase	1 - 2		
Carbonate		18 - 20		
Biotite		tr		

DESCRIPTION

Coarse irregular patches of highly strained quartz and carbonate occur with a heavily granulated, carbonatized, medium grained rock of granitic composition. Some late veins are lined by dog-tooth quartz and show vuggy textures.

Highly deformed and carbonatized granite

Location	Kolsvik, Norway	Lab. No.	82-141
Sample Description	DDH 15 @ 35.30 m	PTS No.	6833

MINERALS	Est. % by Vol.	Grain Size (m.m.) Max.	Avg.
Feldspar - {Andesine	20 - 25		
Feldspar - {Microcline	15 - 20		
Feldspar - {Orthoclase	50 - 55		
Quartz	4 - 5		
Muscovite	2 - 3		
Garnet	tr		
Apatite	tr		
Carbonate	∿1		

DESCRIPTION

This sample described as "typical red granite" contains only minor amounts of quartz and must be classified as syenite rather than granite.

Syenite

Location	Kolsvik, Norway	Lab. No.	82-141
Sample Description	DDH 15 @ 80.50 m	PTS No.	6834

MINERALS	Est. % by Vol.	Grain Size Max.	(m.m.) Avg.
Feldspar - {Microcline	40 - 45		
Feldspar - {Orthoclase	10 - 15		
Feldspar - {Oligoclase	10 - 15		
Quartz	20 - 25		
Muscovite/Sericite	4 - 5		
Chlorite	tr		
Carbonate	tr		

DESCRIPTION

A medium- to coarse grained granite shows evidence of strong deformation. Quartz invariably has strain shadows or is partly granulated.

Granite

Location	Kolsvik, Norway	Lab. No.	82-141
Sample Description	DDH 18 @ 18.85 m	PTS No.	6835

MINERALS	Est. % by Vol.	Grain Size (m.m.) Max.	Avg.
Feldspar - Orthoclase	25 - 30		
Feldspar - Oligoclase	8 - 10		
Quartz	8 - 10		
Chlorite	20 - 25		
Sericite	12 - 15		
Carbonate	10 - 12		
Sphene	1 - 2		
Apatite, Zircon, Pyrite	tr		
Ilmenite	2 - 3		

DESCRIPTION

The section is characterized by very high sericitic alteration of feldspar and by abundant flakes of chlorite. Carbonate usually occurs in late veinlets occasionally with quartz. Compared to others, the rock is generally fine grained and shows moderate gneissic textures.

Monzonite

Location	Kolsvik, Norway	Lab. No. 82-141
Sample Description	DDH 18 @ 22.65 m	PTS No. 6836

MINERALS	Est. % by Vol.	Grain Size Max.	(m.m.) Avg.
Feldspar - Oligoclase	6 - 8		
Quartz	10 - 12		
Amphibole	2 - 3		
Epidote	25 - 30		
Biotite	4 - 5		
Chlorite	25 - 30		
Carbonate	<1		
Apatite	∿1		
Garnet	1 - 2		
Sphene	∿1		
Pyrite	8 - 10		
Chalcopyrite	<1		

DESCRIPTION

Peculiar textures in pol-thin section shows coarse grained feldspar completely replaced by chlorite. Interstitial to the altered feldspar are coarse epidote, strained quartz, anhedral pyrite and occasional grains of fresh feldspar, biotite and anhedral orange garnet. Weak chalcopyrite mineralization occurs in gangue rather than in the coarse pyrite.

The intensity of alteration makes it difficult to assess the original rock type. However, from the mineral assemblage it likely represents a highly altered monzonite or diorite.

Location	Kolsvik, Norway	Lab. No. 82-141
Sample Description	DDH 18 @ 27.30 m	PTS No. 6837

MINERALS		Est. % by Vol.	Grain Size Max.	(m.m.) Avg.
Feldspar -	Orthoclase	20 - 25		
	Albite-oligoclase	15 - 18		
Quartz		10 - 12		
Biotite		35 - 40		
Rutile		2 - 3		
Carbonate		6 - 8		
Zircon		tr		

DESCRIPTION

PTS-6837 consists of about 60:40 host rock to vein material. The latter is about 90% coarse grained, highly strained quartz with crosscutting veinlets and patches of carbonate. The texture of the biotite-rich host is almost sedimentary rather than igneous with grains of feldspar and quartz surrounded by biotite. Rutile is prominent in the section as small blocky translucent brown grains. Carbonate occurs as late shear infillings.

The rock is classified as a biotite schist of uncertain origin which is heavily penetrated by quartz/carbonate veinlets.

Location	Kolsvik, Norway	Lab. No. 82-141
Sample Description	DDH 18 @ 31.95 m	PTS No. 6838

MINERALS	Est. % by Vol.	Grain Size Max.	(m.m.) Avg.
Feldspar - {Orthoclase	20 - 25		
Feldspar - {Microcline	20 - 25		
Feldspar - {Oligoclase $\pm$ An_{28}	15 - 20		
Quartz	25 - 30		
Chlorite	~1		
Muscovite/Sericite	~1		
Carbonate	4 - 5		
Apatite, Sphene	tr		
Arsenopyrite	2 - 3		
Sphalerite	~1		

DESCRIPTION

Euhedral arsenopyrite grains in this granite are adjacent to or within shears. Patches of sphalerite are also invariably accompanied by shear infillings of carbonate.

Granite

Location	Kolsvik, Norway	Lab. No. 82-141
Sample Description	DDH 18 @ 39.95 m	PTS No. 6839

MINERALS		Est. % by Vol.	Grain Size Max.	(m.m.) Avg.
Feldspar -	Orthoclase	30 - 35		
	Oligoclase	10 - 12		
Quartz		12 - 15		
Hornblende		20 - 25		
Biotite		8 - 10		
Epidote		1 - 2		
Carbonate		2 - 3		
Sphene		3 - 4		
Chlorite		<1		
Apatite, Pyrrhotite, Chalcopyrite		tr		
Pyrite		∿1		

DESCRIPTION

This is a good example of a medium - to fine grained quartz monzonite which is closely approaching monzonite (<10% quartz) in composition. Sphene is very prominent throughout PTS-6839 in small subhedral to euhedral grains.

Location	Kolsvik, Norway	Lab. No. 82-141
Sample Description	DDH 18 @ 82.95 m	PTS No. 6840

MINERALS		Est. % by Vol.	Grain Size Max.	(m.m.) Avg.
	{Orthoclase	30 - 35		
Feldspar -	{Microcline	15 - 20		
	{Oligoclase	15 - 20		
Quartz		25 - 30		
Muscovite/Sericite		4 - 5		
Chlorite		1 - 2		
Biotite		~1		
Magnetite, Pyrrhotite		tr		

DESCRIPTION

Good example of a leucocratic granite.

FALCONBRIDGE METALLURGICAL LABORATORIES
QUALITATIVE SPECTROGRAPHIC ANALYSIS

DISTRIBUTION: REPORT No. Q-1281

ANALYTICAL METHOD:

REQUESTED BY: DATE: May 12, 1982

RECEIVED FROM: CHARGE: JO#3064

SAMPLE No.: L#82-141 No. of SAMPLES: 15

SAMPLE DESCRIPTION: Miscellaneous Rocks

Kolsvik, Norway

	DDH 14 @ 61.95 m	DDH 14 @ 74.70 m	DDH 14 @ 97.40 m
10 – 100%	Si	Si	Si
3 – 30%	Fe, Al	Fe,Al	Fe, Al
1 – 10%	K, Ca	K,Ca	Mg, K, Ca
0.3 – 3%	Na, Ti	Na, Ti	Ti
0.1 – 1%	Mg	Mg	
0.03 – 0.3%			Cr
0.01 – 0.1%	Mn, Cr	Mn, Cr	Mn
0.003 – 0.03%	As, V, Zr, Ni	As, V, Zr, Ni	V, Zr, Ni
0.001 – 0.01%	Cu	Cu	Co
0.0003 – 0.003%	Co, Ba	Ba	Cu, Ba
0.0001 – 0.001%		Co	-
< 0.0003%			
I	Sr	Sr	Sr
S			Na>1%

I = Interference prevents positive identification.
S = Strong spectral lines, unable to estimate amount.
Unless specified above, the following were not detected at the approx. ppm lower limits of 0.5 Cu,Ag; 1 Mn; 5 Mg, Cr; 10 Ba, Be, Bi, Ca, Co, Ni, V; 25 Ge, Fe, Pb, Mo, Si, Sr, Sn, Ti, Zr, Tl, Pd; 50 Al, Sb, B, Cd, Ga, In, Li, Zn; 100 As, Au, Na; 200 Rh, Re, Ir, Pt, Ru, Sc; 300 Te, Os; 1000 K, U, Th; 2000 P.

FML-1017 Analyst

FALCONBRIDGE METALLURGICAL LABORATORIES

QUALITATIVE SPECTROGRAPHIC ANALYSIS

DISTRIBUTION: REPORT No. Q-1281

ANALYTICAL METHOD:

REQUESTED BY: DATE: May 12, 1982

RECEIVED FROM: CHARGE: JO#3064

SAMPLE No.: L#82-141 No. of SAMPLES: 15

SAMPLE DESCRIPTION: Miscellaneous Rocks

Kolsvik, Norway

	DDH 14 @ 100.70 m	DDH 14 @ 125.40 m	DDH 15 @ 30.60 m
10 – 100%	Si	Si	Si
3 – 30%	Fe, Al	Fe, Al	Al
1 – 10%	Mg, K, Ca	Mg, K, Ca	K, Ca
0.3 – 3%	Ti	Ti	Fe, Na, Ti
0.1 – 1%			As, Mg
0.03 – 0.3%	Cr	Cr	Cr
0.01 – 0.1%	Mn	Mn	
0.003 – 0.03%	V, Zr, Ni	V, Zr, Ni	Mn
0.001 – 0.01%	Co	Co	
0.0003 – 0.003%	Cu, Ba	Cu, Ba	Pb, Cu, Zr, Ni
0.0001 – 0.001%			V, Ba
< 0.0003%			Ag
I	Sr	Sr	Sr
S	Na>1%	Na>1%	

I = Interference prevents positive identification.

S = Strong spectral lines, unable to estimate amount.

Unless specified above, the following were not detected at the approx. ppm lower limits of 0.5 Cu,Ag; 1 Mn; 5 Mg, Cr; 10 Ba, Be, Bi, Ca, Co, Ni, V; 25 Ge, Fe, Pb, Mo, Si, Sr, Sn, Ti, Zr, Tl, Pd; 50 Al, Sb, B, Cd, Ga, In, Li, Zn; 100 As, Au, Na; 200 Rh, Re, Ir, Pt, Ru, Sc; 300 Te, Os; 1000 K, U, Th; 2000 P.

FML-1017

Analyst

FALCONBRIDGE METALLURGICAL LABORATORIES

QUALITATIVE SPECTROGRAPHIC ANALYSIS

DISTRIBUTION: REPORT No. Q-1281

ANALYTICAL METHOD:

REQUESTED BY: DATE: May 12, 1982

RECEIVED FROM: CHARGE: JO#3064

SAMPLE No.: L#82-141 No. of SAMPLES: 15

SAMPLE DESCRIPTION: Miscellaneous Rocks

Kolsvik, Norway

	DDH 15 @ 35.30 m	DDH 15 @ 80.50 m	DDH 18 @ 18.85 m
10 – 100%	Si	Si	Si
3 – 30%		Al	Fe,Al
1 – 10%	K	K	Mg, Ca
0.3 – 3%	Fe	Fe	Ti, K
0.1 – 1%	Ca	Mg, Ca	
0.03 – 0.3%		Ti	Cr
0.01 – 0.1%	Mg, Al, Cr	Cr	Mn, Ni
0.003 – 0.03%	As, Pb	As	As, V, Zr
0.001 – 0.01%	Mn, Ni	Mn	Cu, Co
0.0003 – 0.003%	Cu, Ti	Pb, Cu, Zr, Ni	
0.0001 – 0.001%		Ba	Ba
< 0.0003%	Ag		Ag
I	Sr	Sr	Sr
S	Na>1%	Na>1%	Na>1%

I = Interference prevents positive identification.

S = Strong spectral lines, unable to estimate amount.

Unless specified above, the following were not detected at the approx. ppm lower limits of 0.5 Cu, Ag; 1 Mn; 5 Mg, Cr; 10 Ba, Be, Bi, Ca, Co, Ni, V; 25 Ge, Fe, Pb, Mo, Si, Sr, Sn, Ti, Zr, Tl, Pd; 50 Al, Sb, B, Cd, Ga, In, Li, Zn; 100 As, Au, Na; 200 Rh, Re, Ir, Pt, Ru, Sc; 300 Te, Os; 1000 K, U, Th; 2000 P.

FML-1017 Analyst

FALCONBRIDGE METALLURGICAL LABORATORIES

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DISTRIBUTION: REPORT No. Q-1281

ANALYTICAL METHOD:

REQUESTED BY: DATE: May 12, 1982

RECEIVED FROM: CHARGE: JO#3064

SAMPLE No.: L#82-141 No. of SAMPLES: 15

SAMPLE DESCRIPTION: Miscellaneous Rocks

Kolsvik, Norway

	DDH 18 @ 22.65 m	DDH 18 @ 27.30 m	DDH 18 @ 31.95 m
10 – 100%	Si, Fe	Si	Si
3 – 30%	Al, K	Fe, Al	Al, K
1 – 10%	Mg, Ca	Mg, K, Ca	
0.3 – 3%	Ti	Ti	Fe, Ti, Ca
0.1 – 1%	Na		As, Mg
0.03 – 0.3%		As	
0.01 – 0.1%	Mn,Cr	Pb, Cr	Pb, Cr
0.003 – 0.03%	As,V,Cd,Zn,Zr,Ni	Mn, V, Zr	Zn
0.001 – 0.01%	Cu, Co		Mn, Zr
0.0003 – 0.003%	Sn	Cu, Ni, Ba	Cu, Ni
0.0001 – 0.001%	Mo	Co	V, Ba
< 0.0003%	Ag	Ag	Ag
I	Sr	Sr	Sr
S		Na>1%	Na>1%

I = Interference prevents positive identification.
S = Strong spectral lines, unable to estimate amount.
Unless specified above, the following were not detected at the approx. ppm lower limits of 0.5 Cu,Ag; 1 Mn; 5 Mg, Cr; 10 Ba, Be, Bi, Ca, Co, Ni, V; 25 Ge, Fe, Pb, Mo, Si, Sr, Sn, Ti, Zr, Tl, Pd; 50 Al, Sb, B, Cd, Ga, In, Li, Zn; 100 As, Au, Na; 200 Rh, Re, Ir, Pt, Ru, Sc; 300 Te, Os; 1000 K, U, Th; 2000 P.

FML-1017 Analyst

FALCONBRIDGE METALLURGICAL LABORATORIES

QUALITATIVE SPECTROGRAPHIC ANALYSIS

DISTRIBUTION: REPORT No. Q-1281

ANALYTICAL METHOD:

REQUESTED BY: DATE: May 12, 1982

RECEIVED FROM: CHARGE: JO#3064

SAMPLE No.: L#82-141 No. of SAMPLES: 15

SAMPLE DESCRIPTION: Miscellaneous Rocks

Kolsvik, Norway

	DDH 18 @ 39.95 m	DDH 18 @ 82.95 m
10 – 100%	Si	Si
3 – 30%	Fe, Al, Ca	Al
1 – 10%	Mg, K	K
0.3 – 3%	Ti	Fe, Ca
0.1 – 1%		Mg
0.03 – 0.3%		Ti
0.01 – 0.1%	Mn	Cr
0.003 – 0.03%	As, V, Zr, Ni	As
0.001 – 0.01%	Cu, Cr	Mn
0.0003 – 0.003%	Co, Ba	Pb, Cu, Zr, Ni
0.0001 – 0.001%		Ba
< 0.0003%	Ag	
I	Sr	Sr
S	Na>1%	Na>1%

I = Interference prevents positive identification.
S = Strong spectral lines, unable to estimate amount.
Unless specified above, the following were not detected at the approx. ppm lower limits of 0.5 Cu,Ag; 1 Mn; 5 Mg, Cr; 10 Ba, Be, Bi, Ca, Co, Ni, V; 25 Ge, Fe, Pb, Mo, Si, Sr, Sn, Ti, Zr, Tl, Pd; 50 Al, Sb, B, Cd, Ga, In, Li, Zn; 100 As, Au, Na; 200 Rh, Re, Ir, Pt, Ru, Sc; 300 Te, Os; 1000 K, U, Th; 2000 P.

FML-1017 Analyst

An Investigation of

THE RECOVERY OF GOLD

from samples

submitted by

A/S SULFIDMALM

Progress Report No. 1

Project No. L.R. 2570

Note:

This report refers to the samples as received.

LAKEFIELD RESEARCH OF CANADA LIMITED
Lakefield, Ontario
April 23, 1982

I N T R O D U C T I O N

In a letter dated December 18, 1981, Mr. Frank Nixon of A/S Sulfidmalm requested metallurgical tests on two samples of a gold-arsenopyrite ore from a Falconbridge Nickel Mines property in Bindal, Norway.

LAKEFIELD RESEARCH OF CANADA LIMITED

D. M. Wyslouzil

D.M. Wyslouzil, P. Eng.,

Manager.

R.G. Williamson

R.G. Williamson, P. Eng.,

Senior Project Engineer.

Investigation by: R.G. Irwin

B. Thomas

I N D E X

S U M M A R Y

1. Head Analysis

Representative samples were removed from C and F zone ore for analysis.

