THE SOCIALIST REPUBLIC OF THE UNION OF BURMA

REPORT ON GEOLOGICAL SURVEY OF THE MONYWA AREA

(STRIPPING AND UNDERGROUND MINING
CONSTRUCTION OF PILOT PLANT
CONCENTRATING TEST)

PHASE III & IV
(VOL. IV)

July 1976

METAL MINING AGENCY
JAPAN INTERNATIONAL COOPERATION AGENCY
GOVERNMENT OF JAPAN

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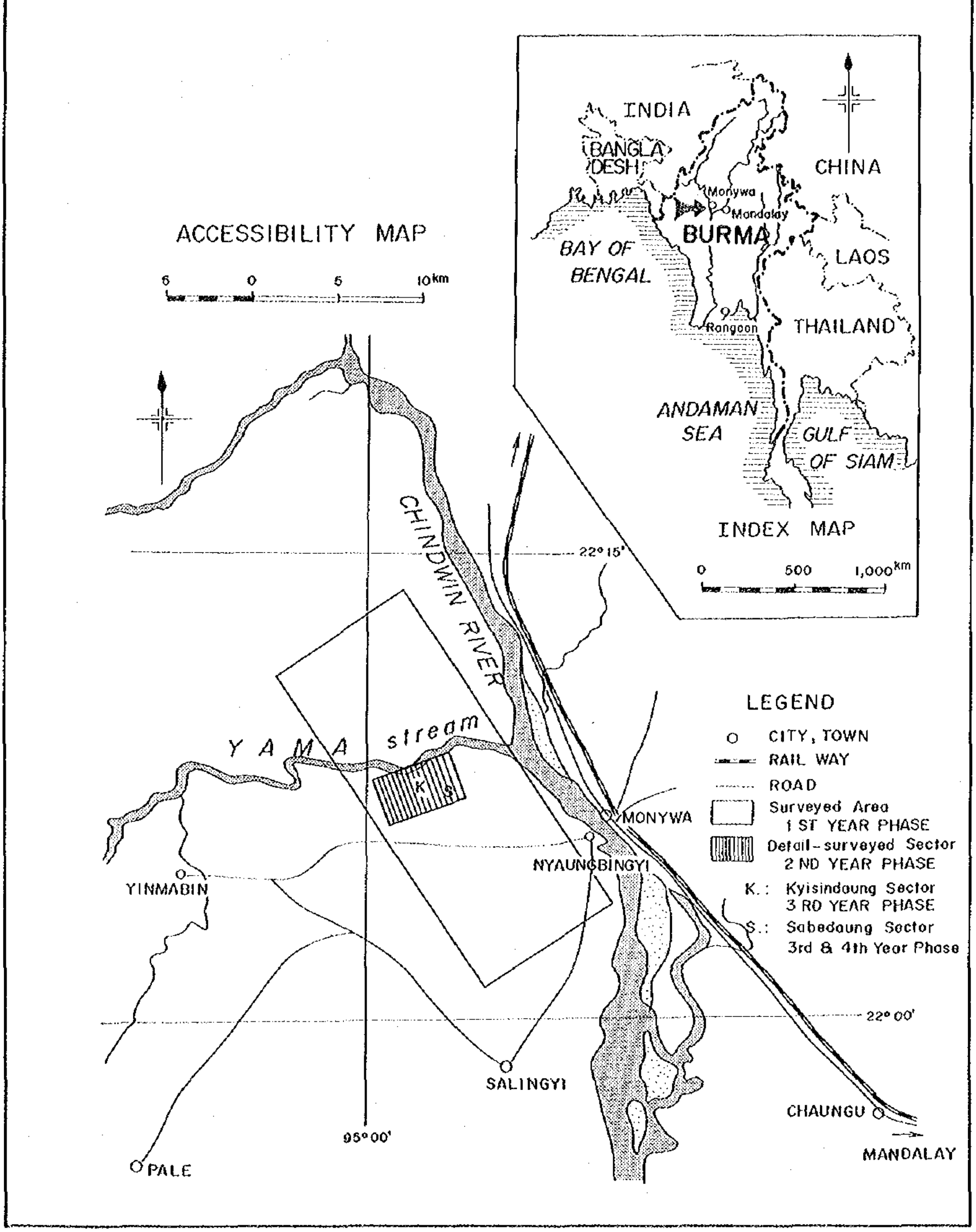
Fig. 1

LOCATION MAP OF THE SURVEYED AREA



Summary

In Phase I and II of the mineral survey project, under the Bi-govermental agreement between Burma and Japan, in the Monywa area of the Socialist Republic of the Union of Burma, geological reconnaissance, geophysical survey (IP method), and selective core drilling were carried out in those districts which contain Sabedaung, Kyisindaung and Letpadaung.

By means of these survey activities, the geological structure and the occurrence of copper minerals in the Monywa area have been established.

Furthermore, by the metallurgical testing of the core samples collected from Sabedaung, the concentrating method of the copper mineral can be regarded with some degree of confidence.

Based on all of these survey results, in Phase III, a pilot plant was constructed at the north eastern foot of the Kyisindaung hill. At the same time a partial stripping of the overburden on Sabedaung deposit was also performed in order to obtain the necessary technical information on open pit mining.

The construction of the pilot plant started in April 1975. First, the ground arrangement was carried out.

Then, ferro-concrete foundations, setting up of Mill machinery and a 250 KVA diesel generator, miscellaneous pipe and wire setting, housing construction, water and fuel supply installations, banking for the waste tailing pond, and surrounding road construction, etc. were successively materialized to complete the 50 t/d pilot plant.

After the completion of the plant on October 28, a metallurgical test was undertaken between November 11, 1975 and February 14, 1976, to collect data

necessary for deciding the operational system of the concentraing mill, after grasping the ore characteristics for the mineral treatment by flotation.

On the other hand, on the mining side, the testing location was selected at around JS-9 drill-hole in the southwest of the Sabedaung deposit with a bottom level of 108m above sea level.

Then, on May 13, 1975, a bulldozer started the stripping of the mineralized zone.

Afterwards, an open channel was cut by means of drilling and blasting to make a shorter access to the ore deposit.

In Phase IV, a tunneling method was adopted, in replacing the initially designed open pit because of considerations both of the limited time for supplying crude ore for the mill test, and of the amount of explosives available at the time. The mining activity by the tunneling method continued until Feb., 10, 1976.

During these periods, Phase III and IV, the following main performances and results were obtained:

Removed overburden	$9,100m^3$	
Rock cut at Channel	$1,138m^3$	
Total of Stripping	$10,238m^3$	
Length of Tunnel driven	91.7m	
Obtained ore from Tunneling	1,758t	*
Obtained ore from Wall & Roof	2,085t	
Cutting of the tunnel		
Total of mined ore	3,843t	

* 232 ton of lowest grade ore is included.

Ore hauled to Surface	3,788t *
Ore transported to Mill	2,738t
Ore treated at Mill	2,138t
Days of Mill operation	68
Avarage ore treated at Mill	31.4t/d

Recovery in Flotation (Inferred for ore of 0.9% Cu)

Method used	Recovery	Conc. grade	Tail grade
Conventional method	70.7%	30%	0,27%
Scavenger-Cleaner method	75.0%	30%	0.23%

Chief machiney used was as follows.

Bulldozer	2	Jeep	1
Shovel Loader	2	Air Compressor (17m^3/min)	1
Dump truck	2	Leg drill	4
Single toggle crusher	2	Conditioner	2
Tube mill	2	Negaclon	4
Flotator	22 cells	Diesel generator 250 KVA	1

Through out all the processes and tests mentioned above, the following major items were understood:

1 Overburden stripping, for soil and soft rock, can be efficiently done to the depth of 4 to 5 meter from the surface by means of heavier machinery.

2 Easy drilling and effective blasting were assured for the ore body, as well as the rocks of the surrounding area. And yet almost no natural scaling on the roof and wall has occured underground even in enlarging the tunnel to a width of 7 meters allowing non-timber tunneling.

3 It was positively inferred from the mill test that the copper minerals can be economically recovered by flotation from the Sabedaung ore. The flotation test gave a concentrate grade of 30%, in copper content, higher than the initially expected value of 20%. Recovery was proved to be 75% against an initial estimate of 80%.

4 The prevention of oxidation of the ore, throughout the entire period of mining, concentrating, storage, and transportation, will have to be studied, because of the remarkable speed of oxidation of the copper ore which was observed during the processes of mining and milling

Furthermore, the in-place leaching method will be of importance for the recovery of the copper metal from the lower grade ore, in addition to the flotation tailings.

Part I

Stripping and Underground Mining

Part I Stripping and Underground Mining

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Chapter 1 Introduction

1-1 Purpose of Survey

On the basis of the survey results of the geological reconnaissance, geophysical survey, and core drilling, all of which had been undertaken in Monywa area since Phase I, 1972 fiscal year, a pilot plant with 50t/d capacity was constructed at Kyisindaung in Phase III.

To keep pace with this construction, stripping work started at Sabedaung to excavate the ore for the pilot plant.

Here in this volume, the Stripping work mentioned above, together with the relevant channel cut, and the subsequent Underground Mining by an exprolation tunnel, as of Phase IV, are introduced.

This work was carried out to clarify both the geological structure and the mineralized zone of the Sabedaung deposit, as well as to acquire the technical imformation necessary for the future mining of the deposit, and also to supply the amount of ore necessary for concentrating tests at the pilot plant.

1-2 Outline

1-2-1 Location (Fig. I-1-1)

The southwestern part of Sabedaung deposit was selected for the Survey location.

In particular the JS-9 hole, drilled by the Japanese survey team in Phase II, was marked as the center of the mine site.

The bottom level of the site was fixed at 108m above sea level (hereafter, ASL), which is 5 meter higher than the nearest mine road.

1-2-2 Sequence of Operations (Fig. I-1-2)

Bulldozers removed surface soil, and scratched the soft rock away from the site. Then, an open channel was cut towards the JS-9 hole by means of drilling and blasting to enable an earlier start of the mining to get ore for the mill.

With the trust and understanding of the concerned personnel of both Burma and Japan, a tunnel was driven from the front end of the channel for the JS-9 hole.

The tunnel was first extended straight after hitting the JS-9 hole at 27.4m from the portal, then a branch was cut from the above intersection at JS-9 hole toward DH-30 hole. Later the ends of these tunnels were connected by another short tunnel.

Then, the ore portions of these tunnels were enlarged by wall and roof cutting to obtain the crude ore for Mill Tests.

During these processes, underground geological surveys for the mineralized zone and the collection of technical data for future mining were also carried out.

1-2-3 Operations Carried Out (Fig. I-1-2, I-1-3)

	Earth Removal by Bulldozer	3,900m^3	
	Soft Rock Cutting by Bulldozer	5,200m^3	
	Open Channel Cut	1,138m^3	
A)	Total volume of Stripping	10,238m^3	
	Straight tunnel for JS-9 hole	48.3m	(section 3m X 3m)
	Branch tunnel for DH-30 hole	19.0m	(ditto)
	Connecting tunnel	8.6m	(section 2m X 2m)
	Sub-tunnel for wall cutting	15.8m	(ditto)
B)	Total lenght of tunneling	91.7m	
	Ore from Tunneling	1,758t	
	Ore from Wall and Roof Cutting	2,085t	

C)	Total of Mined Ore	3,843 t
	Ore hauled to Surface	3,556 t
	Broken ore left in tunnel	55 t
	Lowest grade ore for Waste	232 t
	(also hauled to Surface)	
D)	Total of Handled Ore	3,843 t
E)	Ore Transported for Mill	2,738 t

1-2-4 Period of Survey

From March 31, 1975 until Feb., 15, 1976, a period of 10.5 months was spent for the survey activities at the mine site.

However, actual stripping started only on May 13, 1975, because of an adjustment of the survey program which will be mentioned in detail later in Chapter 3.

So the period could be practically considered as being 9 months in total.

1-3 Members

Field work and the necessary compilation of the data concerned were undertaken by Mitsui Kinzoku Engineering Service Co., Ltd. (hereafter, MESCO) with the cooperation of Myanma Mineral Development Corporation (hereafter, MMDC). The menbers, concerned with the field work, are listed below:

1) Liasion and Coordination

Choki Okura	MESCO
Hidenori Sasaki	"
Nobuo Nagata	"
U Kyaw Aung	MMDC
U Ko Ko	"

U Thein Aung	MMDC
U Kyi	"
U Ye Win	"
U Zaw Lwin	"

2) Stripping and Mining

Michio Tanaka	MESCO
U Tin Maung	MMDC
U Saw Htu Tha	"
U Aye	"
U Win Maung	"

3) Geological Survey and Assay

Tsutomu Otsubo	MESCO
U Myo Myint	MMDC
U Toe Maung	"
U Aunt Kyaw	"
U Maung Maung Latt	"
U Tin Oo	"
U Sein Taik	"
U Myint Thein	"
U Than Maung	"

In addition to the menbers listed above, Japan International Cooperation Agency (hereafter, JICA) sent N. Nagata and T. Otsubo to Burma to cooperate in the Monywa project, as a Mining expert for 6 months and as a Survey expert for 4 months, respectively, under the Colombo Plan after March, 1975.

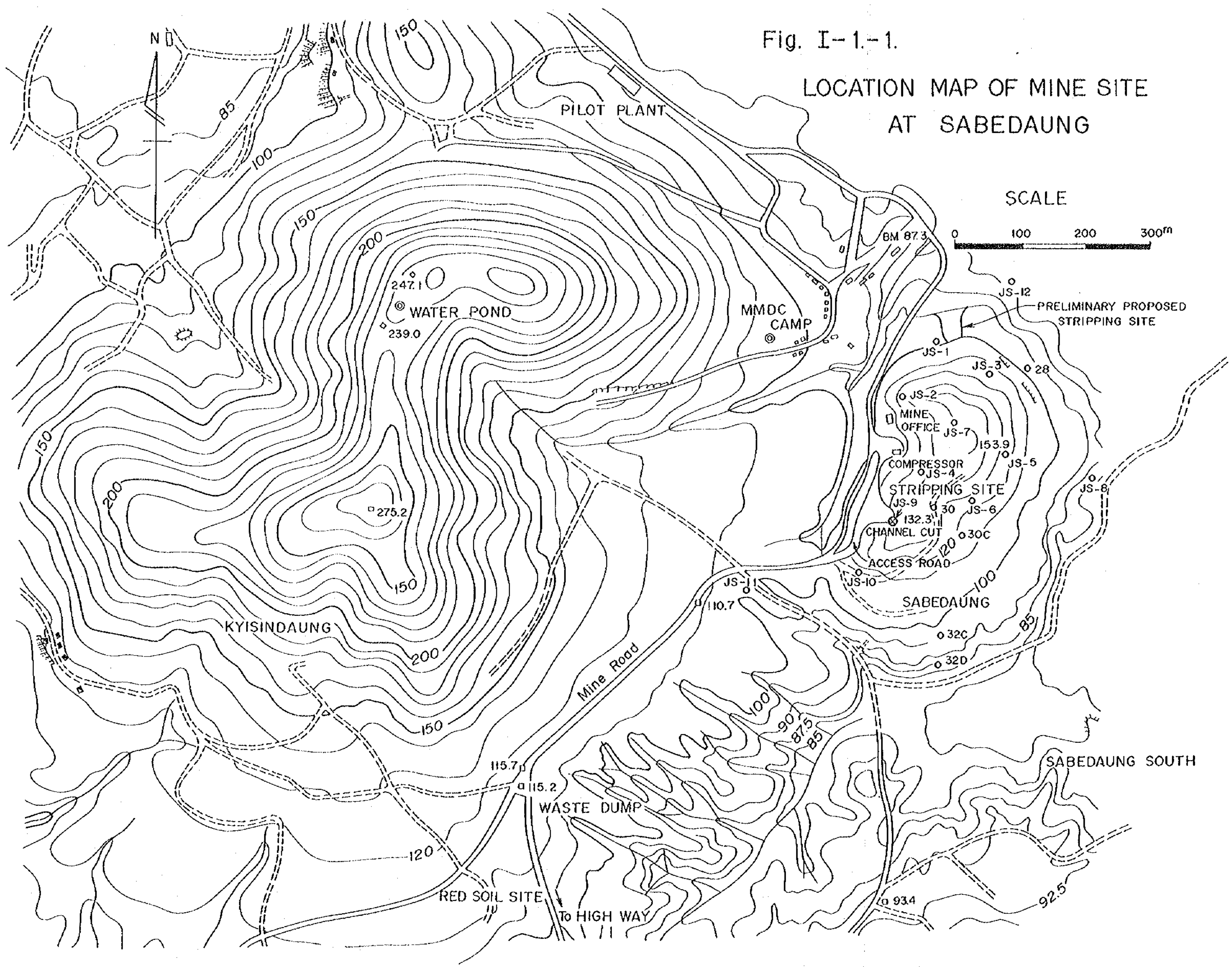


Fig. I-1.-1.
LOCATION MAP OF MINE SITE
AT SABEDAUNG
SCALE
0 100 200 300m
N
PILOT PLANT
WATER POND
247.1
239.0
275.2
MMDC CAMP
BM 87.3
JS-12
PRELIMINARY PROPOSED STRIPPING SITE
JS-1
JS-3
28
JS-2
MINE OFFICE
JS-7
153.9
COMPRESSOR
JS-4
JS-5
STRIPPING SITE
JS-9
30
JS-6
132.3
CHANNEL CUT
30C
ACCESS ROAD
JS-10
JS-11
JS-8
110.7
SABEDAUNG
32C
32D
KYISINDAUNG
Mine Road
115.7
115.2
WASTE DUMP
RED SOIL SITE
To HIGH WAY
SABEDAUNG SOUTH
93.4
92.5
85
90
87.5
100
120
150
200

FIG. I-1-2

LAYOUT OF SABEDAUNG MINE SITE

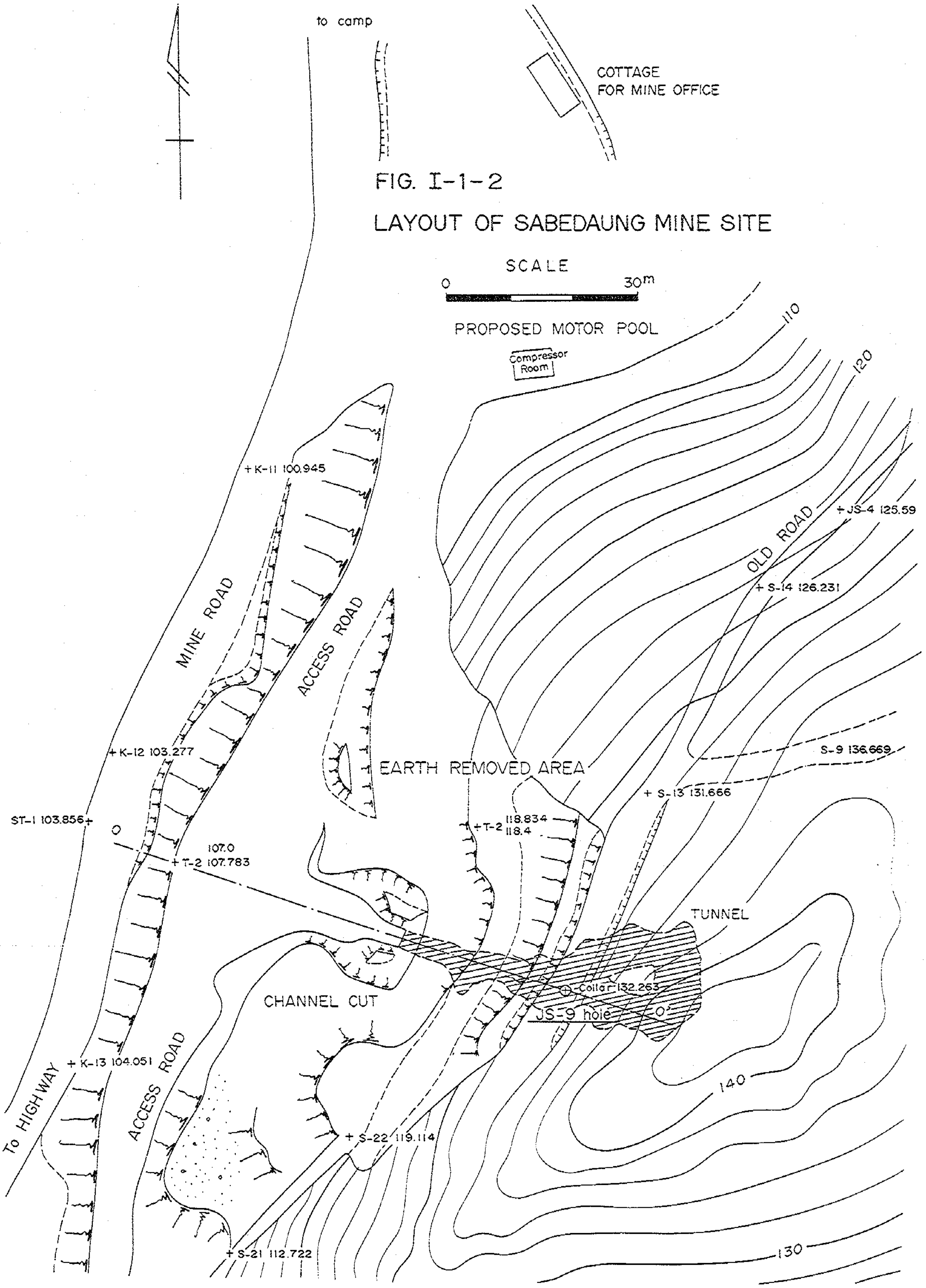
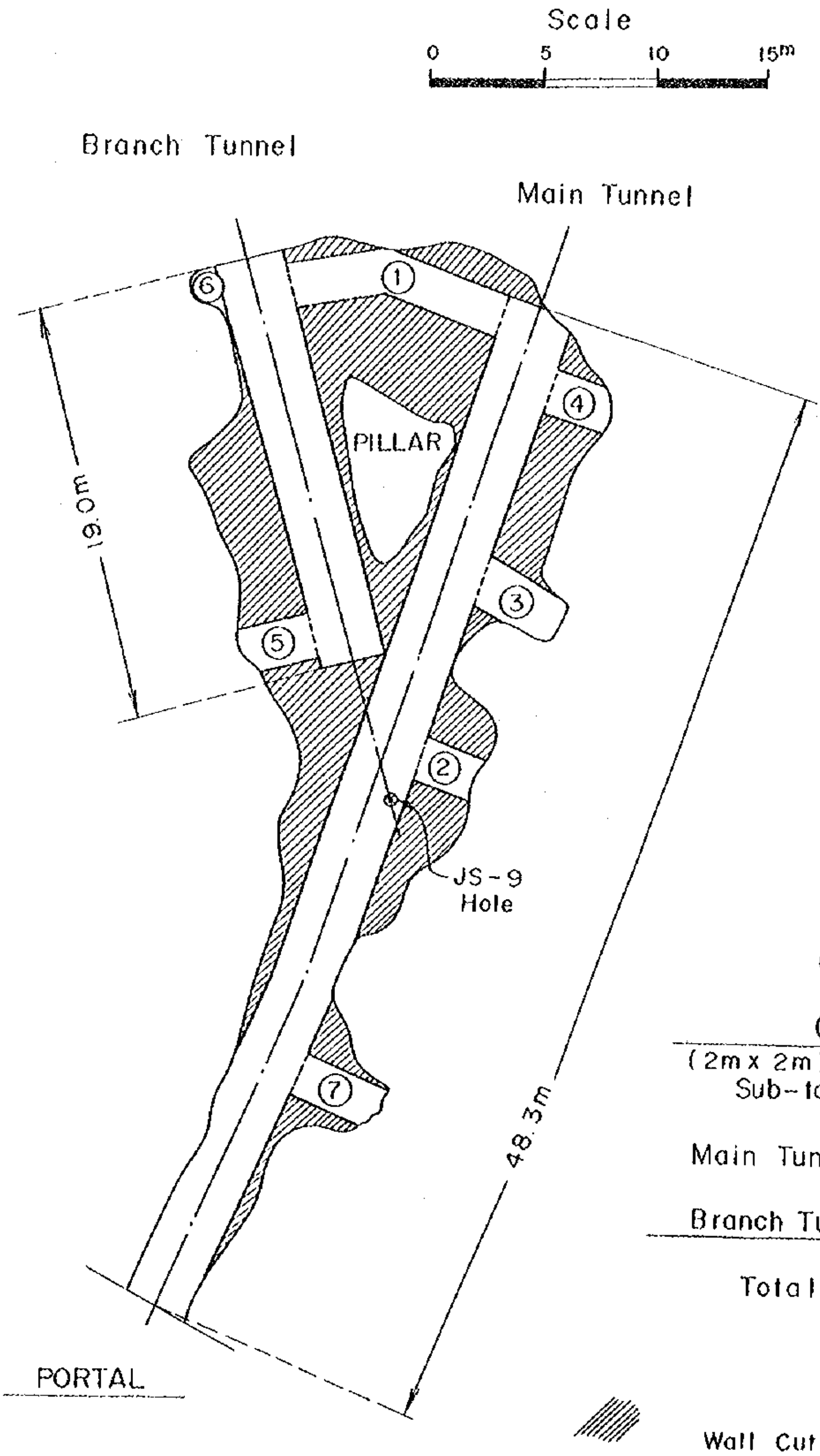