Element	C Zone	F Zone
Au (g/t)	39.1* (40.9)	7.77**(7.89)
Ag (g/t)	3.3	2.3
As (%)	7.71 (7.26)	10.9 (10.8)
Fe (%)	6.13	9.26
S (%)	3.39 (3.25)	5.05 (4.97)

* average of 32.9, 42.0, 42.4 g/t Au from three head samples

** average of 8.75, 4.97 and 9.60 g/t Au from three head samples

( ) average from testwork

XRF Semi-Quantitative Analysis

Element	C Zone	F Zone
Titanium	ND	T
Chromium	ND	FT
Manganese	T	FT
Iron	LM	M
Cobalt	ND	ND
Nickel	FT	FT
Copper	ND	FT
Zinc	FT	FT
Arsenic	MH	MH
Bismuth	ND	ND
Lead	ND	ND
Uranium	ND	ND
Thorium	ND	ND
Yttrium	FT	FT
Columbium	ND	ND
Molybdenum	ND	ND
Silver	ND	ND
Cadmium	ND	ND
Tin	ND	ND
Antimony	ND	ND

Code:

H - 10% plus

MH - 5-15%

M - 1-10%

LM - .5-5%

L - .1-1%

TL - .05-.5%

T - .01-.1%

FT - Less than .01%

ND - Not detected

Summary - Continued

2. Mineralogy

The mineralogy of the gold-arsenopyrite ores was described in a letter from Mr. Frank Nixon to Lakefield Research, dated December 18, 1981.

C Zone High grade gold mineralization was associated with arsenopyrite in quartz veins. Native gold was intergrown with masses of arsenopyrite grains and as free grains. A few gold inclusions were observed in arsenopyrite grains, which were strongly fractured.

F Zone The granite host rock was cut by arsenopyrite veins which were associated with chlorite alteration along fracture zones. The granite which had been strongly shattered consisted of coarse interlocking feldspars with lesser interstitial and fracture filling quartz. Very little visible gold was observed.

3. Gold Association

The gold association in both ores was determined by a sequential amalgamation and leaching procedure. Each ore was ground to approximately 50 and 80 percent minus 200 mesh. The ground pulp was amalgamated and cyanided to recover available gold. The cyanide residue was leached with HCl and cyanided to determine the gold associated with carbonates. The cyanide residue was leached with HCl and $SnCl_2$ and cyanided to determine the gold associated with iron and metal oxides. The cyanide residue was finally leached with aqua regia to determine the gold associated with sulphides. Gold in the residue from the aqua regia leach was associated with silicates.

The results are presented in Table No. 1. At a grind of about 80% minus 200 mesh a total of 94% of the gold in Sample C was available for recovery by amalgamation/cyanidation. Only 4% was associated with sulphides. At a similar grinding size on Sample F, a total of 88% of the gold was available for recovery by amalgamation/cyanidation. 10% of the gold was associated with sulphides.

Summary - Continued

3. Gold Association - Cont'd

At a coarser grind of about 50 % minus 200 mesh the amount of gold locked into a sulphide matrix increased to 17 % in Sample F.

Table No. 1 - Gold Association

Sample	Zone C		Zone F	
Grind % -200 mesh	46	77	47	82
Available by amalgamation	45	68	41	56
Available by cyanidation	42	26	39	32
Associated with carbonates	6	1	2	2
Associated with iron oxides etc.	1	<1	1	<1
Associated with sulphides	6	4	17	10
Associated with silicates	<1	1	<1	<1

4. Cyanidation

Cyanidation tests were conducted on both samples at three grinding sizes in bottle tests on rolls (1 g/L NaCN, 33 % solids, pH 10.5-11.5, 2 x 24 h). The results are presented in Table No. 2.

A total of 93 % of the gold in Sample C could be recovered by cyanidation at a primary grind of 70 % minus 200 mesh leaving a residue assaying 2.5 g/t Au. A finer grind to 98 % minus 200 mesh reduced the residue assay to 2.1 g/t Au.

A total of 80 % of the gold in Sample F could be recovered by cyanidation at a primary grind of 76 % minus 200 mesh leaving a residue assaying 1.6 g/t Au.

Summary - Continued

4. Cyanidation - Cont'd

Cyanide consumption ranged from 2.6 to 3.4 kg/t and reducing powers ranged from 200 to 260 mL 0.1 N $KMnO_4$/L pregnant solution.

Additional tests are being conducted to examine various methods of reducing the cyanide consumption.

Table No. 2 - Cyanidation of Ore

Test No.	Sample Zone	Grind % -200 mesh	Reagent Cons. NaCN kg/t	Reagent Cons. CaO kg/t	Gold Ext'n %	Residue Assay Au, g/t	Head Assay Au g/t	Reducing Power*	pH Range
27	C	40	1.5	1.2	90	3.70	37.1	120	10.3-11.4
28	C	70	2.9	1.2	93	2.47	36.8	200	10.3-11.1
29	C	98	3.4	1.5	94	2.06	36.0	220	10.3-11.2
30	F	45	1.3	1.2	73	1.72	6.32	122	10.1-11.5
31	F	76	2.8	2.0	80	1.57	7.76	218	10.1-11.2
32	F	99	3.0	2.1	80	1.37	6.76	259	10.0-11.1

*mL 0.1 N $KMnO_4$/L pregnant solution

5. Flotation

Flotation tests were conducted on both samples at two grinding sizes (approximately 80 and 98 % minus 200 mesh).

Sample C: A primary grind of 80 % minus 200 mesh produced a flotation tailing which represe ted 75 % of the feed weight and assayed 1.5 g/t Au. The rougher concentrate assayed 170 g/ Au, 26 % As, and 12 % S at 98 % gold recovery.

Increasing the grinding fineness to 98 % minus 200 mesh did not significantly reduce gold losses in the flotation tailing.

Summary - Continued

5. Flotation - Cont'd

Cleaning tests reduced the concentrate weight by about 50 % with a loss of about 6 % of the gold. The cleaner concentrate from Test 9 represented 13 % of the feed weight and assayed 306 g/t Au, 41 % As, and 19 % S at 92 % gold recovery.

Sample F: A primary grind of 80 % minus 200 mesh produced a flotation tailing which represented 65 % of the feed weight and assayed 0.5 g/t Au. The rougher concentrate assayed 19 g/t Au, 28 % As, and 13 % S at 96 % gold recovery.

Increasing the grinding fineness did not reduce gold loss in the flotation tailing.

Cleaning tests reduced the concentrate weight by about 50 % with a loss of about 10 % of the gold in the cleaner tailings. The cleaner concentrate from Test 10 represented 19 % of the feed weight and assayed 32 g/t Au, 40 % As, and 19 % S at 86 % gold recovery.

The flotation test conditions and results are contained in Table No. 3. Gold grade versus recovery cleaning curves are illustrated in Figures 1 and 2.

Table No. 3 - Flotation Test Conditions and Results

Conditions

Test No.	Sample	Grind %-200 mesh	Rougher Flotation			Regrind min	Cleaner Flotation			Cleaner Feed % -200 mesh
			AX350 g/t	AP208 g/t	Time min.		Stages	AX350 g/t	AP208 g/t	
5	C Zone	77	40	40	12	-	-	-	-	-
9	C Zone	77	40	40	12	-	3	5	5	92
11	C Zone	77	40	40	12	10	3	15	15	99
13	C Zone	97	70	70	15	-	3	10	10	99
15*	C Zone	-	40	40	12	-	3	5	5	-
7	F Zone	82	40	40	12	-	-	-	-	-
10	F Zone	82	40	40	12	-	3	5	5	92
12	F Zone	82	40	40	12	15	3	20	20	99
14	F Zone	97	70	70	15	-	3	10	10	99
16*	F Zone	-	40	40	12	-	3	5	5	-

*10 kg charge for concentrate production

Table No. 3 - Flotation Test Conditions and Results - Cont'd

Results

Test No.	Cleaner Concentrate							Rougher Concentrate							Rougher Tailing						
	Wgt. %	Assay %,g/t			% Dist.			Wgt. %	Assay %,g/t			% Dist.			Wgt. %	Assay %,g/t			% Dist.		
		Au	As	S	Au	As	S		Au	As	S	Au	As	S		Au	As	S	Au	As	S
5	-	-	-	-	-	-	-	27.4	140	25.9	11.3	98	96	97	72.6	1.27	0.39	0.13	2	4	3
9	13.3	306	40.8	19.2	92	77	78	25.4	170	26.0	12.0	98	93	96	74.6	1.46	0.63	0.29	3	7	7
11	9.1	362	39.8	18.7	86	50	53	31.7	118	21.5	9.6	98	95	95	68.3	1.32	0.55	0.26	2	5	6
13	12.0	324	40.2	19.1	93	67	70	36.3	113	18.8	8.7	98	94	95	63.8	1.27	0.69	0.27	2	6	5
15	11.7	303	43.5	19.8	89	69	71	21.3	179	32.2	14.4	96	93	94	78.7	1.90	0.68	0.26	4	7	6
7	-	-	-	-	-	-	-	36.0	20.8	28.5	13.0	97	95	95	64.0	0.42	0.93	0.38	4	6	5
10	19.2	31.5	39.6	18.9	86	72	74	34.7	19.3	28.4	13.3	96	93	94	65.3	0.47	1.10	0.44	5	7	6
12	12.8	51.9	40.7	18.9	75	48	49	40.2	21.0	25.4	11.5	95	94	95	59.8	0.78	1.07	0.44	5	6	5
14	16.7	38.9	37.9	18.9	84	60	63	46.8	15.9	21.0	10.1	96	93	94	53.2	0.54	1.37	0.57	4	7	6
16	17.5	29.8	39.9	19.0	76	64	65	30.5	20.8	32.2	15.2	92	89	91	69.5	0.77	1.69	0.67	8	11	9

FIGURE No 1

ORE ZONE C GOLD GRADE VS RECOVERY

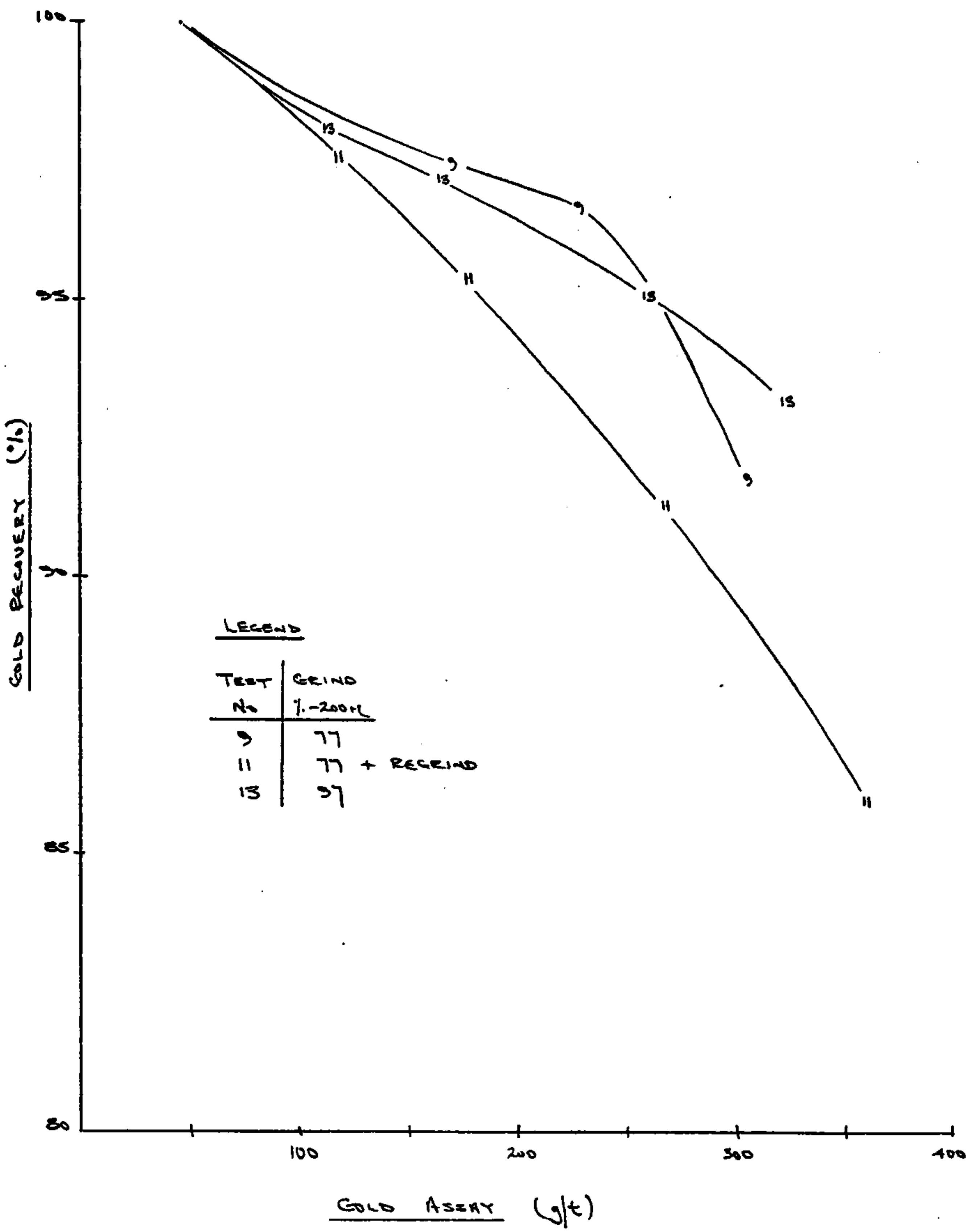
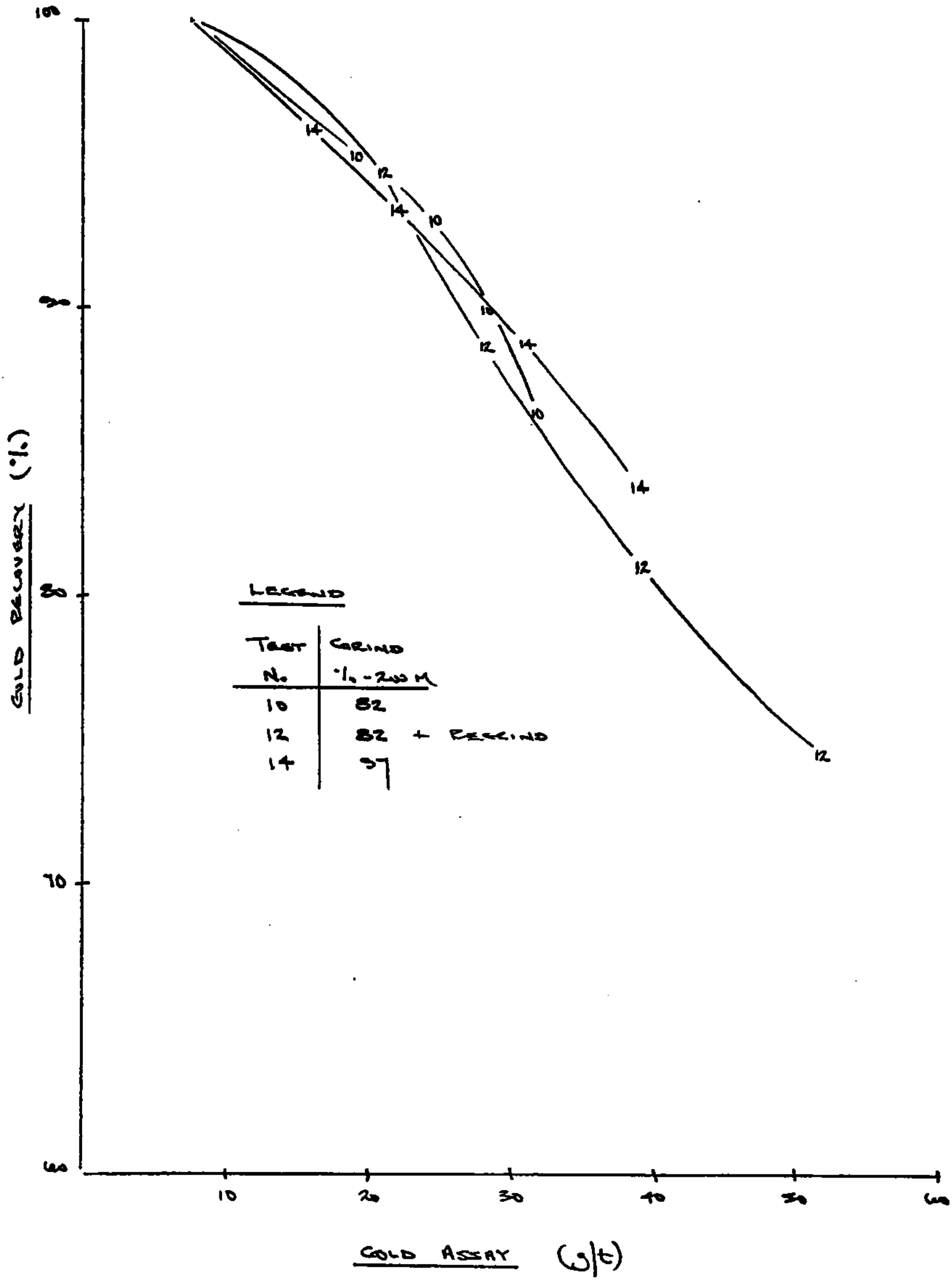


FIGURE No 2

ORE ZONE F   GOLD GRADE VS RECOVERY



Summary - Continued

6. Cyanidation of Flotation Products

Cyanidation tests were conducted on the cleaner concentrate, combined cleaner tailing, and rougher tailing from flotation tests on both samples C and F. The test conditions were 1 g/L NaCN, 33 % solids, pH 10.5-11.5, 2 x 24 h, in bottle test on rolls.

The results which are contained in Table No. 4 showed 85 % gold extraction from the cleaner concentrate, 90 % gold extraction from the rougher concentrate and 93 % overall gold extraction from Sample C.