Fig. I-1-3

Detailed Map of Sabedaung Tunnel



①	8.6m
②	2.6m
③	4.1m
④	2.6m
⑤	2.6m
⑥	1.3m
⑦	2.6m
(2m x 2m) Sub-total	24.4m
Main Tunnel	48.3m
Branch Tunnel	19.0m
Total	91.7m

Chapter 2 Selection of Mine Site

2-1 Choice of Deposit

In reviewing all the results of the core drilling and the geological survey, the individual deopsits were ranked as follows,

A. Sabedaung deposit········Shallow overburden, high grade ore

B. Kyisindaung deposit·······Thick overburden, comparatively low ore grade

C. Sabedaung South deposit···(Still under confirming prospecting)

As a result the Sabedaung deposit was chosen to be the present Mine Site, because it required the shortest survey period and the least expense.

Futhermore there was the strong possibility that it might be selected in future as the initial object of the development of the mine.

2-2 Choice of Location (Fig. I-1-1)

At first, a northern part of Sabedaung was selected for the mine site in consideration not only of the fact that it was expected to have the shallowest overburden, but also because it was located at the shortest distance from the pilot plant. Unfortunately, after the preliminary stripping had been done by MMDC,the rock of the aforementioned site was found to be a very low grade oxidized zone with a large amount of white clay.

It was considered desirable that the feed ore, which was to be tested at the Mill, should have the average ore grade and quality of the deposit.

Furthermore even the ore, expected to be recoverable from deep within the site, came to be judged unfit for the planned mill test.

In addition, it was estimated that the volume of stripping would have to be increaced considerably in order to get ore of the proper quality from this location.

This raised difficulties in connection with the permissible time limit, as well as in the procurement of the necessary equipment and materials.

Accordingly it was requested that the stripping at northern Sabedaung be stopped as it was.

Then, the following 3 sectors were considered as possible sites for the test mining with stripping.

Eastern Sabedaung·················Vicinity of DH-28, JS-8 holes

Southern Sabedaung················ Vicinity of DH-39C, 32D holes

Center to Southwestern Sabedaung····Vicinity of DH-30, 30C, JS-9 holes

Of these sectors, the one which could fulfill the following criteria was to be chosen as the new location:

1) Shallowest overburden, with reliable information about the lateral extension of the ore deposit, preferably confirmed by core drilling.

2) Closest distance from the existing mine road for the convenience of the transportation of ore, debris, materials and the personnel concerned.

3) With efficiently higher core recovery, the expected ore should be fresh and should fullfill the average grade of Sabedaung deposit. It should also have an average ratio of copper to its accompanying sulphur, if possible similar to the relevant ratio of the Sabedaung deposit itself.

Based on these premises, careful and practical selection was made, and the new and final location was decided to be the Southwestern part of Sabedaung near JS-9 hole.

The bottom level of the mine site was set at 108m ASL in consideration of the collar elevation of JS-9 hole, 132.3m, and the depth of the ore body hit by the hole, 19.1m from the collar.

Chapter 3 Mining Method

3-1 Initial Understanding (Fig. 1-3-1)

At first, typical open pit mining was planned with full stripping of the surface soil and its underlying soft rock.

In this plan, a 51m long channel was designed, which was to be cut with a bottom width of 5m at 110m ASL, at the south western foot of Sabedaung, from the mine road straight to the JS-9 hole. Then a point was to be chosen as the center of a circular pit bottom with a radius of 10m at 12 meter on this side of the JS-9 hole.

The pit was designed to be completed with a final pit slope of 45 degree to form a semi-circular open pit.

In this case the total volume, partly including the ore zone, would amount to 22,000 cubic meter, and the requirement of explosives and detonators was estimated at as much as about 12 tons and over 20,000 pieces, respectively.

Futhermore, the working period necessary to prepare for the ore production was estimated to be at least four months, even if everything went according to schedule.

3-2 Checking of Method

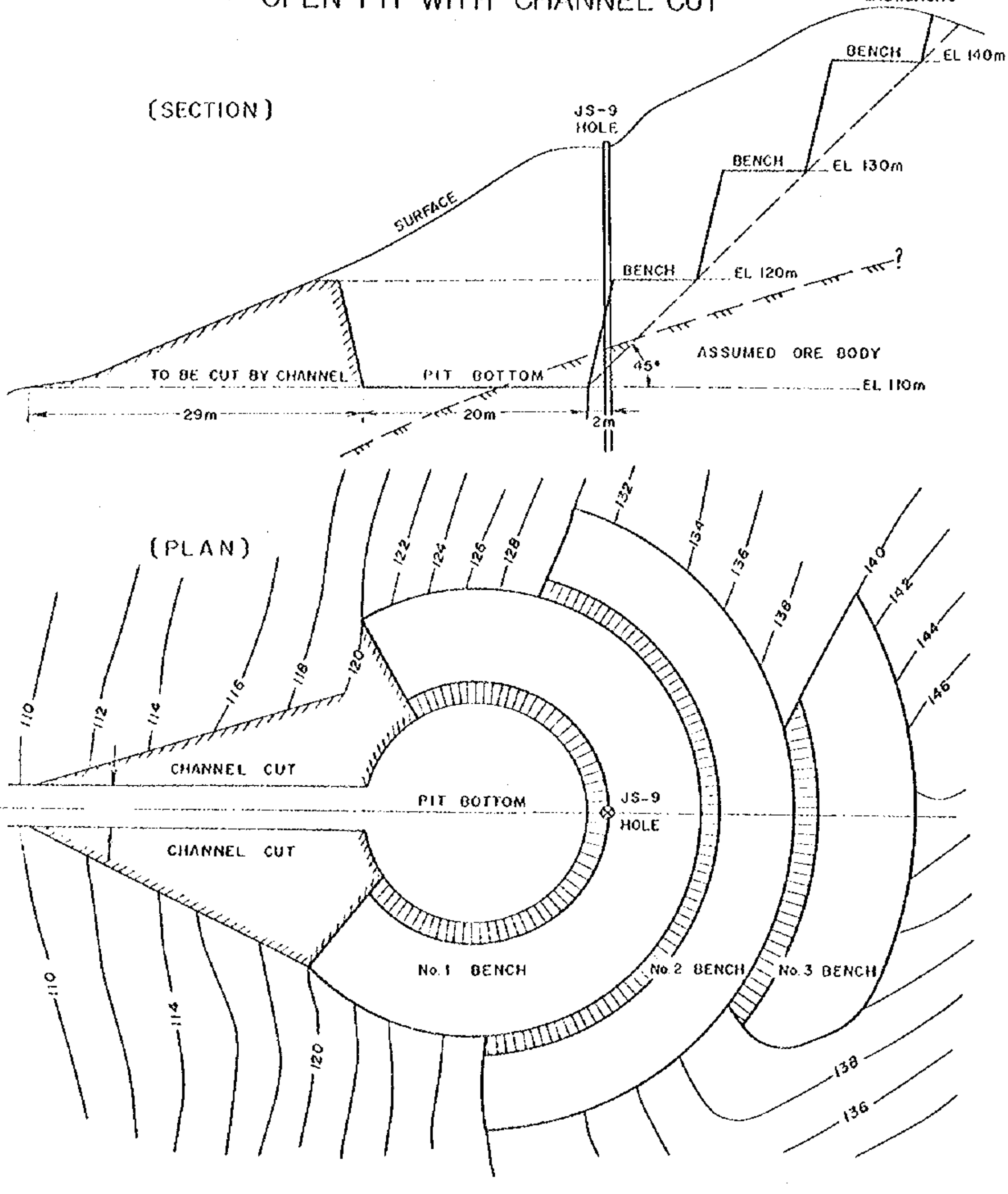
In considering the ideas mentioned above, the following problems were seriously discussed,

1) The amount of work would become too much in comparison with the original plan prepared for Phase III.

2) At that time the available explosive was only 1.72 ton from MMDC. Of this, 1.07 ton was a hygroscopic Ammonium Nitrate Explosive (Polar Ajax, made in 1968) and was already damp and unfit for use.

FIG. I-3-1

CONCEPTUAL DESIGN OF OPEN PIT WITH CHANNEL CUT



There were only 2,000 detonators in all, insufficient to meet the requirements.

Additional supply of explosives and detonators was thought to be very difficult.

3) Actual stripping started only in the middle of May, though the mill test had been expected to start in early September, and a minimun of another 4 months was required to prepare for the start of producing ore for the mill.

This might have created yet another delay in the schedule due to other unforeseenable factors.

4) Shortage in the capability of machinery, including drills, might also invite delay in the schedule, because even if the required machinery was ordered immediately, as an urgent requirement, delivery was expected to take a long time.

In addition, there were still several unknown factors related to the following items,

a) Thickness of the removable overburden, which could be bulldozed, without rock blasting.

b) Drillability of the hard rock

c) Consumption rate of Explosives

d) Skill of the workers

On the other hand, it was thought most important to give the first priority to ensuring the most reliable supply of test ore for the pilot plant immediately after its completion.

Then, the following plan of action came under practical consideration:

1) Remove as much soil and soft rock as possible away from the planned site

by bulldozer.

2) On encountering the hard rock zone, start open-channel cutting by drilling and blasting to approach as close as possible to JS-9 hole.

The floor level of the channel should be set at 108m ASL with an allowance for safety in getting ore against the possibly uneven boundary plane at the top of the ore body.

3) By following the plan of action mentioned above, it might be possible to clarify the unknown factors. Then, after paying due consideration to both the permissible drive period remaining and the available machinery, materials, etc. either of the following ways could be selected for further operations:

a) Continue the open pit mining

b) Change to underground mining which is mainly based on tunneling.

The decision on the selection of the method was postponed also due to the necessity of getting the clear understanding and consent of the personnel concerned.

3-3 Adopted Method (Fig. I-6-1)

The actual mining method can be described as follows:

First of all, a piece of red cloth was fixed near the JS-9 hole to show the center line of the mine site. Then the bulldozer started the earth removal, horizontally toward north and south, selectively on levels of 110m, 115m, 120m ASL in the early stage.

The width of the bulldozed area was approximately 100m on the lower level, and about 50m on the higher level.

It made a shape of depressed topography along the center line towards JS-9 hole, exposing the underlying soft rock after the earth removal.

After changing the surface of the mine site into an artificial open valley, the bulldozer began to cut the exposed soft rock, along the aforementiond center line, within 20 to 30 meters in width by edge scratching with its heavy blade. Thus, the central depressed portion became deeper and narrower.

Then, open channel cutting was introduced for the depression, elongating toward JS-9 hole.

In channel cutting, drilling and blasting were repeated against the narrow hard rock exposure of 15 to 20m in width, until the floor of the channel reached the planned bottom level of 108m ASL with a floor width of about 6 meters.

At this stage, some portion of the channel became as steep as a cliff, and suggested the need of preventing roll-stone accidents. Then unstable stones on the exposed rock surface were swept away. Also, a wood-plank fence, supported by broken rods which were fixed in short downward holes, was installed along the top end of the open channel, for holding any fallen stones.

At that time, there appeared a large scale vertical fissure with a Slickenside at the end of the channel at rectangles to the direction of the channel to the aforestated center line toward JS-9 hole. The fissure provided a chance to change the method to underground mining, because it was large enough to allow the driving of a tunnel conveniently and also the surrounding earth was solid enough to drive the tunnel.

The tunnel was given a standard section of 3m x 3m after considering the dimensions of the shovel loader HL-8, of which the maximum width is 2.2m, and effective height 2.5m in operation.

The tunnel was driven, together with supplementary wall and roof cutting, to confirm the underground geology and the ore body itself, as well as to produce the necessary amount of feed ore for the mill test.

Chapter 4 Machinery and Installations

4-1 Machinery and Equipment

The following list shows the main vehicles, construction and mining machines used either in stripping or in mining;

Name of Machine	Type/Specification	Q'ty	Remarks
Bulldozer	Komatsu D-80-12	1	
"	" D-85AH-12	1	
Shovel loader	Mitsui HL-8, capacity 0.8m^3	1	
"	Czechoslovakian	1	only for auxiliary use
Dump truck	Isuzu TSD-43, 6.5t	1	
"	Hino 6.5t	1	
Jeep	Toyota Land Cruiser	1	
Motor grader	5t class	1	
Land roller	Macadam 3t class	1	
Portable Compressor	Hokuetsu PDR-600, 17m^3/min	1	
Leg-drill	Furukawa 317-D	4	
Sinker	British	1	rarely used
Bit sharpener	Atlas Copco, beaver LSD-61	1	
Local fan	3.5 HP.		for underground ventilation
Gasoline Generator	Gasoline generator 100V, 1kW	1	

4-2 Materials

Materials and tools used are shown as follows,

Item	Type/Specification	Q'ty	Remarks
Rock bit	32m/m, 34m/m taper Carr bit	31	
Drill rod	22m/m, Hexagonal, 1.8m	25	
Insert bit	32m/m Gage, 22m/m Hex. 2.4m	3	
Grindstone	Bowl type, Carbolex	7	

Items	Type/Specification	Q'ty	Remarks
Hose	19m/m internal diameter	160m	for air
Hose	12m/m -ditto-		for water
Explosive	Gelignite NG 42%, dia. 22m/m		Germany Wasag Chemie
Electric detonator	Millisecond, lead wire 1.8m		I.C.I. Britain
Glaster	Nihon Kayaku, Capacity 50 pcs	2	
Circuit Tester	100 ohm type	2	
" "	Photo cell type	1	
Electric Wire	0.75 sqm/m 2 codes vinyl coating	640m	
Electric lead wire	0.45m/m dia. 2 codes vinyl coating		
Portable siren	6B type, hand rotation	2	
Fuel oil	Diesel oil and Gasoline		
Oil and Lubricant	Lubricating oil, Grease etc.		
Woods	Posts, Plank, etc.		
Roof material			for hut/cottage
Gas pipe/coupling	1"1/2	1,200	
Polyethylene pipe/ coupling	1"1/2	500	
Hand tools	Miscellaneous uses		
Survey tools	Transit, Level, Compass etc.		
Illumination	Honda generator, Kerosene lamp,etc..		
Assay tools	for Wet analysis, sand bath, etc.		
Others	Stemming, spacer, etc.		

4-3 Setting of Pipes

At first,compressed air was supplied from time to time, by the portable compressor, which was temporarily set close to the drilling site using a short connection of pipes.

Later on, the compressor was fixed at the compressor station, about 90m north of the portal, because the blasting location was brought underground. Underground blasting restricted much of the flying stones. At that time, the air was delivered into the tunnel by a pipe line of 1" 1/2 diameter.

On the other hand, water was supplied for the drills at the mine site directly from the water pond on the top of Kyisindaung, to maintain the constant water pressure which was necessary for drilling.

A pipe line of 1" 1/2 internal diameter, was extended for about 600m from the existing line at the southeastern foot of Kyisindaung.

Whenever it was necessary, the water was fed underground after reducing the water pressure to approximately 3kg/cm^2 at the portal of the tunnel.

In channel cutting, drilling was done without water supply, because it was performed on the surface.

Chapter 5 Review of Survey

5-1 Trend of Schedule

Bulldozer stripping started on May, 13, 1975, and after completing the earth and soft rock removal on June, 18, open channel cutting by means of drilling and blasting started on the next day.

On July 18th, only one week after changing over to tunneling on July, 11, the tunnel hit a part of the ore body at 6.6m from the portal. It encountered the JS-9 hole about one month later at 27.4m from the portal.

Until October, 1975, a total of 67.3m of large sectional tunnel had been driven, then 24.4m of small sectional tunnel was cut supplementarily.

In parallel to the latter tunneling, wall and roof cutting was carried out to produce the necessary amount of ore.

Simultaneously, the mined ore was gradually hauled out to the surface until early February, 1976.

The ore was then transported from near the portal to the pilot plant and was stored up there. To meet the completion of the plant, the ore transportation began on October 20, and continued until the end of January, 1976.

The trend of the major field work during Phase III and IV can be illustrated as shown in Table I-5-1.

5-2 Team arrangement and Working time

At the stage of bulldozer stripping and soft rock removal, 5 to 6 persons made a team which was composed of the bulldozer operater with his assistant, one counterpart, and 2 ~ 3 helpers.

Later when channel cutting, and tunneling started with drilling and blasting,

Table 1-5-1 TREND OF FIELD WORKS, PHASE III & IV

- SABEDAUNG -

	1975 – 1976 (May, June, July, Aug., Sep., Oct., Nov., Dec., Jan.)	Remarks
(MAIN WORKS)		
Earth Removal	May 13 – June 5	3,900 m^3
Soft rock Removal	June 6 – 8	5,200 "
Channel cut	June 19 – July 10	1,138 (2,844 t)
Tunneling	July 11 – Dec. 4; Jan. 19 – 23	91.7 m
Wall & Roof out	Nov. 23 – 2	2,085 t
Ore haulage	July 13 – 3	3,788 t
Ore Transport to Mill Site	Oct. 20 – 5	2,738 t
(MISCELLANEOUS)		
Topo & Ground Survey	May 13; June 5, 16; July 2, 11; Oct. 15, 19; Nov. 2, 2, 15, 24; Dec. 14; Jan. 13, 26	
Geological Survey	July 11 15; Aug. 7 8, 12 18; Sep. 1, 29; Oct. 6, 17, 23; Nov. 21 24; Jan. 26 29 – 3 6	
Tunnel Sampling	1 – 6; Jan. 27 – 5	
Assay	Aug. 6 – 15; Jan. 26 – 10	
Compressor Setting	June 14	
Mine Office Const.	June 14 – July 10	
Air Pipe Setting	June 19	
Water Pipe Setting	June 27; July 6	
Debris Transport.	June 10 – July 17	
Mine Road repair	June 10 – July 31	
Red Soil Transport.	June 24 – July 31	
Explosive Handling	June 21 – Jan. 23	
Fence for Roll-Stones	July 8 – 10	
Access Road Const.	May 13; June 16 – 18	
Miners Hut	July 14 – 17	

the mining work was undertaken by the following team arrangement:

Counterpart	2
Miner	6
Compressor operator	2
Shovel loader driver	2
Dumptruck driver	2
Guard and night watch	3
Total	17

The working time at the mine site was principally set on a one shift a day system based on 8 hours duty from 7.30 a.m. to 3.30 p.m.