Sample F produced 64 % gold extraction from the cleaner concentrate, 73 % gold extraction from the rougher concentrate and 76 % overall gold extraction.

Cyanide consumption was significantly lower than cyanide tests on both ground ores. This phenomenon will be examined in further tests.

Table No. 4 - Cyanidation of Flotation Products

Test No.	Sample Zone	Flotation Product	Reagent Cons.*		Gold Ext'n % Au		Residue Assay Au, g/t	Head Assay Au, g/t
			NaCN kg/t	CaO kg/t	Ind.	O'all		
17	C	Cleaner Conc.	0.07	0.07	96	85	13.2	303
19	C	Cleaner Tail.	0.09	0.14	76	5	6.2	29
6	C	Rougher Tail.	0.20	0.30	74	3	0.5	1.9
18	F	Cleaner Conc.	0.15	0.13	85	64	5.6	30
20	F	Cleaner Tail.	0.08	0.13	57	9	3.6	8.7
8	F	Rougher Tail.	0.20	0.30	42	3	0.4	0.8

*overall

Summary - Continued

7. Roasting and Cyanidation of Flotation Products

The cleaner concentrates and combined cleaner tailings from flotation tests on both Samples C and F were roasted in a muffle furnace in two stages at 575°C and 625°C to eliminate the arsenic and exfoliate the sulphides to expose the gold for recovery by cyanidation.

Efficient arsenic and sulphur elimination was achieved in the tests. Gold recovery from Sample C increased by 2 % from the cleaner concentrate and by 3 % from the rougher concentrate. This data confirmed the gold association testwork in Section 3 which showed approximately 4 % of the gold associated with sulphides.

Gold recovery from Sample F increased by 6 % from the cleaner concentrate and by 9 % from the rougher concentrate. This data also confirmed the gold association results in Section 3 which showed approximately 10 % gold association with sulphides at a primary grind of about 80 % minus 200 mesh.

The results are tabulated in Table No. 5.

Table No. 5 - Effect of Roasting in Cyanide Recovery

Test No.	Sample Zone	Flotation Product	Treatment	Reagent Cons.*		Gold Ext'n % Au		Residue Assay		
				NaCN kg/t	CaO kg/t	Ind.	O'all	Au g/t	As %	S %
17	C	Cleaner Conc.	As Rec'd	0.07	0.07	96	85	13.2	43.5	19.8
21	C	Cleaner Conc.	Roasted	0.10	0.19	97	87	15.0	1.1	<0.1
19	C	Cleaner Tail.	As Rec'd	0.09	0.14	76	5	6.2	18.4	7.7
23	C	Cleaner Tail.	Roasted	0.05	0.08	87	6	3.8	3.3	0.5
18	F	Cleaner Conc.	As Rec'd	0.15	0.13	85	64	5.6	39.9	19.0
22	F	Cleaner Conc.	Roasted	0.16	0.33	93	70	5.0	1.3	<0.1
20	F	Cleaner Tail.	As Rec'd	0.08	0.13	57	9	3.6	21.7	10.0
24	F	Cleaner Tail.	Roasted	0.16	0.25	70	12	3.3	3.3	0.2

*overall

RECOMMENDATIONS

Additional cyanidation tests are being conducted on Sample C to determine the cause and to examine methods of reducing cyanide consumption.

SAMPLE PREPARATION

On March 5, 1982, 2 drums of F Zone and 1 drum of C Zone gold-arsenopyrite ore were received from A/S Sulfidmalm.

The 2 drums of F Zone were combined, jaw and cone crushed to -10 mm and riffled to reject 3/4 to storage. The remaining 1/4 was roll crushed to -1.7 mm (10 mesh) and riffled into 15 x 2 kg and 1 x 10 kg test charges and a head sample for analysis. Three separate assay samples were prepared for gold analysis.

The drum of C Zone was jaw and cone crushed to -10 mm and riffled to reject 1/2 to storage. The remaining 1/2 was roll crushed to -1.7 mm (10 mesh) and riffled into 15 x 2 kg and 1 x 10 kg test charges and a head sample for analysis. Three separate assay samples were prepared for gold analysis.

Screen Analyses

Zone F - Head

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 10	0.8	0.8	99.2
14	15.3	16.1	83.9
20	15.5	31.6	68.4
28	14.1	45.7	54.3
35	10.7	56.4	43.6
48	9.5	65.9	34.1
65	7.1	73.0	27.0
100	5.8	78.8	21.2
150	5.1	83.9	16.1
200	3.9	87.8	12.2
270	3.3	91.1	8.9
400	2.4	93.5	6.5
- 400	6.5	100.0	-
Total	100.0	-	-

Sample Preparation - Continued

Screen Analyses - Cont'd

Zone C - Head

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 10	0.6	0.6	99.4
14	16.8	17.4	82.6
20	16.8	34.2	65.8
28	15.9	50.1	49.9
35	10.6	60.7	39.3
48	9.1	69.8	30.2
65	6.0	75.8	24.2
100	5.1	80.9	19.1
150	4.5	85.4	14.6
200	3.2	88.6	11.4
270	2.6	91.2	8.8
400	1.5	92.7	7.3
- 400	7.3	100.0	-
Total	100.0	-	-

INVENTORY

The following samples are on hand at Lakefield.

12 kg	-	10 mm C Zone
1 x 2 kg	-	1.7 mm (10 mesh) C Zone
100 kg	-	10 mm F Zone
3 x 2 kg	-	1.7 mm (10 mesh) F Zone
1 x 250 g	-	Cleaner Concentrate Test 15 C Zone
4 x 250 g	-	Cleaner Concentrate Test 16 F Zone
2 x 250 g	-	Combined Cleaner Tailing Test 16 F Zone

[illegible]
[illegible] —
Control the assay of the [illegible].

DETAILS OF TESTS

Test No. 1

Purpose: To determine the Au association of Zone C Sample.

Procedure: A 2 kg -10 mesh sample was ground and filtered. Two 500 gram samples were cut as opposite 1/8 th's of the filter cake. One 500 gram sample was used as a head sample for screen analysis.
The second 500 gram sample was amalgamated with 10 g of mercury for one hour at pH 10.5 with NaOH. The mercury was recovered by elutriation.
The amalgamation residue was cyanided in a 2.5 litre bottle for 24 hours. The pH was maintained at 11.5 with $Ca(OH)_2$ and the solution strength maintained at 1.0 g/L NaCN.
The cyanide residue was filtered washed 3 times with water and dried. The solution and residue were sampled and analysed for Au and Ag.
A 100 gram sample was riffled from the cyanide leach residue for further leaching tests.
This sample was leached in 200 mL of concentrated HCl for one hour at 100°C (slight boil) to dissolve carbonate(s). The pulp was filtered and the residue was water washed 3 times. The solution was assayed for Au. The acid leached residue was repulped with 200 mL of water and cyanided at pH 11 with 20 g/L NaCN for 1h, filtered, and washed 3 times with water.
The cyanide solution was assayed for Au.
To determine the Au in association with iron oxides, the leach residue was leached for 1 h at 100°C (slight boil) in concentrated HCl with stannous chloride. The pulp was filtered and washed 3 times with water. The residue was cyanided at pH 11.0 in 200 mL of 20 g/L NaCN, for 1 h, filtered and washed 3 times with water.
The leach residue was leached for 1 h at 100°C in 200 mL of aqua regia. The pulp was filtered and washed. The aqua regia leach was repeated once to ensure complete dissolution of the sulphides.
The pulp was filtered, washed and the solution and residue were assayed for Au.

Feed: 2000g minus 10 mesh Zone C ore.

Grind: 20 minutes at 66% solids in lab ball mill.

Conditions:

Amalgamation:	Feed	-	500 g ground ore
	% solids	-	33
	Time	-	1 h
	Mercury	-	10 g
Condition No. 1:	Feed	-	500 g amalgamation tailing
	% Solids	-	33
	Solution	-	pH 10.5 - 11.5 adjusted with $Ca(OH)_2$
		-	NaCN - 1 g/L

Test No. 1 - Continued

Reagent Balance

Time	Added, Grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
0-2	1.05	1.00	1.00	0.72	0.90	-	0.10	-	11.7	-
2-8	0.10	-	0.10	-	0.96	-	0.04	-	11.4	-
8-24	0.04	-	0.04	-	1.00	0.27	-	0.45	11.3	60
Total	1.19	1.00	1.14	0.72	1.00	0.27	0.14	0.45	-	-

Reagent Consumption (kg per metric ton of cyanide feed) NaCN: 0.28 CaO: 0.90

Final Solution Volumes: 2240 mL

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

HCl Leach:

Feed - NaCN No. 1 Leach Residue
% Solids - 33
Solution - Conc. HCl
Temp. - 100°C
Time - 1 h

Cyanidation No. 2:

Feed - HCl Leach Residue
% Solids - 33
Solution - pH 11.0 adjusted with NaOH
- NaCN 20 g/L
Time - 1 h

$HCl/SnCl_2$ Leach:

Feed - NaCN No. 2 Leach Residue
% Solids - 33
Solution - 200 mL Conc. HCl
- 20 mL 5% Sn Cl_2
Temp. - 100°C
Time - 1 h
Observations - No colour change with $SnCl_2$

Cyanidation No. 3:

Feed - $HCl/SnCl_2$ Leach Residue
% Solids - 33
Solution - pH 11.0 adjusted with NaOH
- NaCN 20 g/L
Time - 1 h

Aqua-Regia Leach:

Feed - NaCN No. 3 Leach Residue
% Solids - 33
Solution - 160 mL HCl + 40 mL HNO_3
Temp. - 100°C
Time - 1 h
Repeat - 1 time

Test No. 1 - Continued

Metallurgical Results

Amalgamation and Cyanidation No. 1

Product	Amount	Assays, mg/L, g/t		% Distribution	
		Au	Ag	Au	Ag
1. Hg Amalgam	10 g	-	-	44.6	-
2. 24 h Cyanide Solution	2240 mL	3.84	0.33	41.3	51.7
3. 24 h Cyanide Residue	490.4 g	5.97	1.40	14.1	48.3
Head (Calculated)	490.4 g	42.4	2.91	100.0	100.0

Overall Results

Product	Amount	Assays, mg/L, g/t	% Distribution
		Au	Ag
1. Amalgam	-	-	44.7
2. NaCN Leach No. 1	456.7 mL	3.84	41.4
3. HCl Leach Solution	480 mL	0.006	0.1
4. NaCN Leach No. 2	500 mL	0.49	5.8
5. $HCl/SnCl_2$ Leach	500 mL	0.002	0.0
6. NaCN Leach No. 3	510 mL	0.11	1.3
7. Aqua Regia	890 mL	0.30	6.3
8. Residue	70.0 g	0.22	0.4
Head (Calculated)	100.0 g	42.3	100.0

Screen Analysis - 20 Minutes/2 kg Zone C

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 35	0.3	0.3	99.7
48	2.1	2.4	97.6
65	7.3	9.7	90.3
100	14.2	23.9	76.1
150	16.8	40.7	59.3
200	13.7	54.4	45.6
270	10.9	65.3	34.7
400	7.7	73.0	27.0
- 400	27.0	100.0	-
Total	100.0	-	-

Test No. 2

Purpose: To determine the gold association of Zone F sample.

Procedure: As per test No. 1.

Feed: 2000 grams minus 10 mesh Zone F ore.

Grind: 20 minutes at 66% solids in lab ball mill.

Conditions: As per test No. 1

Reagent Balance - Cyanidation No. 1

Time	Added, Grams				Residual		Consumed		R.P.*
	Actual		Equivalent		Grams		Grams		
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO	
0-2	1.05	1.00	1.00	0.72	0.90	-	0.10	-	-
2-8	0.10	-	0.10	-	0.96	-	0.04	-	-
8-24	0.04	-	0.04	-	1.00	0.27	-	0.45	60
Total	1.19	1.00	1.14	0.72	1.00	0.27	0.14	0.45	-

Reagent Consumption (kg per metric ton of cyanide feed) NaCN: 0.28 CaO: 0.90

Final Solution Volumes: 2120 mL

* Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Metallurgical Results

Amalgamation and Cyanidation No. 1

Product	Amount	Assays, g/t, mg/L		% Distribution	
		Au	Ag	Au	Ag
1. Hg Amalgam	10 g	-	-	40.2	-
2. 24 h Cyanide Solution	2120 mL	0.76	0.14	37.6	28.8
3. Residue	494.1 g	1.92	1.50	22.2	71.2
Head (Calculated)	494.1 g	8.66	2.10	100.0	100.0

Test No. 2 - Continued

Metallurgical Results - Cont'd

Overall Results

Product	Amount	Assays, g/t, mg/L	% Distribution
		Au	Au
1. Amalgam	-	-	41.1
2. NaCN Leach No. 1	429 mL	0.76	38.3
3. HCl Leach Solution	480 mL	0.003	0.1
4. NaCN Leach No. 2	470 mL	0.041	2.2
5. $HCl/SnCl_2$ Leach	550 mL	0.002	0.1
6. NaCN Leach No. 3	540 mL	0.02	1.3
7. Aqua Regia	920 mL	0.15	16.2
8. Residue	66.4 g	0.09	0.7
Head (Calculated)	100.0 g	8.52	100.0

Screen Analysis

20 minutes/2 kg Zone F

Mesh Size (Tyler)	% Retained		% Passing
	Individual	Cumulative	Cumulative
+ 20	0.1	0.1	99.9
28	0.1	0.2	99.8
35	0.2	0.4	99.5
48	1.6	2.0	98.0
65	5.7	7.7	92.3
100	13.0	20.7	79.3
150	17.8	38.5	61.5
200	14.8	53.3	46.7
270	11.2	64.5	35.5
400	8.4	72.9	27.1
- 400	27.1	100.0	-
Total	100.0	-	-

Test No. 3

Purpose: To repeat test No. 1, but at a finer grind.

Procedure: As per test No. 1.

Feed: 2000 grams minus 10 mesh Zone C ore.

Grind: 40 minutes at 66% solids in lab ball mill.

Conditions: As per test No. 1.

Reagent Balance - Cyanidation No. 1

Time	Added, Grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
0-2	1.05	1.00	1.00	0.72	0.88	-	0.12	-	11.7	-
2-8	0.12	-	0.12	-	0.94	-	0.06	-	11.4	-
8-24	0.06	-	0.06	-	1.00	0.24	-	0.48	11.3	60
Total	1.23	1.00	1.18	0.72	1.00	0.24	0.18	0.48	-	-

Reagent Consumption (kg per metric ton of cyanide feed) NaCN: 0.36 CaO: 0.96

Final Solution Volumes: 2100 mL

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Test No. 3 - Continued

Metallurgical Results

Amalgamation & Cyanidation No. 1

Product	Amount	Assays, mg/L, g/t		% Distribution	
		Au	Ag	Au	Ag
1. Hg Amalgam	10 g	-	-	67.3	-
2. 24 h Cyanide Solution	2100 mL	3.25	0.29	25.4	48.4
3. Cyanide Residue	498.7 g	3.94	1.30	7.3	51.6
Head (Calculated)	498.7 g	53.9	2.53	100.0	100.0

Overall Results

Product	Amount	Assays, mg/L, g/t	% Distribution
		Au	Au
1. Amalgam	-	-	67.7
2. NaCN Leach No. 1	421 mL	3.25	25.6
3. HCl Leach Solution	470 mL	0.002	0.0
4. NaCN Leach No. 2	460 mL	0.16	1.4
5. HCl/$SnCl_2$ Leach	510 mL	<0.001	-
6. NaCN Leach No. 3	500 mL	0.04	0.4
7. Aqua Regia	880 mL	0.26	4.3
8. Residue	67.9 g	0.50	0.6
Head (Calculated)	100.0 g	53.5	100.0

Screen Analysis

40 minutes/2 kg Zone C

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	0.1	0.1	99.9
100	1.3	1.4	98.6
150	6.6	8.0	92.0
200	14.6	22.6	77.4
270	16.6	39.2	60.8
400	13.3	52.5	47.5
- 400	47.5	100.0	-
Total	100.0	-	-

Test No. 4

Purpose: To repeat test No. 2, but at a finer grind.

Procedure: As per test No. 1.

Feed: 2000 grams minus 10 mesh Zone F ore.

Grind: 40 minutes at 66% solids in lab ball mill.

Conditions: As per test No. 1.

Reagent Balance - Cyanidation No. 1

Time	Added, Grams				Residual		Consumed		R.P.*
	Actual		Equivalent		Grams		Grams		
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO	
0-2	1.05	1.00	1.00	0.72	0.88	-	0.12	-	-
2-8	0.12	-	0.12	-	0.91	-	0.09	-	-
8-24	0.09	-	0.09	-	1.00	0.30	-	0.42	60
Total	1.26	1.00	1.21	0.72	1.00	0.30	0.21	0.42	-

Reagent Consumption (kg per metric ton of cyanide feed) NaCN: 0.42 CaO: 0.84

Final Solution Volumes: 2060

* Reducing Power : mL 0.1 N $KMnO_4$/L of pregnant solution

Test No. 4 - Continued

Metallurgical Results

Amalgamation & Cyanidation No. 1

Product	Amount	Assays, mg/L, g/t		% Distribution	
		Au	Ag	Au	Ag
1. Amalgam	10 g	-	-	53.5	-
2. 24 h Cyanida Solution	2060 mL	0.71	0.13	30.0	25.4
3. Cyanide Residue	495.9 g	1.61	1.60	16.5	74.6
Head (Calculated)	495.9 g	9.80	2.1	100.0	100.0

Overall Results

Product	Amount	Assays, mg/L, g/t	% Distribution
		Au	Au
1. Amalgam	-	-	55.7
2. NaCN Leach No. 1	416 mL	0.71	31.6
3. HCl Leach Solution	440 mL	0.004	0.2
4. NaCN Leach No. 2	500 mL	0.027	1.5
5. $HCl/SnCl_2$ Leach	540 mL	<0.001	-
6. NaCN Leach No. 3	480 mL	0.012	0.6
7. Aqua Regia	940 mL	0.10	10.1
8. Residue	66.4 g	0.04	0.3
Head (Calculated)	100.0 g	9.34	100.0

Screen Analysis

40 minute Grind

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	0.1	0.1	99.9
100	0.8	0.9	99.1
150	4.8	5.7	94.3
200	11.9	17.6	82.4
270	17.4	35.0	65.0
400	14.6	49.6	50.4
- 400	50.4	100.0	-
Total	100.0	-	-

Test No. 5

Purpose: To examine the flotation response of C Zone ore.