During the bulldozing period, a 2 shift a day system was temporarily adopted to prolong the actual daily operating hours of the bulldozer.

At this time the operator's duty hours were practically limited to about 5 hours because of the heavier working conditions caused by the uncomfortable heat and vibration.

In principle, Sunday was a complete holiday, and Saturday was a half day at the working site, as well as at the mine office.

5-3 Mine Safety

In good consultation with our Burmese counterparts, much attention was paid to general mine safety. Also extreme care was paid to the handling of explosives to prevent accidents, as well as for the maintenance of the mine sanitation.

Fortunately, through the entire survey period, all of the planned work was accomplished without any accident or casualty. Utmost care was paid especially to the following points:

1) Book keeping and handling of explosives and detonators, defence and warn-

ing on blasting.

2) Inspection of after damp, checking and disposal of misfired detonators, and dynamite.

3) Inspection of underground scaling stone, and its preliminary removal to prevent accidents

4) Elimination of haulage accidents caused by shovel loader, dump truck, etc.

5) Prevention of both rolling stones on the surface and the inflow of surface mud into the tunnel.

6) Under ground illumination and ventilation

7) Correct procedure for each job, and right working postures.

On these maters, there is still much to be improved and the concern of the personnel for mine safety and sanitation can be increased.

5-4 Supply and Quality of Explosives and P. O. L.

In Burma, there is very strict control by the Burmese Army regarding the procurement, transportation, storage, and usage of all explosives.

In addition, import of explosives involves extremely strict procedures and the permission of the higher authorities concerned.

It is therefore almost impossible to choose arbitrarily the brand, quality, and quantity of explosives, caps, and fuses.

In the present survey, MMDC arranged to obtain 1,724kg of explosives and 2,000pcs of electric detorators beforehand from the magazine at the army camp to the north of Monywa city.

However, as mentioned already in Chapter 3, 1,066kg out of the total 1,724kg was highly hygroscopic ammonium nitrate explosive, Polar Ajax (manufactured in 1968), and could not be detonated,because it was in an almost completely damp condition.

As a result, the blasting was started with the residual amount of 658kg of dynamite (Gelignite NG 42%, manufactured in 1974).

From then on,the necessary explosives and detonators were supplied as required, from Rangoon and other places through the cooperation of Burmese Army.

The detonators obtained at the mine site were milli-second delay electric ones with 10 stages, so all the blasting was fired by electric detonators. Some of the detonators were very old with blue luster on their surface, and caused misfire so often as to result in trouble.

Some of them were found good in circuit tests even after they had been misfired and recovered from the mullock. This meant that the sparking match of the detontor must have become damp and become useless.

Regarding the fuel supply, the shortage of diesel oil invited trouble and sometimes it resulted in having to stop the compressor for a few days.

Gasoline and lubricant were properly supplied in most cases, and caused no trouble.

Sometimes water or even a slight mud, was found in the gasoline and diesel oil. This sort of contamination frequently caused serious trouble to the movement of vehicles.

Chapter 6 Work to be performed (Fig. I-1-2)

6-1 Setting of Working Site

A stone peg, later named as ST-1 (elevation of 103.856m ASL) was set firmly at the nearest location to JS-9 hole, on the side of the mine-road which runs along the south western foot of Sabedaung.

The line, connecting ST-1 and JS-9, was named as "center line", 0 ~ 0'

The level of 108m ASL was selected as the base level of the mine site.

The horizontal distance between ST-1 and JS-9 is 81m, and stripping has been done between these two stations. The original inclination of the topography was 23 degree along the center line within the stripped area.

Tunneling was also carried out later based on 108m ASL along the same center line.

6-2 *Stripping of Surface Soil (Fig. I-6-2)*

First of all, the bulldozer made a temporary access road of its own from the mine road to the basal level of the working site on 108m ASL. Then, stripping of the overburden was gradually carried out by bulldozers up to the vicinity of JS-9 hole (Collar elevation 132.3m ASL).

The vegetation was mostly of deciduous shrub with few tall trees, so without bush cutting the bulldozer pushed away all the vegetation toward the edge of the mine site together with surface earth.

Specifications of the work are as follows

1) Machinery : D-80 and D-85A Bulldozer

Both of them were often under repair, etc.

2) Performonce : Removed soil 3, 900 m^3

Average thickness removed Approx. 1. 8 m

Sectional area along center line 118. 7 m^2

3) Working Efficiency : Running hours of bulldozer 82 hr

Removed volume per hour 47. 6 m^3/hr

4) Period of Work : From May, 13 until June, 5

Full 16 days (244 m^3/d), excluding

8 days for repairs and holidays

6-3 Soft Rock Removal

Soft rock under the surface soil, which is mostly of the argillized, partly weathered, biotite porphyry was cut and removed by bulldozer.

The rock gradually changed into fresh, hard one in pace with depth from surface, but fortunately the cutting by the edge of the bulldozer proceeded better than was expected because of a well developed crack or joint in the rock.

To save the consumption of explosives, as well as the work-period, bulldozer was used as much as possible for soft rock cutting.

By this cutting, the formation of the floor level of the mine site on 108 m ASL was partly completed, and the bench below JS-9 hole was formed simultaneously.

Regarding the waste rock, overflow from the work site was transferred to the designated waste dump by dump trucks, after the training of shovel loader drivers had been completed.

Some of the waste was also used for repairs of the mine road.

Dimensions of Soft Rock Removal are shown as follows:

1) Machinery : D-85A/D-80 bulldozers, Occasionally the two were in use simultaneously

HL-8 Shovel loader 1

6.5 T Dump truck 2

2) Performance : Cut and Removed Volume 5,200 m^3

Average thickness Approx. 2.1 m

Sectional area along center line 129.5 m^2

3) Working Efficiency : Running hours of bulldozer 117 hr

Removed volume per hour 44.6 m^3/hr

4) Period of Work : June, 6 to June, 18, 10 full days

In addition, an access road to 108 m level from the mine road was also constructed for the easy passage of dump trucks.

6-4 Open Channel Cut Fig. I-6-1, I-6-2, Table I-6-1

Open Channel cutting toward JS-9 hole on 108 m level, along the disigned center line, was started at the time when Soft rock cutting by bulldozer was no longer possible. In practice on surface, drilling and blasting were carried out against the hard rock, and bulldozers gathered the broken waste. Then, after mucking by the shovel loader, dump trucks hauled the waste away from the mine site.

The channel cut was carried out for 27 m along the center line, both with its average bottom width of 4.5 m and its side-slope inclination of 45 to 50 degree.

The channel hit a shuttering fissure with big Slickenside, inclining toward this side at about 80 degree, which came out in a rectangular direction from the center line at the end of the channel.

So the stripping work with channel cutting was terminated on July, 10, when the final pit slope formed was approximately 45 degree in profile along the center line.

Dimensions of Channel cutting are as follows,

1)	Machinery :	Bulldozer D-85A	1
		Shovel loader HL-8	1
		Dump truck 6.5T	2
		Portable compressor, $17 m^3/min$	1
		Air-leg drill 317-D	2
		Blasting devices	1 set
2)	Performance :	Removed cut Volume	$1,137.6 m^3$ (2,844 t)
		Length of Channel	27 m
		Average width	6 m
		Average depth	4.2 m
		Sectional area along center line	$113.6 m^2$
		Hauled waste volume	$487 m^3$
3)	Working Efficiency :	Drilling and Blasting	$8.3 m^3$/min-shift (20.8t/ms)
		Running hours of bulldozer	25 hr
		Removed volume per hour	$27.6 m^3/hr$
		Running hours of shovel loader	29 hr
		Mucked volume per hour	$23.2 m^3/hr$
4)	Period of Work :	June, 19 to July, 10	19 full days

Because of the argillized, weatherd rock, drilling performance was good with both an efficient high speed of drilling, and with less amount of bit and rod consumed than had been anticipated.

For reference, the data on drilling and blasting can be shown as follows :

Number of holes	471
Length of holes	767.2 m (1.63m/hole)

Hole interval	0.96 m in avęrege
Volume per meter	1.48 m^3/m (3.71 t/m)
Damaged bit	4 pcs
Damaged rod	6 "
Pneumatic pressure	6.7 kg/cm^3
Drilling speed	75.7 cm/min
No. of blasting	20
No. of fired hole	471
Consumed detonator	485
Consumed explosives	207.2 kg
Explosives per Unit	182.5 g/m^3 (73 g/t)
No. of misfired detonators	14
Percentage of misfired detonators	2.9%

6-5 Tunneling (Fig. I-6-1, I-6-2, I-6-3, Table I-6-2)

Against the large fissure encountered at the end of the open channel, the tunneling started rectangularly towards JS-9 hole along the center line 0~0. In practice, a small sectional tunnel of 2m x 2m was first driven at the proposed site of portal for 2 rounds, then it was enlarged to 3m x 3m dimensions by additional drilling and blasting from the peripheral portion of the small tunnel. Because of the height of the additional holes, drilling was done on the waste which had been piled by the previous blasting.

Afterwards the rock was found to be solid enough, and the tunnel, with section of 3m x 3m, was cut directly.

Double staged drilling was adopted for the convenience of the miners.

A normal, standard drilling pattern was introduced for the large tunnel with

42 holes (in cluding one void hole) as shown in Fig. I-6-3.

Because of the frequent occurence of misfired detonators, 2 stage blasting was adopted in tunneling. The pilot blasting under Burnt Cut method was first applied, and then the lest of the holes were blasted after checking the former.

Whenever, the result of the pilot blasting was found to have been unsuccessful, the relevant holes were recharged and fired again to raise the rate of advance of the tunneling.

Typical charge for the holes may be explained as follows;

1) 10 sticks of gelignite to each of the four pilot holes, totaling 40 sticks

2) 7 sticks to each of the 6 holes around the centering pilot holes, totaling 42

3) 5 sticks to the residual 31 holes, totaling 155.

Grand total of 237 sticks (Approximate 50 lbs) of gelignite were in Principle consumend for one cycle of tunneling.

This means a consumption rate of 16.9 kg/m for a tunnel advance of 1.35 m, and 17.5 kg/m for a advance of 1.30 m, respectively.

The mucking of the blasted ore or waste was usually started after blowing in fresh air for about 30 minutes from the compressor close to the heading of the tunnel.

At the same time, the dust and after damp caused by blasting were guided to the surface by means of 2 local fans and an attached air duct.

Technical data on the tunneling are as shown in Table I-6-2.

6-6 Wall and Roof Cutting (Fig. I-6-4, Table I-6-3)

Wall and roof cutting is a means of mining in which those ore portions of either a tunnel or a stope are to be enlarged, usually the wall and or roof, to recover the ore.

In tunneling, it was ascertained that the earth was sufficiently firm and suitable for suppoting itself without timbering or logging even after the tunnel was enlarged into a wide space, although the tunnel lay in some places only a few meters and utmost only 30m from the overlying surface. Furthermore, the tunnel was included in a part of the oxidized zone with argillization.

Thus, it was decided to enlarge the ore zone of the tunnel to a width of 7.0 m with a height of 4 m, that is, to cut both sides of the tunnel wall by 2.0 m each and the roof by 1.0 m.

In practice, first, the small sized tunnel (2.0 m x 2.0 m) connecting both ends of the bigger tunnels was enlarged, then 6 auxiliary short cut tunnels were driven place to place right on the planning.

These 6 cuts totaled 15.8 m, and the object was to obtain a supplementary area of free face for more effective blasting in carrying out the enlargement. The holes for wall cutting were drilled parallel to the wall of these short cut tunnels with 1.2 m horizontal spacing. On the other hand, some holes for the other part of the main tunnel, were drilled at an angle oblique to the wall.

1.8 m long hexagonal 22 m/m rods with taper Carr bits were chiefly used in drilling, though some holes were drilled using 2.4 m long inserted carr bits integrated with the same hexagonal 22 m/m rods. The drilling was continued for several days to prepare a sufficient number of holes, and the blasting was undertaken selectively in 3 to 4 rows for each free face to ensure a good result.

Usually 40 to 50 holes were blasted at a time, simultaneously on both tunnel walls, in 6 to 8 rows in total.

Some roof holes were also blasted at the same time.

For the wall cutting holes, usually 5 sticks of gelignite (about 500 g in weight) were charged for a meter of the hole drilled. However, for the roof holes, four sticks or less were inserted to avoid possible overcharging caused by the oblique intersection of the hole to the roof of the tunnel.

The dust and afterdamp were also led to surface in a similar way as in tunneling.

Much attention was paid to the unstable scaling stone, especially on the roof of the tunnel, using specially manufactured light iron bars of 4 m length.

The technical data on Wall and Roof Cutting is shown in Table I-6-3.

6-7 Haulage and Storage of Ore

The broken ore from both tunneling and wall/roof cutting was hauled to the surface, and stored in the service yard near the portal.

It was separated into two groups by appearance, higher grade ore and lower grade ore.

Some of the ore was kept there for a few months, but oxidation of the ore was observed only on the surface of the pile. Oxidation was seldom found inside the pile. On October, 20, shortly before the completion of the pilot plant, the transportation of the aforementioned ore was started by the combination of a shovel loader and a 6.5 T dump truck from the mine site to the top yard at the pilot plant.

Whenever it was necessary, a bulldozer was called for either gathering or transferring of the piled ore at both store yards above mentioned. The ore was left as it was in the open.

6-8 Road Repairs

There was a rough mine road of 4 km in length between Sabedaung and the paved national road (here after in short, highway).

However, very often it was impossible for automobiles to use the road for as long as one or two days during and after rain, because of the muddy road condition.

To solve the problem, road repairs were carried out with the waste from the stripping work in Phase III. This also enabled the transportation of the construction materials and equipment for the pilot plant, as well as public traffic, even on rainy days. The broken waste from the mine site was carried by dump trucks to the necessary portions of the road, then it was spread flat to a 25 cm thickness by a motergrader.

A road roller pressed the new layer tight to form the foundation of the road.

On the other hand, red soil with small pebbles, the weathered laterite was also collected and hauled from the southern foot of Kyisindaung to those prepared portions to make good filtration coverage, and was spread and pressed flat into a 5 cm thickness to complete the repair work.

Thus, the mine road became 6 to 8 m in width, with improved surface condition, and enabled the smooth movement of the vehicles even in or immediately after rain fall.

The amount of waste and laterite soil, spread and rolled on the mine road, amounted to 750 m^3 and 1,040 m^3, respectively at the end of July, 1975.

Road repairs were also carried out around both the mine site and the pilot plant.

6-9 Work Performance

The outline of the work, performance and other technical information concerning mainly the post stripping field work can be given as follows;

1) Produced ore Table I-6-4

Out of 3,843t (average grade 0.63% Cu) of the excavated ore, 2,738t was hauled to the pilot plant.

Of this 2,138t (0.70% Cu) was supplied for the concentrating test.

The balance of the production, 1,105 t, was mostly piled in the service yard near the portal except for those 55t of broken ore which were left in the tunnel.

Though the average copper grade of the ore zone was as high as 0.86 %, that of the produced ore was 0.66 % lower than had been expected owing to the unforeseen occurrence of a very low grade leached zone deep in the tunnel with copper content of only 0.12 %. This low grade leached zone accounted for the high ratio of 27 % of the total produced ore.

2) Mining and Ore Haulage Table I-6-5

Out of 3,843 t of the excavated ore, 1,758 t came from the tunneling, the residual 2,085 t was produced by wall and roof cuttings.

On other hand, the ore hauled to the surface amounted to 3,788 t, of which 2,738 t was transferred to the mill site to meet the concentrating test at the pilot plant, which was conducted from November, 1975 to early February, 1976.

3) Main Data on Stripping and Underground Mining

The total work carried out in Phase III and IV resulted in the following figures;

Earth and Soft rock removal	9, 100 m^3
Open Channel Cut	1, 138 m^3 (2, 844 t)
Tunneling	91. 7 m (obtained ore 1, 758 t)
Wall and Roof Cutting	2, 085 t
Total No. of workers	3, 014 man-shifts
Consumed Explosives	2, 173. 79 kg
Consumed Detonators	4, 334 pcs

Of these, 1, 507 man-shifts, 1, 468. 87 kg, and 3, 070 pcs respectively were spent for the underground work. These figures correspond to 50 % of the man-shifts, 68 % of the explosives, and 71 % of the detonators, required for these phases.

Thus, it can be understood that the underground work constituted the major part of all the activities in Phase III and IV.

4) Working Efficiency (Table I-6-7)

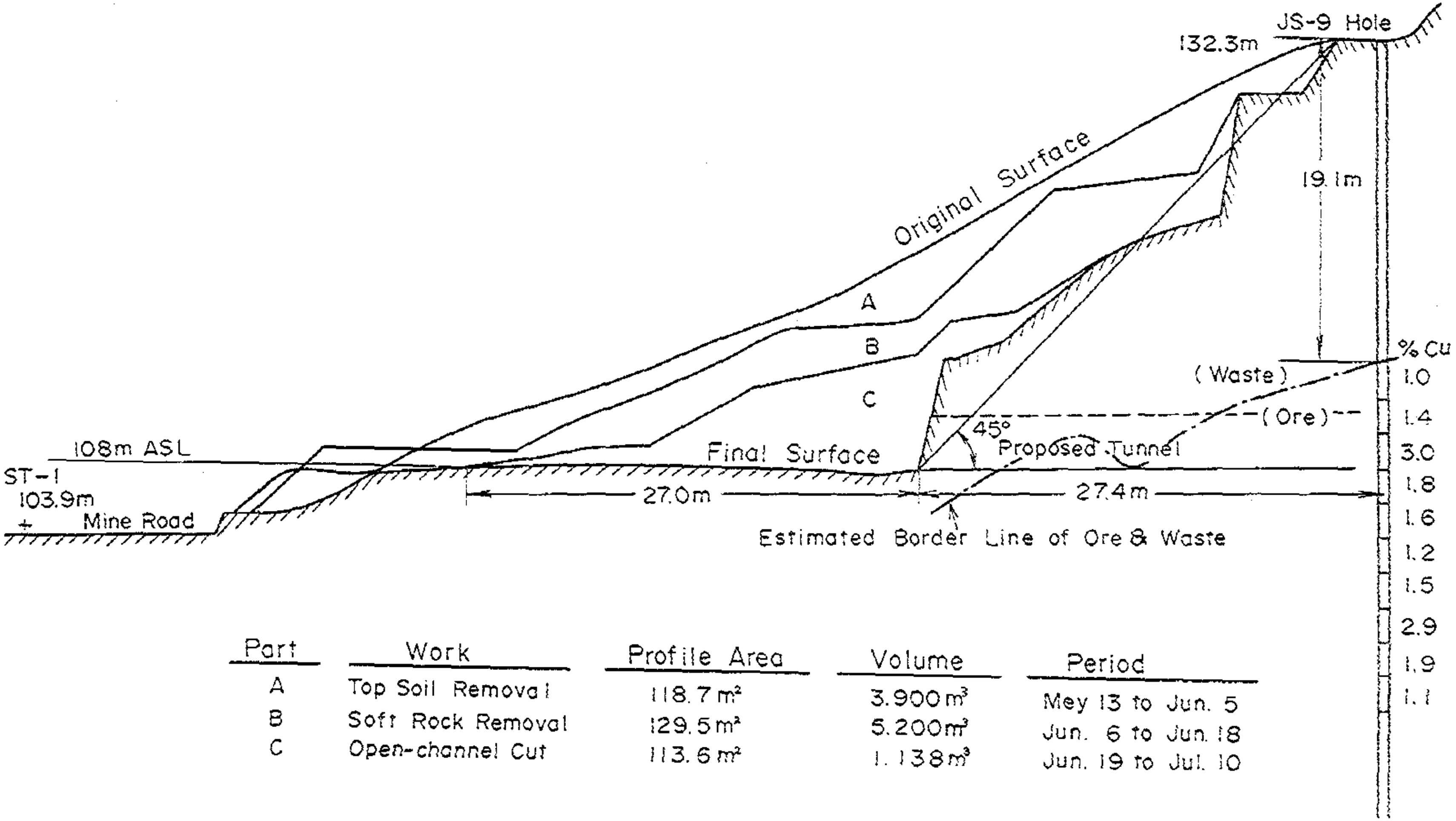
In most of the operations carried out, the working efficiency was found to be very low both in Main efficiency and in Total efficiency. This was because of the local situation in Burma i. e. the necessity to employ large numbers of workers, as well as to give them better opportunites for technical training.

The above mentioned Main efficiency is concerned only with the principal work of drilling and blasting, where as the Total efficiency includes the work carried out by operators and assistants of shovel loader, dumptrucks, and compressor as well as guards and night watchmen.

Not withstanding the above classification of the efficiencies, the operator of the bulldozer and his assistant were included under Main efficiency for the sake of convenience.