Procedure: As below.

Feed: 2 kg minus 10 mesh C Zone

Grind: 40 minutes at 66% solids in ball mill.

Conditions:

Stage	Reagents Added, g/tonne			Time, minutes			pH
	AX350	AF208	DF250	Grind	Cond.	Froth	
Grind	-	-	-	40	-	-	-
Rougher 1	20	20	10	-	2	4	7.8
Rougher 2	10	10	5	-	1	4	-
Rougher 3	10	10	5	-	1	4	-

Stage	Rougher
Flotation Cell	1000 g D-1
Speed: r.p.m.	1800
% Solids	33

Observations:

The ground sample appeared liberated.
The arsenopyrite floated very well.
Strong first stage flotation.
The second and third stages floated coarse arsenopyrite and some middlings.

Test No. 5 - Continued

Metallurgical Results

Product	Weight %	Assays, %, g/tonne			% Distribution		
		Au	As	S	Au	As	S
1. Rougher Conc. 1	19.61	189.27	32.3	14.2	94.4	85.9	87.4
2. Rougher Conc. 2	4.31	22.30	12.8	5.25	2.4	7.5	7.1
3. Rougher Conc. 3	3.44	8.99	5.94	2.32	0.8	2.8	2.5
4. Rougher Tailing	72.64	1.27	0.39	0.13	2.3	3.8	3.0
Head (Calculated)	100.00	39.31	7.37	3.18	100.0	100.0	100.0

Calculated Grades and Recoveries

Products 1 & 2	23.92	159.2	28.8	15.6	96.8	93.4	94.5
Products 1 to 3	27.36	140.3	25.9	11.3	97.6	96.2	97.0

Test No. 6

Purpose: To cyanide the flotation tailing from test No. 5.

Procedure: The sample was pulped with water in a two litre bottle. NaCN and lime were added and the cyanidation was carried out on rolls in 1 x 24 hour stage. The pulp was filtered and the residue washed three times with water.

Feed: 500 g flotation tailing from test No. 5.

Solution Volume: 1000 mL Pulp Density 33% solids

Solution Composition: 1.0 g/L NaCN

pH Range: 10.5 - 11.5 with $Ca(OH)_2$

Grind: Nil

Reagent Balance

Time	Added, Grams				Residual		Consumed		pH		R.P.*
	Actual		Equivalent		Grams		Grams				
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO			
0-2	1.0	0.17	0.95	0.13	0.84	-	0.11	-	10.9	10.4	-
2-7	0.12	0.07	0.11	0.05	0.95	0.02	0	0.16	10.9	10.7	-
7-24	-	0.05	-	0.04	0.93	0.03	0.02	0.03	11.0	10.6	22.4
Total	1.12	0.29	1.06	0.22	0.93	0.03	0.13	0.19	-	-	-

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.26 CaO: 0.38

* Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Metallurgical Results

Product	Amount	Assays, mg/L, g/t	% Distribution
		Au	Au
1. Solution	2210 mL	0.22	74.2
2. Residue	497.2 g	0.34	25.8
Head (Calc.)	497.2 g	1.32	100.0

Comments: Efficient recovery of gold from rougher tailing.

Test No. 7

Purpose: To repeat test No. 5 on F Zone ore.

Procedure: As below.

Feed: 2 kg minus 10 mesh F Zone.

Grind: 40 minutes at 66% solids in ball mill.

Conditions:

Stage	Reagents Added, g/tonne			Time, minutes	
	AX350	AF208	DF250	Cond.	Froth
Grind	-	-	-	-	-
Rougher 1	20	20	10	2	4
Rougher 2	10	10	5	1	4
Rougher 3	10	10	5	1	4

Stage	Rougher
Flotation Cell	1000 g D-1
Speed: r.p.m.	1800
% Solids	23

Observations:

Sample appeared high grade arsenopyrite.
Some arsenopyrite present as pepper inclusions in silicates. These particles remained in the flotation tailing.

Metallurgical Results

Product	Weight	Assays, %, g/tonne			% Distribution		
	%	Au	As	S	Au	As	S
1. Rougher Conc. 1	22.40	30.53	35.8	16.5	88.1	73.9	75.3
2. Rougher Conc. 2	9.39	5.76	20.3	8.79	7.0	17.6	16.8
3. Rougher Conc. 3	4.18	2.74	7.82	3.45	1.5	3.0	2.9
4. Rougher Tailing	64.03	0.42	0.93	0.38	3.5	5.5	5.0
Head (Calculated)	100.00	7.76	10.8	4.91	100.0	100.0	100.0

Calculated Grades and Recoveries

Products 1 & 2	31.79	23.21	31.3	14.22	95.1	91.5	92.1
Products 1 to 3	35.97	20.83	28.5	12.97	96.6	94.5	95.0

Test No. 8

Purpose: To cyanide the flotation tailing from test No. 7.

Procedure: The sample was pulped with water in a two litre bottle. NaCN and lime were added and the cyanidation was carried out on rolls in 1 - 24 hour stage. The pulp was filtered and the residue washed three times with water.

Feed: 500 g flotation tailing from test No. 7.

Solution Volume: 1000 mL Pulp Density 33% solids

Solution Composition: 1.0 g/L NaCN

pH Range: 10.5 - 11.5 with $Ca(OH)_2$

Reagent Balance:

Time	Added, Grams				Residual		Consumed		pH		R.P.*
	Actual		Equivalent		Grams		Grams				
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO			
0-2	1.0	0.20	0.95	0.15	0.83	--	0.12	-	11.2	10.3	-
2-5	0.13	0.08	0.12	0.06	0.92	0.02	0.03	0.19	10.9	10.6	-
5-24	0.03	0.07	0.03	0.05	0.93	0.03	0.02	0.04	11.0	10.5	32.4
Total	1.16	0.35	1.10	0.26	0.93	0.03	0.17	0.23	-	-	-

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.34 CaO: 0.46

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Metallurgical Results

Product	Amount	Assays, mg/L, g/t	% Distribution
		Au	Au
1. Solution	2130 mL	0.041	42.2
2. Residue	496.7 g	0.24	57.8
Head (Calc.)	496.7 g	0.41	100.0

Test No. 9

Purpose: To examine the cleaning characteristics of ore sample C.

Procedure: As below.

Feed: 2 kg minus 10 mesh ore sample Zone C.

Grind: 40 minutes at 66% solids in lab ball mill.

Conditions:

Stage	Reagents Added, g/t			Time, minutes		
	AX350	AF208	DF250	Grind	Cond.	Froth
Grind	-	-	-	40	-	-
Rougher 1	20	20	10	-	2	4
Rougher 2	10	10	5	-	1	4
Rougher 3	10	10	5	-	1	4
1st Cleaner	-	-	-	-	1	5
	5	5	-	-	1	2
2nd Cleaner	-	-	-	-	1	5
3rd Cleaner	-	-	-	-	1	4

Stage	Rougher	Cleaner
Flotation Cell	1000 g D-1	500 g D-1
Speed: r.p.m.	1800	1200
% Solids	33	

Observations:

Rougher - as per test No. 5
- some sulphides present as small attachments on large gangue particles

Cleaners - most of the arsenopyrite floated rapidly. The first cleaner required additional collector to float large sulphide particles and sulphide middlings.
- water cleaning was used in later stages
- 2nd & 3rd cleaner tailings contained middlings and large arsenopyrite grains
- all cleaner tailings appeared high in slimes

Test No. 9 - Continued

Metallurgical Results

Product	Weight	Assays, %, g/tonne			% Distribution		
	%	Au	As	S	Au	As	S
1. Cleaner Concentrate	13.27	306.30	40.8	19.2	91.8	76.5	77.8
2. 3rd Cleaner Tailing	1.87	41.43	22.6	10.6	1.8	6.0	6.1
3. 2nd Cleaner Tailing	3.63	38.42	13.6	5.39	3.1	7.0	6.0
4. 1st Cleaner Tailing	6.65	5.63	4.18	1.74	0.8	3.9	3.5
5. Rougher Tailing	74.58	1.46	0.63	0.29	2.5	6.6	6.6
Head (Calculated)	100.00	44.28	7.06	3.27	100.0	100.0	100.0

Calculated Grades and Recoveries

Products 1 & 2	15.14	273.59	38.55	18.14	93.6	82.5	83.9
Products 1 to 3	18.77	228.11	33.73	15.67	96.7	89.5	89.9
Products 1 to 4	25.42	169.91	26.00	12.03	97.5	93.4	93.4
Products 2 to 4	12.15	20.94	9.83	4.19	5.7	16.9	15.6

Screen Analysis

Combined Cleaner Products

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 100	0.1	0.1	99.9
150	1.6	1.7	98.3
200	6.6	8.3	91.7
270	13.5	21.8	78.2
400	14.9	36.7	63.3
- 400	63.3	100.0	-
Total	100.0	-	-

Test No. 10

Purpose: To examine the cleaning characteristics of ore sample F.

Procedure: As below.

Feed: 2 kg minus 10 mesh ore sample Zone F.

Grind: 40 minutes at 66% solids in lab ball mill.

Conditions:

Stage	Reagents Added, g/tonne			Time, minutes		
	AX350	AF208	DF250	Grind	Cond.	Froth
Grind	-	-	-	40	-	-
Rougher 1	20	20	10	-	2	4
Rougher 2	10	10	5	-	1	4
Rougher 3	10	10	5	-	1	4
1st Cleaner	-	-	-	-	1	5
..	5	5	-	-	1	2
2nd Cleaner	-	-	-	-	1	5
3rd Cleaner	-	-	-	-	1	4

Stage	Rougher		Cleaner	
Flotation Cell	1000 g	D-1	500 g	D-1
Speed: r.p.m.	1800		1200	
% Solids	33			

Observations:

Flotation appeared as per test No. 9.
The silica & sulphide middlings present during cleaning consisted of smaller sulphide inclusions.

Test No. 10 - Continued

Metallurgical Results

Product	Weight %	Assays, %, g/tonne Au	As	S	% Distribution Au	As	S
1. Cleaner Concentrate	19.20	31.56	39.6	18.9	86.3	72.0	73.9
2. 3rd Cleaner Tailing	3.14	8.58	28.0	12.8	3.8	8.3	8.2
3. 2nd Cleaner Tailing	4.45	4.80	17.8	7.71	3.0	7.5	7.0
4. 1st Cleaner Tailing	7.94	2.20	7.20	3.18	2.5	5.4	5.1
5. Rougher Tailing	65.27	0.47	1.10	0.44	4.4	6.8	5.8
Head (Calculated)	100.00	7.02	10.6	4.91	100.0	100.0	100.0

Calculated Grades and Recoveries

Products 1 & 2	22.34	28.33	38.0	18.04	90.1	80.3	82.1
Products 1 to 3	26.79	24.42	34.6	16.33	93.1	87.8	89.1
Products 1 to 4	34.73	19.34	28.4	13.32	95.6	93.2	94.2
Products 2 to 4	15.53	4.24	14.4	6.42	9.3	21.2	20.3

Screen Analysis

Combined Cleaner Products

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 100	0.1	0.1	99.9
150	1.4	1.5	98.5
200	6.3	7.8	92.2
270	14.8	22.6	77.4
400	16.4	39.0	61.0
- 400	61.0	100.0	-
Total	100.0	-	-

Test No. 11

Purpose: To determine the effect of regrinding the rougher concentrate before cleaning.

Procedure: As below.

Feed: 2 kg minus 10 mesh ore sample Zone C.

Grind: 40 minutes at 66% solids in lab ball mill.

Conditions:

Stage	Reagents Added, g/tonne			Time, minutes		
	AX350	AF208	DF250	Grind	Cond.	Froth
Grind	-	-	-	40	-	-
Rougher 1	20	20	10	-	2	4
Rougher 2	10	10	5	-	1	4
Rougher 3	10	10	5	-	1	4
Regrind	-	-	-	10	-	-
1st Cleaner	5	5	5	-	1	3
	5	5	5	-	1	3
	5	5	5	-	1	3
2nd Cleaner	-	-	-	-	1	5
3rd Cleaner	-	-	-	-	1	4

Stage	Regrind
Flotation Cell	Ball Mill

Test No. 11 - Continued

Metallurgical Results

Product	Weight	Assays, %, g/tonne			% Distribution		
	%	Au	As	S	Au	As	S
1. Cleaner Concentrate	9.09	361.52	39.8	18.7	86.0	50.3	52.8
2. 3rd Cleaner Tailing	3.93	51.58	26.5	11.8	5.3	14.5	14.4
3. 2nd Cleaner Tailing	7.59	20.72	16.4	6.74	4.1	17.3	15.9
4. 1st Cleaner Tailing	11.07	7.55	8.25	3.33	2.2	12.7	11.4
5. Rougher Tailing	68.32	1.32	0.55	0.26	2.4	5.2	5.5
Head (Calculated)	100.00	38.2	7.19	3.22	100.0	100.0	100.0

Calculated Grades and Recoveries

Products 1 & 2	13.02	267.97	35.8	16.61	91.3	64.8	67.2
Products 1 to 3	20.61	176.91	28.6	12.98	95.4	82.1	83.1
Products 1 to 4	31.68	117.73	21.5	9.61	97.6	94.8	94.5
Products 2 to 4	22.59	19.63	14.2	5.95	11.6	44.5	41.7

Screen Analysis

Combined Cleaner Products

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 150	0.1	0.1	99.9
200	0.8	0.9	99.1
270	3.5	4.4	95.6
400	7.1	11.5	88.5
- 400	88.5	100.0	-
Total	100.0	-	-

Test No. 12

Purpose: To examine the effect of regrinding sample F before cleaning.

Procedure: As below.

Feed: 2 kg minus 10 mesh ore sample Zone F.

Grind: 40 minutes at 66% solids in lab ball mill.

Conditions:

Stage	Reagents Added, grams/tonne			Time, minutes			pH
	AX350	AF208	DF250	Grind	Cond.	Froth	
Grind	-	-	-	40	-	-	-
	-	-	-	-	-	-	7.9
Rougher 1	20	20	10	-	2	4	-
Rougher 2	10	10	5	-	1	4	-
Rougher 3	10	10	5	-	1	4	-
Regrind	-	-	-	15	-	-	-
1st Cleaner	10	10	5	-	1	5	-
	10	10	5	-	1	5	-
2nd Cleaner	-	-	-	-	1	5	-
3rd Cleaner	-	-	-	-	1	4	-

Observations:

Roughing appeared normal
1st cleaner tailing high in silicates with small sulphide inclusions.
3rd cleaner tailing contained concentrate free arsenopyrite.

Test No. 12 - Continued

Metallurgical Results

Product	Weight %	Assays, %, g/tonne Au	As	S	% Distribution Au	As	S
1. Cleaner Concentrate	12.82	51.86	40.7	18.9	74.5	48.0	49.4
2. 3rd Cleaner Tailing	5.73	10.29	30.7	14.1	6.6	16.2	16.5
3. 2nd Cleaner Tailing	9.63	7.07	19.9	8.80	7.6	17.6	17.3
4. 1st Cleaner Tailing	12.03	4.53	11.1	4.70	6.1	12.3	11.5
5. Rougher Tailing	59.79	0.78	1.07	0.44	5.2	5.9	5.3
Head (Calculated)	100.00	8.93	10.9	4.91	100.0	100.0	100.0

Calculated Grades and Recoveries

Products 1 & 2	18.55	39.0	37.6	17.4	81.1	64.2	65.9
Products 1 to 3	28.18	28.1	31.6	14.5	88.7	81.8	83.2
Products 1 to 4	40.21	21.0	25.4	11.5	94.8	94.1	94.7
Products 2 to 4	27.39	6.63	18.3	8.11	20.3	46.1	45.3

Screen Analysis

Combined Cleaner Products

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 150	0.1	0.1	99.9
200	0.7	0.8	99.2
270	3.0	3.8	96.2
400	7.7	11.5	88.5
- 400	88.5	100.0	-
Total	100.0	-	-

Test No. 13

Purpose: To repeat test No. 9, but increase primary grind.

Procedure: As below.

Feed: 2 kg minus 10 mesh ore sample Zone C.

Grind: 60 minutes at 66% solids.

Conditions:

Stage	Reagents Added, g/tonne			Time, minutes			pH
	AX350	AF208	DF250	Grind	Cond.	Froth	
Grind	-	-	-	60	-	-	-
Rougher 1	20	20	10	-	2	5	7.8
Rougher 2	15	15	5	-	1	5	-
Rougher 3	15	15	5	-	1	5	-
1st Cleaner	-	-	-	-	1	5	-
	10	10	5	-	1	5	-
2nd Cleaner	-	-	-	-	1	5	-
3rd Cleaner	-	-	-	-	1	4	-

Test No. 13 - Continued

Metallurgical Results

Product	Weight	Assays, %, g/tonne			% Distribution		
	%	Au	As	S	Au	As	S
1. Cleaner Concentrate	12.03	324.07	40.2	19.1	93.2	66.6	69.5
2. 3rd Cleaner Tailing	3.30	23.32	20.5	9.33	1.9	9.3	9.3
3. 2nd Cleaner Tailing	9.43	9.33	10.3	4.43	2.1	13.4	12.6
4. 1st Cleaner Tailing	11.49	3.22	2.95	0.99	0.9	4.7	3.4
5. Rougher Tailing	63.75	1.27	0.69	0.27	1.9	6.0	5.2
Head (Calculated)	100.00	41.8	7.26	3.31	100.0	100.0	100.0

Calculated Grades and Recoveries

Products 1 & 2	15.33	259.33	36.0	17.0	95.1	75.9	78.8
Products 1 to 3	24.76	164.12	26.2	12.2	97.2	89.3	91.4
Products 1 to 4	36.25	113.12	18.8	8.65	98.1	94.0	94.8
Products 2 to 4	24.22	8.34	8.20	3.47	4.9	27.4	25.3

Screen Analyses

Rougher Tailing

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 100	0.1	0.1	99.9
150	0.7	0.8	99.2
200	3.9	4.7	95.3
270	12.8	17.5	82.5
400	17.4	34.9	65.1
- 400	65.1	100.0	-
Total	100.0	-	-

Combined Cleaner Products

+ 150	0.1	0.1	99.9
200	1.3	1.4	98.6
270	4.8	6.2	93.8
400	8.8	15.0	85.0
- 400	85.0	100.0	-
Total	100.0	-	-

Test No. 13 - Continued

Screen Analysis of Ground Product

Product	Weight %	Cumulative, % Passing				
		100	150	200	270	400
Combined Cleaner Product	36.3	100.0	99.9	98.6	93.8	85.0
Rougher Tailing	63.7	99.9	99.2	95.3	82.5	65.1
Head (Calculated)	100.0	99.9	99.5	96.5	86.6	72.4

Test No. 14

Purpose: To repeat test No. 10, but increase grinding time.

Procedure: As below.

Feed: 2 kg minus 10 mesh ore sample Zone F.

Grind: 60 minutes at 66% solids.

Conditions:

Stage	Reagents Added, g/tonne			Time, minutes		
	AX350	AF208	DF250	Grind	Cond.	Froth
Grind	-	-	-	60	-	-
Rougher 1	20	20	10	-	2	5
Rougher 2	15	15	5	-	1	5
Rougher 3	15	15	5	-	1	5
1st Cleaner	-	-	-	-	1	5
	10	10	5	-	1	5
2nd Cleaner	-	-	-	-	1	5
3rd Cleaner	-	-	-	-	1	4

Metallurgical Results

Product	Weight	Assays, %, g/t			% Distribution		
	%	Au	As	S	Au	As	S
1. Cleaner Concentrate	16.67	38.9	37.9	18.9	83.8	59.7	62.6
2. 3rd Cleaner Tailing	5.66	6.86	23.9	10.9	5.0	12.8	12.3
3. 2nd Cleaner Tailing	10.01	3.63	14.4	6.46	4.7	13.6	12.9
4. 1st Cleaner Tailing	14.48	1.47	5.09	2.22	2.8	7.0	6.2
5. Rougher Tailing	53.18	0.54	1.37	0.57	3.7	6.9	6.0
Head (Calculated)	100.00	7.74	10.6	5.03	100.0	100.0	100.0

Calculated Grades and Recoveries

Products 1 & 2	22.33	30.78	34.4	16.9	88.8	72.5	74.9
Products 1 to 3	32.34	22.38	28.2	13.6	93.5	86.1	87.8
Products 1 to 4	46.82	15.91	21.0	10.1	96.3	93.1	94.0
Products 2 to 4	30.15	3.20	11.7	5.26	12.5	33.4	31.4

Test No. 14 - Continued

Screen Analyses

Rougher Tailing

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	0.1	0.1	99.9
100	0.1	0.2	99.8
150	0.5	0.7	99.3
200	3.3	4.0	96.0
270	11.3	15.3	84.7
400	17.4	32.7	67.3
- 400	67.3	100.0	-
Total	100.0	-	-

Combined Cleaner Products

+ 150	0.1	0.1	99.9
200	0.9	1.0	99.0
270	4.7	5.7	94.3
400	10.4	16.1	83.9
- 400	83.9	100.0	-
Total	100.0	-	-

Screen Analysis of Ground Product

Product	Weight %	Cumulative, % Passing 100	150	200	270	400
Combined Cleaner Product	46.8	100.0	99.9	99.0	94.3	83.9
Rougher Tailing	53.2	99.8	99.3	96.0	84.7	67.3
Head (Calculated)	100.0	99.9	99.5	97.4	89.2	75.1

Test No. 15

Purpose: To repeat test No. 9, but using a 10 kg charge.

Procedure: As below.

Feed: 10 kg of minus 10 mesh ore sample Zone C.

Grind: 40 minutes at 66% solids in the large ball mill.

Conditions:

Stage	Reagents Added, g/tonne			Time, minutes		
	AX350	AF208	DF250	Grind	Cond.	Froth
Grind	-	-	-	40	-	-
Rougher 1	20	20	10	-	2	4
Rougher 2	10	10	5	-	1	4
Rougher 3	10	10	5	-	1	4
1st Cleaner	-	-	-	-	1	5
	-	5	5	-	1	2
2nd Cleaner	-	-	-	-	1	5
3rd Cleaner	-	-	-	-	1	4

Stage	Rougher	Cleaners
Flotation Cell	Agitair	1000 g D-1
Speed: r.p.m.		1800

Test No. 15 - Continued

Metallurgical Results

Product	Weight	Assays, %, g/tonne			% Distribution		
	%	Au	As	S	Au	As	S
1. Cleaner Concentrate	11.72	302.59	43.5	19.8	89.4	68.9	71.1
2. 3rd Cleaner Tailing	1.34	72.24	34.2	15.6	2.4	6.2	6.4
3. 2nd Cleaner Tailing	3.23	38.00	28.6	11.4	3.1	12.5	11.3
4. 1st Cleaner Tailing	4.99	10.70	7.62	3.25	1.3	5.2	4.9
5. Rougher Tailing	78.72	1.90	0.68	0.26	3.8	7.2	6.3
Head (Calculated)	100.00	39.7	7.40	3.27	100.0	100.0	100.0

Calculated Grades and Recoveries

Products 1 & 2	13.06	278.95	42.5	19.4	91.8	75.1	77.5
Products 1 to 3	16.29	231.18	39.8	17.8	94.9	87.6	88.8
Products 1 to 4	21.28	179.48	32.2	14.4	96.2	92.8	93.7
Products 2 to 4	9.56	28.55	18.4	7.73	6.8	23.9	22.6

Test No. 16

Purpose: To repeat test No. 15, but on sample F.

Procedure: As below.

Feed: 10 kg of minus 10 mesh ore sample Zone F.

Grind: 40 minutes at 66% solids in the large ball mill.

Conditions:

Stage	Reagents Added, g/tonne			Time, minutes		
	AX350	AF208	DF250	Grind	Cond.	Froth
Grind	-	-	-	40	-	-
Rougher 1	20	20	10	-	1	4
Rougher 2	10	10	5	-	1	4
Rougher 3	10	10	5	-	1	4
1st Cleaner	-	-	-	-	1	5
	-	5	5	-	1	2
2nd Cleaner	-	-	-	-	1	5
3rd Cleaner	-	-	-	-	1	4

Stage	Rougher	Cleaners
Flotation Cell	Agitair	1000 g D-1
Speed: r.p.m.		1800

Test No. 16 - Continued

Metallurgical Results

Product	Weight %	Assays, %, g/tonne Au	As	S	% Distribution Au	As	S
1. Cleaner Concentrate	17.50	29.77	39.9	19.0	75.8	63.6	65.4
2. 3rd Cleaner Tailing	3.30	18.32	33.6	15.8	8.8	10.1	10.3
3. 2nd Cleaner Tailing	4.67	7.48	26.0	11.9	5.1	11.1	10.9
4. 1st Cleaner Tailing	4.99	3.50	9.87	4.33	2.5	4.5	4.2
5. Rougher Tailing	69.54	0.77	1.69	0.67	7.8	10.7	9.2
Head (Calculated)	100.00	6.87	11.0	5.08	100.0	100.0	100.0

Calculated Grades and Recoveries

Products 1 & 2	20.80	27.95	38.9	18.5	84.6	73.7	75.7
Products 1 to 3	25.47	24.20	36.5	17.3	89.7	84.8	86.6
Products 1 to 4	30.46	20.81	32.2	15.2	92.2	89.3	90.8
Products 2 to 4	12.96	8.71	21.7	9.98	16.4	25.7	25.4

Test No. 17

Purpose: To perform a standard cyanidation test on the cleaner concentrate from flotation test No. 15.

Procedure: The sample was pulped with water in a two litre bottle. NaCN and lime were added and the cyanidation was carried out on rolls in 1 x 24 hour stage.

Feed: 250 g of cleaner concentrate from test No. 15.

Solution Volume: 500 mL Pulp Density 33% solids

Solution Composition: 1.0 g/L NaCN

pH Range: 10.5 - 11.5 with $Ca(OH)_2$

Reagent Balance:

Time	Added, Grams				Residual		Consumed		pH	
	Actual		Equivalent		Grams		Grams			
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
0-3	0.53	0.25	0.50	0.19	0.40	0.05	0.10	0.14	11.6	11.5
3-8	0.11	0	0.10	0	0.48	0.05	0.02	0	11.5	11.5
8-24	0.02	0	0.02	0	0.48	0.04	0.02	0.01	11.5	11.2
Total	0.66	0.25	0.62	0.19	0.48	0.04	0.14	0.15	-	-

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.56 CaO: 0.60

Metallurgical Results

Product	Amount	Assays, mg/L, g/t	% Distribution
		Au	Au
Solution	1350 mL	52.8	95.6
Residue	247.2 g	13.17	4.4
Head (Calc.)	247.2 g	301.54	100.0

Test No. 18

Purpose: To perform a standard cyanidation test on the cleaner concentrate from flotation test No. 16.

Procedure: The sample was pulped with water in a two litre bottle. NaCN and lime were added and the cyanidation was carried out on rolls in one 24 hour stage.

Feed: 250 g of cleaner concentrate from test No. 16.

Solution Volume: 500 mL Pulp Density 33% solids

Solution Composition: 1.0 g/L NaCN

pH Range : 10.5 - 11.5 with $Ca(OH)_2$

Reagent Balance:

Time	Added, Grams				Residual		Consumed		pH	
	Actual		Equivalent		Grams		Grams			
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
0-3	0.53	0.25	0.50	0.19	0.43	0.04	0.07	0.15	11.5	11.3
3-8	0.07	0	0.07	0	0.46	0.03	0.04	0.01	11.3	11.1
8-24	0.04	0	0.04	0	0.40	0.01	0.10	0.02	11.1	10.6
Total	0.64	0.25	0.61	0.19	0.40	0.01	0.21	0.18	-	-

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.84 CaO: 0.72

Metallurgical Results

Product	Amount	Assays, mg/L, g/t	% Distribution
		Au	Au
Solution	1290 mL	6.15	85.0
Residue	248.0 g	5.63	15.0
Head (Calc.)	248.0 g	37.62	100.0

Test No. 19

Purpose: To perform a standard cyanidation test on the combined cleaner tailings from flotation test No. 15.

Procedure: The sample was pulped with water in a two litre bottle. NaCN and lime were added and the cyanidation was carried out on rolls in one 24 hour stage.

Feed: 250 g of combined cleaner tailings test No. 15.

Solution Volume: 500 mL Pulp Density 33% solids

Solution Composition: 1.0 g/L NaCN

pH Range: 10.5 - 11.5 with $Ca(OH)_2$

Reagent Balance:

Time	Added, Grams				Residual		Consumed		pH	
	Actual		Equivalent		Grams		Grams			
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
0-3	0.53	0.20	0.50	0.15	0.33	0	0.17	0.15	11.4	10.0
3-8	0.18	0.15	0.17	0.11	0.45	0	0.05	0.11	11.2	10.6
8-24	0.05	0.15	0.05	0.11	0.48	0.01	0.02	0.10	11.4	10.6
Total	0.76	0.50	0.72	0.37	0.48	0.01	0.24	0.36	-	-

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.96 CaO: 1.44

Metallurgical Results

Product	Amount	Assays, mg/L, g/t	% Distribution
		Au	Au
Solution	1350 mL	3.54	75.8
Residue	248.5 g	6.17	24.2
Head (Calc.)	248.5 g	25.39	100.0

Test No. 20

Purpose: To perform a standard cyanidation test on the combined cleaner tailings from flotation test No. 16.

Procedure: The sample was pulped with water in a two litre bottle. NaCN and lime were added and the cyanidation was carried out on rolls in one 24 hour stage.

Feed: 250 g of combined cleaner tailings test No. 16.

Solution Volume: 500 mL Pulp Density 33% solids

Solution Composition: 1.0 g/L NaCN

pH Range: 10.5 - 11.5 with $Ca(OH)_2$

Reagent Balance:

Time	Added, Grams				Residual		Consumed		pH	
	Actual		Equivalent		Grams		Grams			
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
0-3	0.53	0.20	0.50	0.15	0.39	0	0.11	0.15	11.5	10.3
3-8	0.12	0.15	0.11	0.11	0.48	0.02	0.02	0.09	11.5	11.1
8-24	0.02	0	0.02	0	0.48	0.01	0.02	0.01	11.1	10.6
Total	0.67	0.35	0.63	0.26	0.48	0.01	0.15	0.25	-	-

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.60 CaO: 1.00

Metallurgical Results

Product	Amount	Assays, mg/L, g/t	% Distribution
		Au	Au
Solution	1350 mL	0.88	57.2
Residue	248.7 g	3.57	42.8
Head (Calc.)	248.7 g	8.36	100.0

Test No. 21

Purpose:	To investigate the effect of roasting the cleaner concentrate from test No. 15 followed by acid leaching and cyanidation.
Roast Feed:	250 g of cleaner concentrate from flotation test No. 15, Zone C ore.
Roast Conditions:	A two stage roast was performed in a muffle furnace with constant rabbling. During the first stage the sample was maintained at a temperature of 575°C for 45 minutes. After the fuming had stopped the sample was brought up to 625°C for 30 mintues. The sample was air cooled and weighed.
Acid Leach Feed:	127.9 g of calcine.
Acid Leach Conditions:	The calcine was acid leached for one hour at 80°C, with a pulp density of 33% solids using 5 g/L of H_2SO_4. After leaching, the sample was filtered and displacement washed 3 times using water.
Cyanidation Feed:	127.9 g of acid leach residue.
Cyanidation Conditions:	A standard cyanidation was carried out on the acid leach residue. The residue was repulped to 33% solids in a two litre bottle, the cyanide strength was controlled at 1.0 g/L and the pH was maintained with lime at 10.5 - 11.5. After 24 hours the pulp was filtered and the residue washed 3 times with water.

Test No. 21 - Continued

Reagent Balance:

Time	Added, Grams				Residual		Consumed		pH	
	Actual		Equivalent		Grams		Grams			
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
0-2	0.27	0.13	0.26	0.10	0.18	0	0.08	0.10	11.6	9.8
2-8	0.08	0.10	0.08	0.08	0.23	0.01	0.03	0.07	11.2	10.8
8-24	0.03	0.05	0.03	0.04	0.26	0.01	0	0.04	11.4	10.8
Total	0.38	0.36	0.37	0.22	0.26	0.01	0.11	0.21	-	-

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.86 CaO: 1.64

Metallurgical Results

Product	Amount	Assays, %, mg/L, g/t			% Distribution
		Au	As	S	Au
1. Acid Leach Solution	860 mL	-	-	-	-
2. Cyanide Solution	1000 mL	67.0	-	-	97.3
3. Residue	125.6 g	14.95	1.13	<0.05	2.7
Head (Calculated)	250.0 g	275.5	-	-	100.0

Test No. 22

Purpose: To repeat test No. 21, but on the cleaner concentrate from test No. 16.

Roast Feed: 250 g of cleaner concentrate from flotation test No. 16, Zone F ore.

Roast Conditions: As for test No. 21.

Acid Leach Feed: 132.2 g of calcine.

Acid Leach Conditions: As for test No. 21.

Cyanidation Feed: 132.2 g of acid leach residue.

Cyanidation Conditions: As for test No. 21.

Reagent Balance:

Time	Added, Grams				Residual		Consumed		pH	
	Actual		Equivalent		Grams		Grams			
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
0-2	0.28	0.13	0.27	0.10	0.15	0	0.12	0.10	11.1	9.4
2-8	0.13	0.15	0.12	0.11	0.27	0	0	0.11	11.1	10.5
8-24	0	0.05	0	0.04	0.27	0	0	0.04	11.1	10.5
Total	0.41	0.33	0.39	0.25	0.27	0	0.12	0.25	-	-

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.91 CaO: 1.89

Metallurgical Results

Product	Amount	Assays, %, mg/L, g/t			% Distribution
		Au	Au	S	Au
1. Acid Leach Solution	865 mL	-	-	-	-
2. Cyanide Solution	1070 mL	7.80	-	-	92.8
3. Residue	130.3 g	4.97	1.30	<0.05	7.2
Head (Calculated)	250.0 g	36.00	-	-	100.0

Test No. 23

Purpose: To investigate the effect of roasting a sample of combined cleaner tailings from test No. 15 followed by acid leaching and cyanidation.

Roast Feed: 250 g of combined cleaner tailing from flotation test No. 15, Zone C ore.

Roast Conditions: As for test No. 21.

Acid Leach Feed: 210.0 g of calcine.

Acid Leach Conditions: As for test No. 21.

Cyanidation Feed: 210.0 g of acid leach residue.

Cyanidation Conditions: As for test No. 21.