FIG. I-6-1

A SECTION ALONG CENTER LINE O—O'

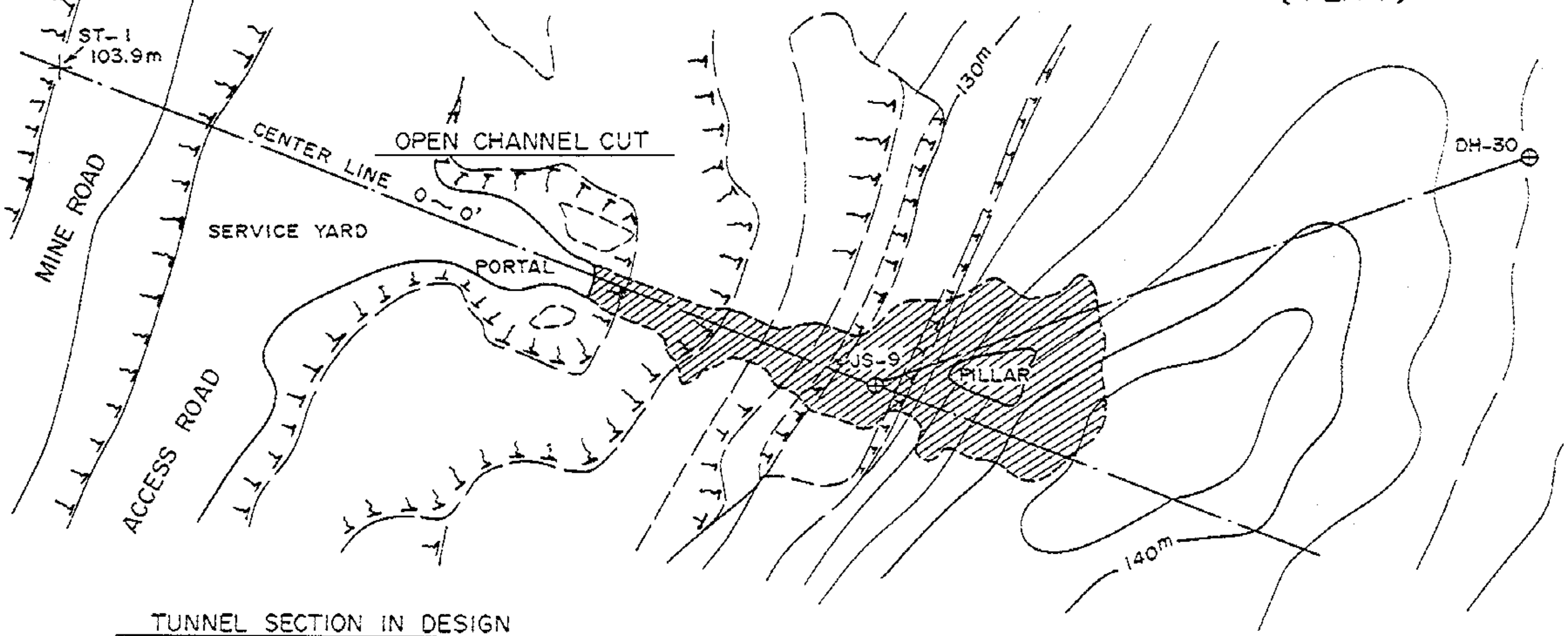


Part	Work	Profile Area	Volume	Period
A	Top Soil Removal	118.7 m²	3.900 m³	Mey 13 to Jun. 5
B	Soft Rock Removal	129.5 m²	5.200 m³	Jun. 6 to Jun. 18
C	Open-channel Cut	113.6 m²	1.138 m³	Jun. 19 to Jul. 10

FIG. I-6-2

RELATION OF OPEN CHANNEL AND TUNNEL

(PLAN)



TUNNEL SECTION IN DESIGN

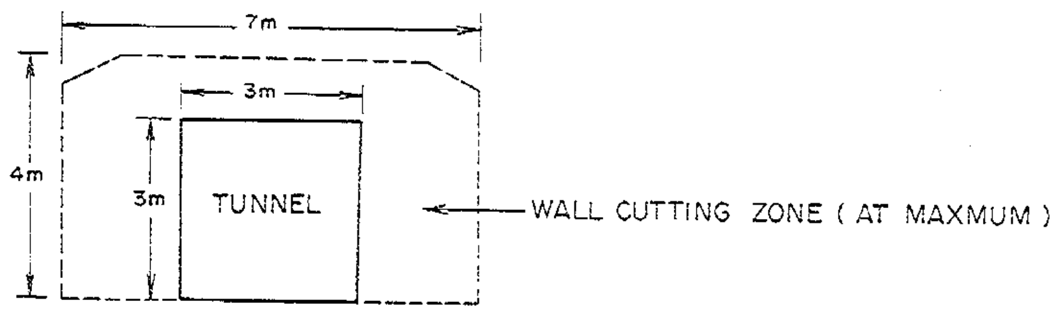
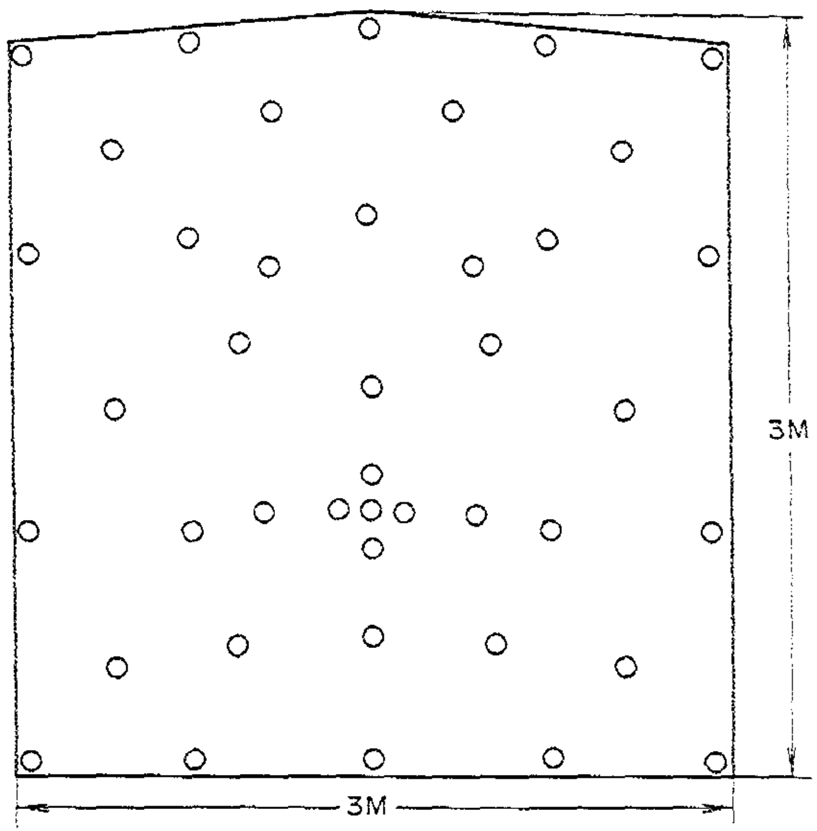
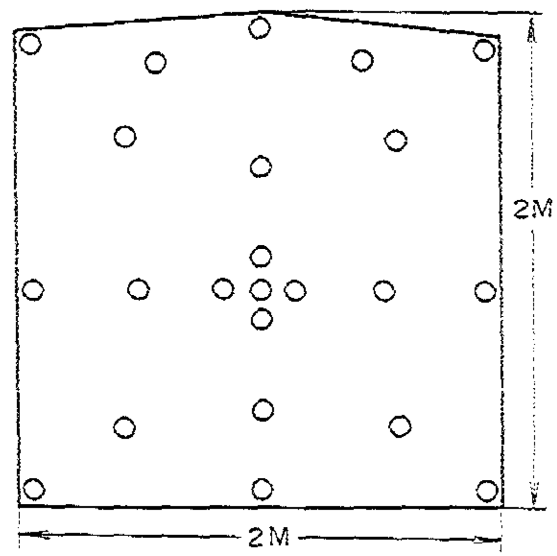


FIG. I-6-3 NORMAL DRILL HOLE PATTERN



Large Sectional Tunnel
No. of holes, 42



Small Sectional Tunnel
No. of holes, 23

Table I-6-1 Channel Cutting in Review

	Tonnage	Main Power		Performance		Consumption		Consumption/Meter		No. of Blasting	No. of Misfire	Drilling				No. of Drills	Remarks
		Main	Total	Main	Total	Explosive	Detonator	Explosive	Detonator			Holes	Length	Bit	Rod		
1975																	
May	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	
	t					kg	pcs.	kg/t	pc/t			hole	m	pcs.	pcs.		
June	2,163	70	175	30.9	12.4	93.53	212	0.043	0.096	8	4	210	346.5	20	20	14	
July	662	62	163	10.7	4.1	105.94	252	0.160	0.380	10	8	242	392.2	38 (4)	38 (6)	12	( ) damaged
Aug.	19	5	12	3.8	1.6	7.75	21	0.408	1.105	2	2	19	28.5	2	2	2	
Total	2,844	137	350	20.8	18.1	207.22	485	0.073	0.171	20	14	471	767.2	60 (4)	60 (6)	28	
												1.63 m/hole					
												hole/t	m/t	pc/t	pc/t		
												0.166	0.270	0.021	0.021		
								t/kg	t/pc			t/hole	t/m	t/pc	t/pc		
								13.7	5.86			6.04	3.71	47.4	47.4		

Table I-6-2 Tunneling in Review

	Advance	Main Power		Performance		Consumption		Consumption/Meter		No. of Blasting	No. of Misfire	Drilling				No. of Drills	Remarks
		Main	Total	Main	Total	Explosive	Detonator	Explosive	Detonator			Holes	Length	No. of Bit	No. of Rod		
1975	m			m/t	m/t			kg									( ) damaged
July	15.4	106	228	0.15	0.07	243.37	501	15.80	32.53	31	29	491	772.3	33(1)	33(3)	28	
Aug.	16.9	116	286	0.15	0.06	293.39	687	17.36	40.65	35	33	584	889.9	35	35(1)	40	Tunnel of 3m x 3m 67.3m
Sep.	22.6	120	364	0.19	0.06	367.63	840	16.27	37.17	30	9	669	1,047.1	33	33(2)	44	
Oct.	12.4	83	294	0.15	0.04	238.43	491	19.23	39.60	9	7	486	793.6	20	20(3)	30	16 pieces of bits were re-ground in Oct.
Nov.	21.8	150	287	0.15	0.08	289.12	482	13.26	22.11	26	13	341	550.2	18(1)	18(2)	29	
Dec.	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Tunnel of 2m x 2m 24.4m
1976 Jan.	2.6	22	48	0.12	0.05	36.93	69	14.20	26.54	8	-	69	110.4	3	3	2	
Feb.	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Bit 31 used Rod 25 used
Total	m 91.7	597	1,507	0.15	0.06	1,468.87	3,070	16.02	33.48	139 time/m 1.52	91 pcs/m 0.99	2,640 hole/m 28.8	4,163.5 m/m 45.4	142(2)	142(11)	173	
3m x 3m Tunnel	m 67.3	425	1,172	0.158	0.057	1,142.82	2,519	16.98	37.43	105 time/m 1.56	78 pcs/m 1.16	2,230 holes/m 33.1	3,502.9 m/m 52.0	121(1)	121(9)		
2m x 2m Tunnel	m 24.4	172	335	0.142	0.073	326.05	551	13.36	22.58	34 time/m 1.39	13* pcs/m 0.53	410 holes/m 16.8	660.6 m/m 27.1	21(1)	21(2)		

Average bit life 208.2 m/pc., Average rod life 258.2 m/pc., Regrinding 403.4 m/time