Reagent Balance:

Time	Added, Grams				Residual		Consumed		pH	
	Actual		Equivalent		Grams		Grams			
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
0-2	0.45	0.15	0.43	0.11	0.35	0.01	0.08	0.10	11.4	10.6
2-8	0.08	0.10	0.08	0.08	0.39	0.03	0.04	0.06	11.5	11.3
8-24	0.04	0	0.04	0	0.43	0.02	0	0.01	11.3	10.8
Total	0.57	0.25	0.53	0.19	0.43	0.02	0.12	0.17	-	-

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.57 CaO: 0.81

Metallurgical Results

Product	Amount	Assays, %, mg/L, g/t			% Distribution
		Au	As	S	Au
1. Acid Leach Solution	990 mL	-	-	-	-
2. Cyanide Solution	1135 mL	4.63	-	-	86.9
3. Residue	204.6 g	3.84	3.34	0.53	13.1
Head (Calculated)	250.0 g	24.2	-	-	100.0

Test No. 24

Purpose: To repeat test No. 23, but on the combined cleaner tailings from test No. 16.

Roast Feed: 250 g of combined cleaner tailing from flotation test No. 16, Zone F ore.

Roast Conditions: As for test No. 21.

Acid Leach Feed: 196.7 g of clacine.

Acid Leach Conditions: As for test No. 21.

Cyanidation Feed: 196.7 g of acid leach residue.

Cyanidation Conditions: As for test No. 21.

Reagent Balance:

Time	Added, Grams				Residual		Consumed		pH	
	Actual		Equivalent		Grams		Grams			
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
0-2	0.42	0.15	0.40	0.11	0.16	0	0.24	0.11	11.1	9.1
2-8	0.25	0.25	0.24	0.19	0.40	0	0	0.19	11.2	10.1
8-24	0	0.10	0	0.08	0.40	0	0	0.08	11.0	10.3
Total	0.67	0.50	0.64	0.38	0.40	0	0.24	0.38	-	-

Reagent Consumption (kg/t of cyanide feed) NaCN: 1.22 CaO: 1.93

Metallurgical Results

Product	Amount	Assays, %, mg/L, g/t			% Distribution
		Au	As	S	Au
1. Acid Leach Solution	1000 mL	-	-	-	-
2. Cyanide Solution	1095 mL	1.37	-	-	70.4
3. Residue	192.0 g	3.26	3.27	0.18	29.6
Head (Calculated)	250.0 g	8.52	-	-	100.0

Test No. 25

Purpose: To cyanide the flotation tailing from Test 13.

Procedure: The sample was pulped with water in a 2 liter bottle. NaCN and lime were added and the cyanidation was carried out on rolls in one 24 hour stage. The pulp was filtered and the residue washed three times with water.

Feed: 500 g flotation tailing from Test 13.

Solution Volume: 1000 mL Pulp Density 33 % solids

Solution Composition: 1.0 gpL NaCN

pH Range: 10.5-11.5 with $Ca(OH)_2$

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
0-2	1.06	0.20	1.00	0.15	1.00	0.00	0.00	0.15	11.0-10.3	-
2-9	0.00	0.10	0.00	0.08	0.90	0.00	0.10	0.08	10.8-10.4	-
9-20	0.11	0.20	0.10	0.15	1.00	0.05	0.00	0.10	11.3-10.9	-
20-24	0.00	0.00	0.00	0.00	1.00	0.04	0.00	0.01	10.9-10.8	46
Total	1.17	0.50	1.10	0.38	1.00	0.04	0.10	0.34	-	-

Reagent Consumption (kg/tonne of cyanide feed) NaCN: 0.20 CaO : 0.68

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Cyanide Preg. + Wash	2280 mL	0.20	74.6
24 h Residue	498.3 g	0.31	25.4
Head (Calculated)	500.0 g	1.22	100.0

Comments: The results were similar to Test 6 at a primary grind of 77 % -200 mesh.

Test No. 26

Purpose: To cyanide the flotation tailing from Test 14.

Procedure: Same as Test 25.

Feed: 500 g flotation tailing from Test 14.

Solution Volume: 1000 mL Pulp Density 33 % solids

Solution Composition: 1.0 gpL NaCN

pH Range: 10.5-11.5 with $Ca(OH)_2$

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
0-2	1.06	0.20	1.00	0.15	0.90	0.00	0.10	0.15	10.9-10.2	-
2-9	0.11	0.10	0.10	0.08	0.90	0.00	0.10	0.08	10.6-10.3	-
9-20	0.11	0.20	0.10	0.15	1.00	0.05	0.00	0.10	11.2-10.9	-
20-24	0.00	0.00	0.00	0.00	1.00	0.04	0.00	0.01	10.9-10.8	85
Total	1.28	0.50	1.20	0.38	1.00	0.04	0.20	0.34	-	-

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.40 CaO: 0.68

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Cyanide Preg. + Wash	2130 mL	0.051	44.7
24 h Residue	498.7 g	0.27	55.3
Head (Calculated)	500.0 g	0.49	100.0

Comments: The results were similar to Test 8 at a primary grind of 82 % -200 mesh.

Test No. 27

Purpose: To investigate the cyanidation response of Sample C.

Procedure: The sample was ground, filtered and pulped with water in a 2 liter bottle. NaCN and lime were added and the cyanidation was carried out on rolls in two 24 hour stages with the solution being changed after each stage. Between each stage the pulp was filtered and the residue washed three times with water. The residue was then repulped with fresh cyanide solution and the test continued.

Feed: 500 g minus 10 mesh Sample C.

Solution Volume: 1000 mL Pulp Density 33 % solids

Solution Composition: 1.0 gpL NaCN

pH Range: 10.5-11.5 with $Ca(OH)_2$

Grind: 5 minutes at 66 % solids in the lab rod mill.

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
1st Stage										
0-2	1.06	0.30	1.00	0.23	0.65	0.01	0.35	0.22	11.4-10.6	-
2-5	0.37	0.20	0.35	0.15	1.00	0.05	0.00	0.11	11.4-11.0	-
5-20	0.00	0.00	0.00	0.00	0.95	0.01	0.05	0.04	11.0-10.3	-
20-24	0.05	0.10	0.05	0.08	0.80	0.04	0.20	0.05	11.1-10.8	120
2nd Stage										
24-30	1.06	0.30	1.00	0.23	1.90	0.05	0.10	0.18	11.6-10.9	-
30-48	0.11	0.00	0.10	0.00	0.95	0.03	0.05	0.02	10.9-10.0	72
Total	2.65	0.90	2.50	0.69	1.75	0.07	0.75	0.62	-	-

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Reagent Consumption (kg/tonne of cyanide feed) NaCN: 1.50 CaO: 1.24

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Cyanide Preg. + Wash	2000 mL	8.15	87.8
48 h Cyanide Preg. + Wash	2000 mL	0.24	2.6
48 h Residue	484.3 g	3.70	9.6
Head (Calculated)	500.0 g	37.14	100.0

Test No. 27 - Continued

Screen Analysis - 48 h Cyanide Residue

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 35	1.3	1.3	98.7
48	6.4	7.7	92.3
65	10.6	18.3	81.7
100	12.9	31.2	68.8
150	15.8	47.0	53.0
200	13.2	60.2	39.8
270	9.8	70.0	30.0
400	6.7	76.7	23.3
- 400	23.3	100.0	-
Total	100.0	-	-

Test No. 28

Purpose: To repeat Test 27 but at a finer grind.

Procedure: Same as Test 27.

Feed: 500 g minus 10 mesh Sample C.

Solution Volume: 1000 mL Pulp Density 33 % solids

Solution Composition: 1.0 gpL NaCN

pH Range: 10.5-11.5 with $Ca(OH)_2$

Grind: 10 minutes at 66 % solids in the lab rod mill.

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
1st Stage										
0-2	1.06	0.30	1.00	0.23	0.15	0.05	0.85	0.18	11.5-11.0	-
2-5	0.89	0.00	0.85	0.00	0.85	0.05	0.15	0.00	11.1-10.9	-
5-20	0.26	0.00	0.25	0.00	0.85	0.01	0.25	0.04	10.9-10.3	-
20-24	0.16	0.10	0.15	0.08	1.00	0.04	0.00	0.05	10.9-10.5	200
2nd Stage										
24-30	1.06	0.30	1.00	0.23	0.85	0.03	0.15	0.20	11.6-10.5	-
30-48	0.16	0.20	0.15	0.15	0.95	0.03	0.05	0.15	11.1-10.1	88
Total	3.59	0.90	3.40	0.69	1.95	0.07	1.45	0.62	-	-

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Reagent Consumption (kg per tonne of cyanide feed) NaCN: 2.90 CaO: 1.24

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Cyanide Preg. + Wash	2000 mL	8.55	93.1
48 h Cyanide Preg. + Wash	2000 mL	0.04	0.4
48 h Residue	484.0 g	2.47	6.5
Head (Calculated)	500.0 g	36.76	100.0

Test No. 28 - Continued

Screen Analysis - 48 h Cyanide Residue

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	0.2	0.2	99.8
100	1.1	1.3	98.7
150	8.7	10.0	90.0
200	19.9	29.9	70.1
270	18.3	48.2	51.8
400	11.9	60.1	39.9
- 400	39.9	100.0	-
Total	100.0	-	-

Test No. 29

Purpose: To repeat Test 28 but at a finer grind.

Procedure: Same as Test 27.

Feed: 500 g minus 10 mesh Sample C.

Solution Volume: 1000 mL Pulp Density 33 % solids

Solution Composition: 1.0 gpL NaCN

pH Range: 10.5-11.5 with $Ca(OH)_2$

Grind: 20 minutes at 66 % solids in the lab rod mill.

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
1st Stage										
0-2	1.06	0.30	1.00	0.23	0.10	0.04	0.90	0.19	11.3-10.6	-
2-5	0.95	0.20	0.90	0.15	0.80	0.03	0.20	0.16	11.2-10.9	-
5-20	0.32	0.00	0.30	0.00	0.85	0.01	0.25	0.02	10.9-10.3	-
20-24	0.16	0.15	0.15	0.11	0.95	0.07	0.05	0.05	10.9-10.6	220
2nd Stage										
24-30	1.06	0.30	1.00	0.23	0.80	0.01	0.20	0.22	11.6-10.4	-
30-48	0.21	0.20	0.20	0.15	0.90	0.03	0.10	0.13	11.0- 9.9	96
Total	3.76	1.15	3.55	0.87	1.85	0.10	1.70	0.77	-	-

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Reagent Consumption (kg per tonne of cyanide feed) NaCN: 3.40 CaO: 1.54

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Cyanide Preg. + Wash	2040 mL	8.35	93.8
48 h Cyanide Preg. + Wash	2000 mL	0.04	0.5
48 h Residue	504.0 g	2.06	5.7
Head (Calculated)	504.0 g	36.01	100.0

Test No. 29 - Continued

Screen Analysis - 48 h Cyanide Residue

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 100	0.1	0.1	99.9
150	0.2	0.3	99.7
200	1.4	1.7	98.3
270	9.2	10.9	89.1
400	18.6	29.5	70.5
- 400	70.5	100.0	-
Total	100.0	-	-

Test No. 30

Purpose: To investigate the cyanidation response of Sample F.

Procedure: Same as Test 27.

Feed: 500 g minus 10 mesh ore Sample F.

Solution Volume: 1000 mL Pulp Density 33 % solids

Solution Composition: 1.0 gpL NaCN

pH Range: 10.5-11.5 with $Ca(OH)_2$

Grind: 5 minutes at 66 % solids in the lab rod mill.

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
1st Stage										
0-2	1.06	0.30	1.00	0.23	0.75	0.02	0.25	0.21	11.4-10.7	-
2-5	0.26	0.20	0.25	0.15	0.95	0.08	0.05	0.09	11.5-11.2	-
5-20	0.05	0.00	0.05	0.00	0.90	0.04	0.10	0.04	11.2-10.6	-
20-24	0.11	0.10	0.10	0.08	0.95	0.06	0.05	0.06	11.2-10.9	122
2nd Stage										
24-30	1.06	0.30	1.00	0.23	1.85	0.07	0.15	0.16	11.6-10.9	-
30-48	0.16	0.00	0.15	0.00	0.95	0.03	0.05	0.04	10.9-10.1	64
Total	2.70	0.90	2.55	0.69	1.90	0.09	0.65	0.60	-	-

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Reagent Consumption (kg/tonne of cyanide feed) NaCN: 1.30 CaO: 1.20

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Cyanide Preg. + Wash	2040 mL	1.11	71.5
48 h Cyanide Preg. + Wash	2000 mL	0.02	1.3
48 h Residue	498.4 g	1.72	27.2
Head (Calculated)	500.0 g	6.32	100.0

Test No. 30 - Continued

Screen Analysis - 48 h Cyanide Residue

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	5.5	5.5	94.5
100	13.1	18.6	81.4
150	20.7	39.3	60.7
200	15.7	55.0	45.0
270	11.8	66.8	33.2
400	8.0	74.8	25.2
- 400	25.2	100.0	-
Total	100.0	-	-

Test No. 31

Purpose: To repeat Test 30 but at a finer grind.

Procedure: Same as Test 27.

Feed: 500 g minus 10 mesh ore.

Solution Volume: 1000 mL Pulp Density 33 % solids

Solution Composition: 1.0 g/L NaCN

pH Range: 10.5-11.5 with $Ca(OH)_2$

Grind: 10 minutes at 66 % solids in the lab rod mill.

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
1st Stage										
0-2	1.06	0.30	1.00	0.23	0.15	0.07	0.85	0.16	11.4-10.7	-
2-6	0.89	0.20	0.85	0.15	0.85	0.10	0.15	0.12	11.2-10.8	-
6-24	0.26	0.10	0.25	0.08	0.91	0.08	0.19	0.10	11.1-10.2	218
2nd Stage										
24-29	1.06	0.30	1.00	0.23	0.85	0.03	0.15	0.20	10.8-10.1	-
29-34	0.16	0.20	0.15	0.15	0.95	0.07	0.05	0.11	10.9-10.2	-
34-45	0.05	0.20	0.05	0.15	1.00	0.05	0.00	0.17	10.8-10.3	-
45-48	0.00	0.20	0.00	0.15	1.00	0.08	0.00	0.12	10.7-10.5	80
Total	3.48	1.50	3.30	1.14	1.91	0.16	1.39	0.98	-	-

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Reagent Consumption (kg/tonne of cyanide feed) NaCN: 2.78 CaO: 1.96

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Solution	2020 mL	1.39	72.2
48 h Solution	2000 mL	0.15	7.7
Residue	501.0 g	1.57	20.1
Head (Calc.)	501.0 g	7.76	100.0

Test No. 31 - Continued

Screen Analysis - 48 h Cyanide Residue

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	0.3	0.3	99.7
100	0.6	0.9	99.1
150	6.0	6.9	93.1
200	17.0	23.9	76.1
270	20.1	44.0	56.0
400	14.0	58.0	42.0
- 400	42.0	100.0	-
Total	100.0	-	-

Test No. 32

Purpose: To repeat Test 31, but at a finer grind.

Procedure: Same as Test 27.

Feed: 500 g minus 10 mesh ore.

Solution Volume: 1000 mL Pulp Density 33 % solids

Solution Composition: 1.0 gpl NaCN

pH Range: 10.5-11.5 with $Ca(OH)_2$

Grind: 20 minutes at 66 % solids in the lab rod mill.

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
1st Stage										
0-2	1.06	0.30	1.00	0.23	0.10	0.07	0.90	0.16	11.5-10.6	-
2-6	0.95	0.20	0.90	0.15	0.85	0.09	0.15	0.13	11.1-10.6	-
6-24	0.26	0.20	0.25	0.15	0.91	0.08	0.19	0.16	11.0-10.0	259
2nd Stage										
24-29	1.06	0.30	1.00	0.23	0.85	0.03	0.15	0.20	10.8-10.0	-
29-34	0.16	0.20	0.15	0.15	0.90	0.06	0.10	0.12	10.9-10.4	-
34-45	0.11	0.20	0.10	0.15	1.00	0.05	0.00	0.16	10.8-10.2	-
45-48	0.00	0.20	0.00	0.15	1.00	0.08	0.00	0.12	10.7-10.5	112
Total	3.60	1.60	3.40	1.21	1.91	0.16	1.49	1.05	-	-

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Reagent Consumption (kg/tonne of cyanide feed) NaCN: 2.98 CaO: 2.10

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Solution	2020 mL	1.24	73.3
48 h Solution	2000 mL	0.11	6.5
Residue	504.2 g	1.37	20.2
Head (Calc.)	504.2 g	6.76	100.0

Test No. 32 - Continued

Screen Analysis - 48 h Cyanide Residue

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 100	0.1	0.1	99.9
150	0.1	0.2	99.8
200	0.4	0.6	99.4
270	5.2	5.8	94.2
400	15.1	20.9	79.1
- 400	79.1	100.0	..
Total	100.0	-	-

LAKEFIELD RESEARCH OF CANADA LIMITED
Lakefield, Ontario
April 23 , 1982 / sem, lmn

An Investigation of

THE RECOVERY OF GOLD

from samples

submitted by

A/S SULFIDMALM

Progress Report No.3

Project No. L.R. 2570

Note:

This report refers to the samples as received.

LAKEFIELD RESEARCH OF CANADA LIMITED
Lakefield, Ontario
April 12, 1983

I N T R O D U C T I O N

In a telex dated February 17, 1983, Mr. Frank Nixon of A/S Sulfidmalm requested testwork on F and C Zone samples to investigate a procedure for assessing an average ore grade. This procedure involved grinding and gravity concentration of the gold.

LAKEFIELD RESEARCH OF CANADA LIMITED

D. M. Wyslouzil

D.M. Wyslouzil, P. Eng.,

Manager.

I Underhill

I. Underhill,

Project Engineer.

Experimental Work by: T.M. Jessup

S U M M A R Y

1. Head Analysis

Duplicate head samples were removed from Zone C and Zone F composites and assayed in duplicate with the following results:

Sample	Assay, g/t Au Zone F	Zone C
Head Sample A	29.1	41.8
	12.6	27.6
Head Sample B	9.59	31.9
	7.20	30.5
Average	14.6	33.0
Average (from Progress Report No.1)	7.77	39.1

2. Zone F

Two tests were conducted to investigate the effect of fineness of grind on the liberation of gold. In both tests, the ground pulp was passed over a Deister concentrating table. The table concentrate was amalgamated for 2 hours in a bottle on rolls. The results are summarized below:

Table No. 1 - Effect of Fineness of Grind

Test No.	% -200 Mesh	Product	Weight %	Assay g/t Au	% Distribution Au
42	46	Amalgam	-	6.46*	8.6
		Amalgam Tailing	4.23	83.9	47.1
		Table Tailing	95.77	3.48	44.3
		Head (calc.)	100.00	7.86	100.0
43	87	Amalgam	-	10.73*	14.8
		Amalgam Tailing	2.11	139.	40.4
		Table Tailing	97.89	3.32	44.8
		Head (calc.)	100.00	7.41	100.0

* mg

Summary - Continued

2. Zone F - Cont'd

The table tailing assay is the average of assays performed on two samples of the tailing. The individual assays on this product were as follows:

Test 42: 4.14, 3.26, 4.03, 2.50 g/t Au

Test 43: 3.32, 3.32 g/t Au

These results would suggest that free gold was present in the tailing of Test No. 42. The finer grind was used in all subsequent testwork.

Due to the high arsenopyrite content in the gravity concentrate, a coating formed on the mercury inhibiting the amalgamation and preventing coalescence. A small xanthate addition was made to overcome the beading of the mercury. This procedure had limited success.

The amalgamation tailing from Test No. 43 was retreated with 20 g mercury, 1 g NaOH and 1 g white lead. The amalgam was easily recovered and was clean. It is thought that the lead precipitated soluble sulphides which were interferring with the amalgamation. The amalgam contained an additional 7.4% of the gold in the original feed.

Based on the results of these two tests, the remaining Zone F composite was treated in 10 kg batches. Each charge was ground to 87% minus 200 mesh and passed over the Deister concentrating table. Two samples of the tailing were taken directly from the table. The table concentrates were amalgamated with mercury, sodium hydroxide and white lead. The results of these tests are presented in Table No. 2.

Summary - Continued

2. Zone F - Cont'd

Table No. 2 - Zone F

Test No.	Amalgam % Dist. Au	Table Tailing g/t Au	Table Tailing % Dist. Au	Head (calc.) g/t Au
45-A	17.1	3.92	48.8	7.92
B	6.1	3.17	44.3	6.98
C	14.5	3.62	47.6	7.39
D	22.0	3.21	47.3	6.66
E	23.5	3.16	44.7	6.84
F	8.7	3.21	48.2	6.49
G	4.2	3.00	41.5	6.88
H	8.6	3.41	47.5	6.98

The average calculated head assay was 7.0 g/t Au with a range from 6.5 g/t to 7.9 g/t Au.

A sample of the table tailing from cycle H was amalgamated to investigate the presence of free gold. Less than 1% of the gold in the tailing (0.4% overall) was recovered in the amalgam.

Similarly, the amalgamation tailing from cycle H was retreated under the same conditions as the first amalgamation. Because of the significant increase in the recovery of free gold, the tailing was amalgamated a third time. The following results were obtained:

Test 45-H

Product	% Dist. Au
Table Conc. Amalgam 1	8.6
Table Conc. Amalgam 2	32.2
Table Conc. Amalgam 3	5.7
Table Conc. Amal. 3 Tail.	6.0
Table Tail. Amalgam	0.4
Table Tail. Amal. Tail.	47.1
Feed	100.0

Summary - Continued

2. Zone F - Cont'd

In each amalgamation, the mercury easily coalesced and appeared clean. The overall amount of free gold in the ore at a grind of 87 % minus 200 mesh was 46.9 % with over 99 % of that gold present in the table concentrate.

By treating a larger sample, a more consistent head assay could be calculated. Amalgamation of the gravity concentrate was difficult due to the high content of arsenopyrite.

3. Zone C

A 10 kg Zone C sample was ground to 85 % minus 200 mesh. A gravity concentrate was recovered and amalgamated with mercury, sodium hydroxide and white lead. The results are given below.

Table No. 3 - Zone C

Test No.	Product	Weight %	Assay, g/t Au	% Distribution Au
44	Amalgam		50.44*	12.7
	Amal. Tail.	2.68	819.	55.1
	Table Tail.	97.32	13.2	32.2
	Head (calc.)	100.00	39.8	100.0

* mg

The calculated head is similar to the direct head assay. From the results of the testwork conducted on Zone F, it would seem quite probable that much of the free gold was not recovered during the amalgamation.

SAMPLE PREPARATION

The remaining ore from the Zone F samples was crushed to minus 10 mesh. Duplicate head samples were removed and the remaining ore riffled into test charges.

The Zone C sample was prepared in a similar manner.

DETAILS OF TESTS

Test No. 42

Purpose: To investigate tabling and amalgamation as a means of calculating a head assay for the ore.

Procedure: The sample was ground in a ball mill and transferred to a 20 liter conditioning tank. The pulp was fed over a Deister concentrating table and a gravity concentrate collected. The concentrate was transferred to a bottle for amalgamation. The pH was raised with NaOH to 11, and the sample was amalgamated with 15 g of mercury for 2 hours. The amalgam was recovered by elutriation.

Feed: 10 kg minus 10 mesh Zone F.

Grind: 20 minutes in a large ball mill at 65% solids.

Observations: A few small pieces of free gold were seen on the table. The recovered mercury was dirty and did not coalesce. 10 mL of xanthate (1% sol'n) were added. This aided in coalescing the amalgam to a limited extent.

Metallurgical Results

Product	Weight %	Assays,mg,g/t Au	% Distribution Au
Amalgam	-	6.46	8.6
Amal. Tailing	4.23	83.9	47.1
Table Tailing	95.77	3.48*	44.3
Head (calculated)	100.00	7.86	100.0

* average of four assays: 4.14 g/t Au
3.26
4.03
2.50

Test No. 42 - Continued

Screen Analysis - Table Tailing

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	10.4	10.4	89.6
100	13.1	23.5	76.5
150	16.5	40.0	60.0
200	14.5	54.5	45.5
270	11.3	65.8	34.2
400	8.6	74.4	25.6
- 400	25.6	100.0	-
Total	100.0	-	-

Test No. 43

Purpose: To repeat Test No. 42 with a finer grind.

Procedure: The ground sample was fed over a Deister concentrating table and a gravity concentrate was collected. The concentrate was amalgamated for 2 hours on rolls at pH 11 with NaOH. Xanthate was added for an additional 30 minutes. The amalgam was recovered by elutriation.

Feed: 10 kg minus 10 mesh Zone F.

Grind: 50 minutes in a large ball mill at 65% solids.

Observations: The amalgam was largely covered with a black coating. The amalgamation tailing was retreated with 1.5 g NaOH, 2 g white lead and 15 g Hg. The recovered amalgam was clean.

Metallurgical Results

Product	Weight %	Assays,mg,g/t Au	% Distribution Au
Amalgam	-	10.73	14.8
Amal. Tailing	2.11	139.	40.4
Table Tailing	97.89	3.32	44.8
Head (calculated)	100.00	7.41	100.0

Amalgam B	-	3.75	7.4
Amal. Tailing B	2.11	110.	33.0
Head (calculated)	-	13.5	40.4

Test No. 43 - Continued

Screen Analysis

Table Tailing

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 100	0.6	0.6	99.4
150	3.7	4.3	95.7
200	9.2	13.5	86.5
270	16.8	30.3	69.7
400	14.8	45.1	54.9
- 400	54.9	100.0	-
Total	100.0	-	-

Screen Analysis

Test No. 44

Purpose: To repeat Test No. 43 on Zone C.

Procedure: As for Test No. 43, except 2 g NaOH, 1 g white lead and 30 g Hg were added to the amalgamation.

Feed: 10 kg minus 10 mesh Zone C.

Grind: 50 minutes in large ball mill at 65% solids.

Observations: The recovered amalgam was clean with many small pieces of gold visible.

Metallurgical Results

Product	Weight %	Assays, mg, g/t Au	% Distribution Au
Amalgam	-	50.44	12.7
Amal. Tailing	2.68	819.	55.1
Table Tailing	97.32	13.2	32.2
Head (calculated)	100.00	39.8	100.0

Screen Analysis

Table Tailing

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	0.1	0.1	99.9
100	0.6	0.7	99.3
150	4.3	5.0	95.0
200	10.1	15.1	84.9
270	14.9	30.0	70.0
400	14.6	44.6	55.4
- 400	55.4	100.0	-
Total	100.0	-	-

Test No. 45

Purpose: To investigate tabling and amalgamation as a means of calculating a head grade for the ore.

Procedure: Each 10 kg charge was ground and fed over the Deister concentrating table. Two table tailing samples were taken directly from the table. The table concentrate was amalgamated with 1 g white lead, 2 g NaOH and 30 g Hg. The amalgam was recovered by elutriation.
Two of the amalgamation tailings and one table tailing were amalgamated under similar conditions to determine if any free gold was present in these products.

Feed: 8 x 10 kg minus 10 mesh Zone F.

Grind: 50 min/10 kg in large ball mill at 65% solids.

Note: A to E: Pb added after 2 hours
A and B - elutriated after 15 minutes with lead
C to E - elutriated after 30 minutes with lead
F to H - lead added initially

Metallurgical Results

Test No.	Product	Weight %	Assay, g/t Au	% Distribution Au
45-A	Amalgam	-	13.54*	17.1
	Amal. Tail.	1.53	177.	34.1
	Table Tail.	98.47	3.92	48.8
	Head (calc.)	100.00	7.92	100.0
45-B	Amalgam 1	-	4.27*	6.1
	Amal. 1 Tail.	2.56	135.	49.6
	Table Tail.	97.44	3.17	44.3
	Head (calc.)	100.00	6.98	100.0
	Amalgam 2	-	7.08*	11.5
	Amal. 2 Tail.	2.56	107.	38.1
	Head (calc.)	2.56	139.	49.6
45-C	Amalgam	-	10.68*	14.5
	Amal. Tail.	2.77	101.	37.9
	Table Tail.	97.23	3.62	47.6
	Head (calc.)	100.00	7.39	100.0

LAKEFIELD RESEARCH OF CANADA LIMITED
Lakefield, Ontario
April 12, 1983 / tmg

Test No. 45 - Continued

Metallurgical Results - Cont'd

Test No.	Product	Weight %	Assay, g/t Au	% Distribution Au
45-D	Amalgam Amal Tail. Table Tail.	- 1.90 98.10	14.63* 108. 3.21	22.0 30.7 47.3
	Head (calc.)	100.00	6.66	100.0
45-E	Amalgam Amal Tail. Table Tail.	- 3.34 96.66	16.11* 65.0 3.16	23.5 31.8 44.7
	Head (calc.)	100.00	6.84	100.0
45-F	Amalgam Amal Tail. Table Tail.	- 2.49 97.51	5.66* 112. 3.21	8.7 43.1 48.2
	Head (calc.)	100.00	6.49	100.0
45-G	Amalgam Amal. Tail. Table Tail.	- 4.64 95.36	2.87* 80.5 3.00	4.2 54.3 41.5
	Head (calc.)	100.00	6.88	100.0
45-H	Amalgam 1 Amal 1 Tail. Table Tail.	- 2.73 97.27	5.99* 112. 3.41	8.6 43.9 47.5
	Head (calc.)	100.00	6.98	100.0
	Amalgam 2 Amal 2 Tail.	- 2.73	20.94* 30.6	32.2 11.7
	Head (calc.)	2.73	115.	43.9
	Amalgam 3 Amal 3 Tail.	- 2.73	2.45* 12.0	5.7 6.0
	Head (calc.)	2.73	23.5	11.7
	Table Tail. Amal Table Tail. Amal Tail.	- 97.27	0.25* 3.08	0.4 47.1
	Head (calc.)	97.27	3.11	47.5

* mg

An Investigation of

THE RECOVERY OF GOLD

from samples

submitted by

A/S SULFIDMALM

Progress Report No. 2

Project No. L.R. 2570

NOTE:

This report refers to the samples as received.

LAKEFIELD RESEARCH OF CANADA LIMITED
Lakefield, Ontario
June 3, 1982

INDEX

Page No.

S U M M A R Y

1. Head Analysis

Representative samples were removed from C and F zone ore for analysis.

Element	C Zone	F Zone
Au (g/t)	39.1 (41.8)	7.77 (6.83)
Ag (g/t)	3.3	2.3
As (%)	7.71	10.9
Fe (%)	6.13	9.26
S (%)	3.39	5.05

(   ) average from testwork.

2. Cyanidation

Cyanidation tests were conducted on both samples to investigate various methods of reducing cyanide consumption as follows:

1. Preaerate at natural pH. Filter and discard solution. Cyanide under standard conditions (1 g/L NaCN, 33 % solids, pH 10.5-11.5, 2x24 h) in bottle test on rolls.
2. Preaerate with 0.5 g/L CaO. No intermediate filtration. Cyanide under standard conditions using preaeration solution.
3. Preaerate with 0.5 g/L CaO. No intermediate filtration. Cyanide under standard conditions using preaeration solution but increase pulp density to 50 % solids.
4. Cyanide under standard conditions but add 0.5 kg/t $Pb(NO_3)_2$ per stage.
5. Cyanide under standard conditions but maintain 0.5 g/L CaO.
6. Cyanide under standard conditions but reduce NaCN concentration to 0.25 g/L.

Preaeration significantly reduced the NaCN consumption. NaCN consumption by Sample C decreased from 2.9 kg/t under standard conditions to 1.3 kg/t with preaeration at natural pH to $\overline{0.9}$ kg/t with preaeration with 0.5 g/L CaO. NaCN consumption by Sample F decreased from 2.8 kg/t under standard conditions to 1.1 kg/t with preaeration at natural pH to $\overline{1.7}$ kg/t with preaeration with 0.38 g/L CaO.

I N T R O D U C T I O N

Additional cyanidation tests were conducted on two samples of gold-arsenopyrite ore from Bindal, Norway as a continuation of Progress Report No. 1. The purpose of the testwork was to reduce cyanide consumption.

In a telex dated May 26, 1982, Mr. Frank Nixon of A/S Sulfidmalm requested that no further testwork be conducted at the present time.

LAKEFIELD RESEARCH OF CANADA LIMITED

D.M. Wyslouzil

D.M. Wyslouzil, P. Eng.,

Manager

Rod Williamson

R.G. Williamson, P. Eng.,

Senior Project Engineer

Investigation by: B. Thomas
L. Paquette

Summary - Continued

2. Cyanidation - Cont'd

The preaeration procedures did not have any significant effect on gold extraction which ranged from 93 to 95 % from Sample C and 75-80 % from Sample F. The higher gold extraction figures reflected higher head assays: the cyanide residue assays were similar.

Cyanidation with high lime and the addition of lead nitrate reduced NaCN consumption by Sample C by 10 and 20 % respectively.

An NaCN concentration of 0.25 g/L NaCN reduced gold extraction from Sample C to 77 % after 48 hours.

The test conditions and results are contained in Table No. 1.

Table No. 1 - Cyanidation Test Conditions and Results

Test No.	Sample	Grind % -200 mesh	Treatment	R.P.*	Reagent Conc.		Reagent Cons.		Gold Ext'n %	Residue Assay Au, g/t	Head Assay Au, g/t
					NaCN g/L	CaO g/L	NaCN kg/t	CaO kg/t			
28	C	70	Standard test, P.R. No. 1	200	1.0	-	2.9	1.2	93	2.47	36.8
33	C	70	Aerate 1 h with no lime Filter and discard solution	129	1.0	-	1.3	1.6**	95	2.48	47.7
34	C	70	Aerate 1 h with 0.5 g/L CaO. No intermediate filtration	130	1.0	0.5	0.7	2.9	93	3.07	44.6
35	C	70	0.5 kg/t $Pb(NO_3)_2$ per stage	187	1.0	-	2.3	1.4	94	2.48	42.7
36	C	70	Repeat Test 28 with high lime	125	1.0	0.5	2.6	1.9	94	2.61	40.7
37	C	70	Repeat Test 34 with higher pulp density (50 % solids)	141	1.0	0.5	1.1	2.5	93	2.63	39.0
38	C	70	Repeat Test 37 but reduce NaCN to 0.25 g/L	149	0.25	0.5	0.3	2.9	77	9.60	41.0
31	F	76	Standard test, P.R. No. 1	218	1.0	-	2.8	2.0	80	1.57	7.76
39	F	76	Aerate 1 h with no lime Filter and discard solution	118	1.0	-	1.1	1.7**	76	1.58	6.61
40	F	76	Aerate 1 h with 0.38 g/L CaO. No intermediate filtration	155	1.0	0.38	1.8	2.9	75	1.58	6.39
41	F	76	Repeat Test 40 with higher pulp density (50 % solids)	176	1.0	0.38	1.6	2.2	76	1.58	6.57

* Reducing Power: mL of 0.1 N $KMnO_4$/L pregnant solution after 24 hours
**Does not include lime required to neutralize filtrate.

DISCUSSION

Gold extraction was essentially complete from both samples within 24 hours under the standard cyanidation conditions. Increasing the lime concentration to 0.38-0.5 g/L CaO slightly reduced the gold dissolution rate. Increasing the pulp density to 50 % solids also slightly reduced the gold dissolution rate such that an additional 2-3 % gold extraction was obtained in the second 24 hour stage.

Cyanide consumptions discussed in the report are NaCN consumption after 2 x 24 hours of leaching.

RECOMMENDATIONS

Determine the cause of the cyanide consumption by analysis of the heavy metal and cyanide complexes.

Examine the effect of alkali chlorination and SO_2/aeration in batch tests on the chemical composition of the barren solution after zinc dust precipitation.

SAMPLE PREPARATION

Described on page 13, Progress Report No. 1, April 1982.

INVENTORY

The following samples are on hand at Lakefield:

- 10 mm C Zone	8 kg
- 1.7 mm (10 mesh) C Zone	1 x 2 kg + 2 x 500g
- 10 mm F Zone	100 kg
- 1.7 mm (10 mesh) F Zone	2 x 2 kg + 1 x 500 g
Cleaner Concentrate Test 15 C Zone	1 x 250 g
Cleaner Concentrate Test 16 F Zone	4 x 250 g
Combined Cleaner Tailing Test 16 F Zone	2 x 250 g

DETAILS OF TESTS

Test No. 33

Purpose: To investigate the effect of preaeration on NaCN consumption.

Procedure: The ground sample was filtered and preaerated for one hour in a Denver flot cell with no lime at 40 % solids. The pulp was filtered and washed three times with water. The residue was then cyanided as for Test 27.

Feed: 500 g minus 10 mesh Sample C.

Solution Volume: 1000 mL   Pulp Density 33 % solids

Solution Composition: 1.0 g/L NaCN

pH Range: 10.5-11.5 with $Ca(OH)_2$

Grind: 10 minutes at 66 % solids in the lab rod mill.

Reagent Balance:

Time Hours	Added, grams Actual NaCN	Actual $Ca(OH)_2$	Equivalent NaCN	Equivalent CaO	Residual Grams NaCN	Residual Grams CaO	Consumed Grams NaCN	Consumed Grams CaO	pH	R.P.*
1 h preaeration										
	-	-	-	-	-	-	-	-	8.1- 7.8	-
Cyanidation - 1st Stage										
0-2	1.06	0.30	1.0	0.23	0.75	0.00	0.25	0.23	11.1-10.0	-
2-6	0.26	0.20	0.25	0.15	0.95	0.00	0.05	0.15	10.9-10.3	-
6-24	0.05	0.20	0.05	0.15	0.86	0.00	0.14	0.15	11.1-10.3	129
2nd Stage										
24-28	1.06	0.30	1.00	0.23	0.90	0.03	0.10	0.20	11.4-10.7	-
28-48	0.11	0.10	0.10	0.08	0.90	0.03	0.10	0.08	11.1-10.3	88
Total	2.54	1.10	2.40	0.84	1.76	0.03	0.64	0.81	-	-

Reagent Consumption (kg/t of cyanide feed)   NaCN: 1.28   CaO: 1.62

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Test No. 33 - Continued

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Preg. + Wash	2150 mL	10.37	93.5
48 h Preg. + Wash	2000 mL	0.15	1.3
48 h Residue	498.8 g	2.48	5.2
Head (Calculated)	500.0 g	47.68	100.0

Test No. 34

Purpose: Preaerate with 0.5 g/L CaO. Add cyanide and continue test, without intermediate filtration.

Procedure: Same as Test 33, but with 0.5 g/L CaO in the preaeration. The pulp was not filtered and washed before cyanidation. 0.5 g/L CaO was maintained throughout cyanidation.

Feed: 500 g minus 10 mesh Sample C.

Solution Volume: 1000 mL   Pulp Density 33 % solids

Solution Composition: 1.0 g/L NaCN
0.5 g/L CaO

Grind: 10 minutes at 66 % solids in the lab rod mill.

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
1 h Preaeration										
	-	0.50	-	0.38	-	0.01	-	0.37	11.2- 8.5	-
Cyanidation - 1st Stage										
0-2	1.06	0.66	1.00	0.50	0.80	0.11	0.20	0.40	11.7-11.2	-
2-6	0.21	0.51	0.20	0.39	0.93	0.30	0.07	0.20	11.7-11.7	-
6-24	0.07	0.26	0.07	0.20	1.00	0.39	0.00	0.11	11.7-11.7	130
2nd Stage										
24-29	1.06	0.66	1.00	0.50	0.95	0.25	0.05	0.25	11.7-11.5	-
29-48	0.05	0.33	0.05	0.25	0.96	0.38	0.04	0.12	11.7-11.6	43
Total	2.45	2.92	2.32	2.22	1.96	0.77	0.36	1.45	-	-

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Reagent Consumption (kg/tonne of cyanide feed)   NaCN: 0.72   CaO: Preaeration - 0.74
Cyanidation - 2.16

Test No. 34 - Continued

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Preg. + Wash	2160 mL	9.40	91.0
48 h Preg. + Wash	2130 mL	0.22	2.1
48 h Residue	499.5 g	3.07	6.9
Head (Calculated)	500.0 g	44.60	100.0

Test No. 35

Purpose:	To repeat Test 28, but with 0.5 kg/t $Pb(NO_3)_2$/stage.
Procedure:	Same as Test 27.
Feed:	500 g minus 10 mesh Sample C
Solution Volume:	1000 mL Pulp Density 33 % solids
Solution Composition:	1.0 g/L NaCN
pH Range:	10.5-11.5 with $Ca(OH)_2$
$Pb(NO_3)_2$:	0.5 kg/t/stage
Grind:	10 minutes at 66 % solids in the lab rod mill.

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
1st Stage										
0-2	1.06	0.30	1.00	0.23	0.20	0.03	0.80	0.20	11.3-10.7	-
2-4	0.84	0.00	0.80	0.00	0.90	0.03	0.10	0.00	10.7-10.6	-
4-7	0.11	0.20	0.10	0.15	1.00	0.07	0.00	0.11	11.3-10.9	-
7-24	0.00	0.00	0.00	0.00	0.85	0.01	0.15	0.06	10.9-10.5	187
2nd Stage										
24-28	1.06	0.30	1.00	0.23	0.90	0.02	0.10	0.21	11.4-10.6	-
28-48	0.11	0.20	0.10	0.15	1.00	0.06	0.00	0.11	11.4-10.6	64
Total	3.18	1.00	3.00	0.76	1.85	0.07	1.15	0.69	-	-

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Reagent Consumption (kg/tonne of cyanide feed) NaCN: 2.30 CaO: 1.38

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Preg. + Wash	2120 mL	9.48	94.0
48 h Preg. + Wash	2000 mL	0.018	0.2
48 h Residue	500.3 g	2.48	5.8
Head (Calc.)	500.3 g	42.73	100.0

Test No. 36

Purpose: To repeat Test 28, but with 0.5 g/L CaO.

Procedure: Same as Test 27.

Feed: 500 g minus 10 mesh Sample C.

Solution Volume: 1000 mL Pulp Density 33 % solids

Solution Composition: 1.0 g/L NaCN
0.5 g/L CaO

Grind: 10 minutes at 66 % solids in the lab rod mill.

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
1st Stage										
0-2	1.06	0.66	1.00	0.50	0.13	0.19	0.87	0.31	11.7-11.4	-
2-4	0.92	0.41	0.87	0.31	0.90	0.35	0.10	0.15	11.8-11.7	-
4-7	0.11	0.20	0.10	0.15	0.95	0.42	0.05	0.08	11.8-11.8	-
7-24	0.05	0.11	0.05	0.08	0.88	0.46	0.12	0.04	11.9-11.9	125
2nd Stage										
24-28	1.06	0.66	1.00	0.50	0.90	0.25	0.10	0.25	11.7-11.6	-
28-48	0.11	0.33	0.10	0.25	0.95	0.40	0.05	0.10	11.8-11.7	32
Total	3.31	2.37	3.12	1.79	1.83	0.86	1.29	0.93	-	-

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Reagent Consumption (kg/tonne of cyanide feed) NaCN: 2.58 CaO: 1.86

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Preg. + Wash	2075 mL	9.07	92.3
48 h Preg. + Wash	2000 mL	0.14	1.4
48 h Residue	501.8 g	2.61	6.3
Head (Calc.)	501.8 g	40.65	100.0

Test No. 37

Purpose: To repeat Test 34, but at higher pulp density.

Procedure: Same as Test 34, but the preaeration was performed at 60 % solids and the cyanidation at 50 % solids.

Feed: 500 g minus 10 mesh Sample C.

Solution Volume: 500 mL   Pulp Density 50 % solids

Solution Composition: 1.0 gpl NaCN
0.5 gpL CaO

Grind: 10 minutes at 66 % solids in the lab rod mill.

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
1 h Pre-aeration										
	-	0.22	-	0.17	-	0.00	-	0.17	9.6- 8.0	-
Cyanidation - 1st Stage										
0-1	0.53	0.33	0.50	0.25	0.25	0.01	0.25	0.24	11.5-10.7	-
1-5	0.26	0.32	0.25	0.24	0.45	0.04	0.05	0.21	11.7-11.2	-
5-21	0.05	0.28	0.05	0.21	0.45	-	0.05	-	11.7-11.0	-
21-24	0.05	0.00	0.05	0.00	0.44	0.05	0.06	0.20	11.0-11.0	141
2nd Stage										
24-28	0.53	0.33	0.50	0.25	0.48	0.05	0.02	0.20	11.8-11.3	-
28-43	0.02	0.26	0.02	0.20	0.45	0.06	0.05	0.19	11.8-11.4	-
43-48	0.05	0.25	0.05	0.19	0.45	0.22	0.05	0.03	11.7-11.6	58
Total	1.49	1.99	1.42	1.51	0.89	0.27	0.53	1.24	-	-

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Reagent Consumption (kg/tonne of cyanide feed) NaCN: 1.06 CaO: Preaeration - 0.34
: Cyanidation - 2.14

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Preg. + Wash	1760 mL	9.67	87.2
48 h Preg. + Wash	1800 mL	0.66	6.1
48 h Residue	499.9 g	2.63	6.7
Head (Calc.)	500.0 g	39.04	100.0

Test No. 38

Purpose: To repeat Test 37, but with 0.25 g/L NaCN.

Procedure: Same as Test 37, but with 0.25 g/L NaCN.

Feed: 500 g minus 10 mesh Sample C.

Solution Volume: 500 mL   Pulp Density 50 % solids

Solution Composition: 0.25 gpl NaCN
0.50 gpL $Ca(OH)_2$

Grind: 10 minutes at 66 % solids in the lab rod mill.

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
1 h - preaeration										
	-	0.22	-	0.17	-	0.00	-	0.17	9.8- 8.1	-
Cyanidation - 1st Stage										
0-1	0.14	0.33	0.13	0.25	0.08	0.00	0.05	0.25	11.3-10.3	-
1-3	0.05	0.33	0.05	0.25	0.13	0.03	0.00	0.22	11.5-11.0	-
3-5	0.00	0.29	0.00	0.22	0.10	0.08	0.03	0.17	11.7-11.4	-
5-21	0.03	0.22	0.03	0.17	0.09	-	0.04	-	11.7-10.9	-
21-24	0.04	0.00	0.04	0.00	0.13	0.06	0.00	0.19	10.9-10.9	149
2nd Stage										
24-28	0.14	0.33	0.13	0.25	0.11	0.05	0.02	0.20	11.5-11.1	-
28-43	0.02	0.26	0.02	0.20	0.13	0.06	0.00	0.19	11.7-11.2	-
43-48	0.00	0.25	0.00	0.19	0.13	0.19	0.00	0.06	11.6-11.5	49
Total	0.42	2.23	0.40	1.70	0.26	0.25	0.14	1.45	-	-

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Reagent Consumption (kg/tonne of cyanide feed) NaCN: 0.28 CaO: Preaeration = 0.34
: Cyanidation = 2.56

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Preg. + Wash	1860 mL	5.67	51.4
48 h Preg. + Wash	1740 mL	2.98	25.3
48 h Residue	498.2 g	9.60	23.3
Head (Calc.)	500.0 g	41.04	100.0

Test No. 39

Purpose: To repeat Test 33, but on Sample F.

Procedure: As for Test 33.

Feed: 500 g minus 10 mesh Sample F.

Solution Volume: 1000 mL Pulp Density 33 % solids

Solution Composition: 1.0 gpl NaCN

pH Range: 10.5-11.5 with $Ca(OH)_2$

Grind: 10 minutes/500 g at 66 % solids in a lab rod mill.

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
Pre-aeration										
0-1	-	-	-	-	-	-	-	-	8.3-7.9	-
1st Stage										
0-2	1.0	0.28	0.95	0.21	0.73	0.00	0.22	0.21	10.9-10.0	-
2-18	0.23	0.25	0.22	0.19	0.83	0.00	0.12	0.19	11.0-10.0	-
18-24	0.13	0.24	0.12	0.18	0.95	0.08	0.00	0.10	10.9-10.5	118
2nd Stage										
0-1½	1.0	0.25	0.95	0.19	0.88	0.06	0.07	0.13	10.7-10.5	-
1½-17½	0.07	0.15	0.07	0.11	0.84	0.02	0.11	0.15	10.9-10.4	-
17½-24	0.12	0.16	0.11	0.15	0.95	0.10	0.00	0.07	10.8-10.5	39
Total	2.55	1.33	2.42	1.03	1.90	0.18	0.52	0.85	-	-

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Reagent Consumption (kg/tonne of cyanide feed)

	24 hours	48 hours
NaCN	0.69	1.05
CaO	1.01	1.72

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Preg. Solution	950 mL	2.36	68.5
24 h Wash Solution	1150 mL	0.20	7.0
48 h Preg. + Wash	2030 mL	0.01	0.6
48 h Residue	494.9 g	1.58	23.9
Head (Calc.)	494.9 g	6.61	100.0

Test No. 40

Purpose: To repeat Test 34, but on Sample F.

Procedure: As for Test 34.

Feed: 500 g minus 10 mesh Sample F.

Solution Volume: 1000 mL Pulp Density 33 % solids

Solution Composition: 1.0 gpl NaCN
0.5 gpL $Ca(OH)_2$

Grind: 10 minutes/500 grams at 66 % solids in a lab rod mill.

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
Preaeration										
0-1	-	0.50	-	0.38	-	0.00	-	0.38	11.0-8.6	-
1st Stage										
0-2	1.0	0.50	0.95	0.38	0.28	0.14	0.67	0.24	11.3-11.1	-
2-18	0.71	0.32	0.67	0.24	0.85	0.09	0.10	0.29	11.5-10.7	-
18-24	0.11	0.38	0.10	0.29	0.93	0.28	0.02	0.10	11.2-11.2	155
2nd Stage										
0-1½	1.0	0.50	0.95	0.38	0.91	0.18	0.04	0.20	11.3-11.3	-
1½-17½	0.04	0.26	0.04	0.20	0.86	0.17	0.09	0.21	11.4-11.2	-
17½-24	0.09	0.28	0.09	0.21	0.95	0.35	0.00	0.03	11.2-11.3	21
Total	2.95	2.62	2.80	1.99	1.88	0.63	0.92	1.36	-	-

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Reagent Consumption (kg/t of cyanide feed)

	Preaeration	24 hours	48 hours
NaCN	-	1.58	1.84
CaO	0.76	2.02	2.90

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Preg. Solution	920 mL	2.28	65.8
24 h Wash Solution	2060 mL	0.12	7.8
48 h Preg. + Wash	2000 mL	0.024	1.6
48 h Residue	499.3 g	1.58	24.8
Head (Calc.)	499.3 g	6.39	100.0

Test No. 41

Purpose: To repeat Test 40, but at a higher pulp density.

Procedure: As for Test 37.

Feed: 500 g minus 10 mesh Sample F.

Solution Volume: 500 mL Pulp Density 50 % solids

Solution Composition: 1.0 gpL NaCN
0.50 gpL $Ca(OH)_2$

Grind: 10 minutes/500 g at 66 % solids in a lab rod mill.

Reagent Balance:

Time Hours	Added, grams				Residual		Consumed		pH	R.P.*
	Actual		Equivalent		Grams		Grams			
	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO		
Pre-aeration										
0-1	-	0.25	-	0.19	-	0.00	-	0.19	9.6-8.2	-
1st Stage										
0-2	0.50	0.25	0.48	0.19	0.09	0.02	0.39	0.17	11.0-10.0	-
2-18	0.41	0.22	0.39	0.17	0.22	0.00	0.16	0.19	11.1-9.9	-
18-24	0.17	0.25	0.16	0.19	0.42	0.04	0.06	0.15	11.1-10.5	176
2nd Stage										
0-1½	0.50	0.25	0.48	0.19	0.42	0.04	0.06	0.15	11.2-10.8	-
1½-17½	0.06	0.20	0.06	0.15	0.41	0.01	0.07	0.18	11.2-10.4	-
17½-24	0.07	0.24	0.07	0.18	0.43	0.10	0.05	0.09	11.2-10.9	58
Total	1.71	1.58	1.64	1.20	0.85	0.14	0.79	1.06	-	-

*Reducing Power: mL 0.1 N $KMnO_4$/L pregnant solution

Reagent Consumption (kg/t of cyanide feed)

	Preaeration	24 hours	48 hours
	NaCN : -	NaCN : 1.24	NaCN : 1.61
	CaO : 0.38	CaO : 1.40	CaO : 2.24

Metallurgical Results

Product	Amount	Assays, mg/L,g/t Au	% Distribution Au
24 h Preg. Solution	440 mL	4.48	61.0
24 h Wash Solution	780 mL	0.49	11.8
48 h Preg. + Wash	1380 mL	0.074	3.1
48 h Residue	491.5 g	1.58	24.1
Head (Calc.)	491.5 g	6.57	100.0