Table I-6-3 Wall Cutting in Review

	Tonnage	Main Power		Performance		Consumption		Consumption/Meter		No. of Blasting	No. of Misfire	Drilling				No. of Drills	Remarks
		Main	Total	Main	Total	Explosive	Detonator	Explosive	Detonator			Holes	Length	Bit	Rod		
1976	t			t/main	t/main	kg	pcs	kg	pcs				m				
Sep.	65	5	14	12.9	4.6	15.41	38	0.239	0.589	3	0	34	49.6	3	3	2	
Oct.	-	37	96	-	-	-	-	-	-	-	-	406	661.6	22	22	14	
Nov.	54	25	45	2.2	1.2	12.99	26	0.239	0.478	2	-	79	180.0	10	10	4	
Dec.	1,172	221	403	5.3	2.9	279.82	387	0.239	0.330	23	8	175	297.2	17	17	17	
1978 Jan.	794	145	279	5.5	2.8	189.48	328	0.239	0.413	18	1	188	298.8	13	13	13	
Feb.	-	5	59	-	-	-	-	-	-	-	-	23	36.8	1	1	1	
Total	t 2,085	436	896	4.8	2.3	kg 497.70	pcs 779	0.239	0.374	46	9	905	1,524.0	53	53	51	

1.68 m/hole

		time/t	pc	hole/t	m/t
		0.022	0.0043	0.434	0.731
t/kg	t/pc	t/time	t/pc	t/hole	t/m
4.19	2.67	45.3	231.6	2.30	1.37

Table I-6-4　Balance Sheet of Ore Production

	Entrance Portion (10. 3m)			Ore Portion			Leached Low Grade Portion			Total			Remarks
	Tonnage	Grade	Metal content	Tonnage	Grade	Metal content	Tonnage	Grade	Metal content	Tonnage	Grade	Metal content	
Waste	t 232	Cu % 0. 10	kg 232	-	-	-	-	-	-	232	0. 10	232	Roof cutting started at 20m from the portal
Piled ore (Portal)	-	-	-	586	0. 75	4. 246	250	0. 12	300	818	0. 56	4. 546	
Dressed ore	-	-	-	1. 611	0. 89	14. 334	527	0. 12	633	2. 138	0. 70	14. 967	Subtotal 2. 738 t 19. 152 kg
Piled ore (Mill)	-	-	-	450	0. 89	4. 005	150	0. 12	180	600	0. 70	4. 185	
Broken ore (Tunnel)	-	-	-	10	0. 89	89	45	0. 12	46	55	0. 25	135	
Total	232	0. 10	232	2, 639	0. 86	22, 674	972	0. 12	1. 159	3. 843	0. 63	24. 065	
For reference				2, 639 73%	0. 86	22, 674	972 27%	0. 12	1. 159	3. 611 100%	0. 66	23. 833	

Table I-6-5 Mining and Ore Haulage

Unit : ton

	Amount of Excavated Ore			Hauled Ore to Surface	Broken Ore left in Tunnel	Hauled Ore to Mill	Piled Ore at Mine Site	Remarks
	Tunnel	Wall cut.	Sub-total					
1975								
July	346	-	346	331	15	-	331	Tunneling started on July 11th. 1975
Aug.	380	-	380	379	16	-	710	
Sep.	509	65	574	439	151	-	1,149	
Oct.	279	-	279	376	54	11	1,514	Haulage to Mill started on Oct. 20th, 1975
Nov.	218	54	272	312	14	283	1,543	
Dec.	-	1,172	1,172	894	292	1,205	1,232	
1976 Jan.	26	794	820	822	290	866	1,188	
Feb.	-	-	-	235	55	373	1,050	
Total	1,758	2,085	3,843	3,788	55	2,738	1,050	
				3,843 Excavated ore		3,788 Hauled ore to Mill		

Table I-6-6 Main Data on Stripping & Underground Mining

	Earth Removal				Channel Cut				Tunneling				Wall Cutting				Total		
	Volume (m^3)	Man Power (men)	Explo-sive	Detona-tor	Tonnage	Man Power (men)	Explo-sive (kg)	Detona-tor (pcs)	Advance (m)	Man Power (men)	Explo-sive (kg)	Detona-tor (pcs)	Tonnage (t)	Man Power (men)	Explo-sive (kg)	Detona-tor (pcs)	Man Power (men)	Explo-sive (kg)	Detona-tor (pcs)
1975 May	2. 600	92	-	-	-	-	-	-	-	-	-	-	-	-	-	-	92	-	-
June	6. 500	169	-	-	2. 163	175	93. 53	212	-	-	-	-	-	-	-	-	344	93.53	212
July	-	-	-	-	662	163	105. 94	252	15. 4	228	243. 37	501	-	-	-	-	391	349.31	753
Aug.	-	-	-	-	19	12	7. 75	21	16. 9	286	293. 39	687	-	-	-	-	298	301.14	708
Sep.	-	-	-	-	-	-	-	-	22. 6	364	367. 63	840	65	14	15. 41	38	378	383.04	878
Oct.	-	-	-	-	-	-	-	-	12. 4	294	238. 43	491	-	96	-	-	390	238.43	491
Nov.	-	-	-	-	-	-	-	-	21. 8	287	289. 12	482	54	45	12. 99	26	332	302.11	508
Dec.	-	-	-	-	-	-	-	-	-	-	-	-	1. 172	403	279. 82	387	403	279.82	387
1976 Jan.	-	-	-	-	-	-	-	-	2. 6	48	36. 93	69	794	279	189. 48	328	327	226.41	397
Feb.	-	-	-	-	-	-	-	-	-	-	-	-	-	59	-	-	59	-	-
Total	9. 100	261	-	-	2. 844 (1. 138m^3)	350	207. 22	485	91. 7 (1,758 t)	1. 507	1.468. 87	3. 070	2. 085	896	497. 70	779	3. 014	2.173.79	4. 334

Table I-6-7 Working Efficiency

Unit of Man Power : Man-shift

	Earth Removal					Channel Cut					Tunneling					Wall Cutting					Total	
	Volume	Man Power		Performance		Tonnage	Man Power		Performance		Advance	Man Power		Performance		Tonnage	Man Power		Performance		Man Power	
	(m^3)	Main	Total	Main (m^3)	Total (m^3)	(t)	Main	Total	Main (t)	Total (t)	(m)	Main	Total	Main (m)	Total (m)	(t)	Main	Total	Main (t)	Total (t)	Main	Total
1975 May	2,600	29	92	89.7	28.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	29	92
June	6,500	77	169	84.4	38.5	2,163	70	175	30.9	12.4	-	-	-	-	-	-	-	-	-	-	147	344
July	-	-	-	-	-	662	62	163	10.7	4.1	15.4	106	228	0.15	0.07	-	-	-	-	-	168	391
Aug.	-	-	-	-	-	19	5	12	3.8	1.6	16.9	116	286	0.15	0.06	-	-	-	-	-	121	298
Sep.	-	-	-	-	-	-	-	-	-	-	22.6	120	364	0.19	0.06	65	5	14	13.0	4.6	125	378
Oct.	-	-	-	-	-	-	-	-	-	-	12.4	83	294	0.15	0.04	-	37	96	-	-	120	390
Nov.	-	-	-	-	-	-	-	-	-	-	21.8	150	287	0.15	0.08	54	25	45	2.2	1.2	175	332
Dec.	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1,172	221	403	5.3	2.9	221	403
1976 Jan.	-	-	-	-	-	-	-	-	-	-	2.6	22	48	0.12	0.05	794	145	279	5.5	2.8	167	327
Feb.	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5	59	-	-	5	59
Total	9,100	106	261	85.8	34.9	2,844	137	350	20.8	8.1	91.7	597	1,507	0.15	0.06	2,085	438	896	4.8	2.3	1,278	3,014

Chapter 7 Technical Aspects

7-1 Workability of Soil and Rock

In stripping, it was found that the surface soil had a thickness of around 1.8 m, and the underlying soft rock about 2.1 m which could have been cut by bulldozer.

These thicknesses are of variable nature, depending greatly on the geology near the surface, place to place, as well as on the type and capacity of the construction machinery to be used together with its usage.

However, from the field experience at Sabedaung it can be fairly certainly deduced that the shallow portion of 4 to 5 meter from surface can be efficiently removed without blasting by heavier machinery, except some exposures of hard rock.

On the other hand, in the underground mining work, it had been felt necessary at first to prepare for underground timbering, and some square posts and planks were collected at the motor pool near the portal.

However, it proved to be unnecessary to use them for support, because of the satisfactory ground condition.

The ground was argillized porphyry, partly of the oxidized zone which could be characterized by brown colored gossan, and the location was close to the surface. Despite all these disadvantages, the ground was really firm, though not too hard, allowing the tunnel to be self-supporting even after the enlargement of the walls and roof.

7-2 Bulldozing

The bulldozer played an inportant role in overburden stripping, soft rock removal, and open channel cutting, in which blasted waste was dozed out of the channel by the bulldozer, then swept by a shovel loader.

The working efficiency in overburden stripping was found to be 244 m^3/day with a dozing distance of 30 to 50 meters. The average hourly disposal was approximately 48 m^3.

In consideration of the inevitably increasing distance, dozing scrapers, shovel dozers, etc. may have to be introduced in the future development of the mine.

Furthermore the skill of the bulldozer operators greatly affects the performance, so that the improvement of their ability through field training will also be indispensable.

7-3 Drilling (Table I-7-1)

Both in the open channel cut and in tunneling, an air-leg drill, ASD-317 was used for drilling with a Carr type tapered bit.

Though partially very hard because of the silicified rock, drilling was easily done at a comparatively high speed for the most of the underground rock because of the prevailing argillization.

In wet drilling it took only 2-1/2 ~3 minutes utmost for the drill to complete the full hole length of 1.65 m after collaring the hole.

This means a drilling speed of 55 to 66 cm/min. Table I-7-1 shows in detail the performance of the various bits and rods used in Phase III and IV.

The average meter, drilled by these introduced bits of 31 piecs and rods of 27 sticks were 208 m and 258 m respectively per a piece or a stick where the total drilled length was recorded as 6,454.7 m. And also per a damaged number of bit and rod the drilled length was favourably proved as long as 1,076 m in bit, and 380 m in rod.

Because of the contrastingly quicker wear on the holding metal matrix of the bits compared to the corresponding tungsten carbide chips, the use of slightly harder matrix may later become necessary for soft rock drilling at the Monywa mine.

Concerning the rods, damage usually occured a few centimeter from the shank. However, on the whole, the consumption of bits and rods was proved to be very small in relative amount, from which it could be concluded that the drilling in the region would be comparatively easy except in some silicified zones.

There was little trouble during or after drilling caused by collapse or choking of the hole, and this also supported higher drilling efficiency.

7-4 Blasting (Table I-7-2)

Both channel cutting and tunneling were carried out by using electric detonators and gelingnite, as shown in Table I-7-2, "Consumption of Explosives."

Out of the total consumption of 1,468.87 kg of explosives and 3,070 pcs of detonators, as already mentioned in 6-9, 68 % and 71 % of them respectively were used for tunneling.

In this major tunneling work, the consumption of explosives per meter of advance was 16.98 kg for a 3 m x 3 m sectional tunnel, and 13.36 kg for a 2 m x 2 m tunnel, respectively on average. These consumption rates may be evaluated as being low, at least in comparison with other cases in Japan.

For reference, specifications, together with relevant powder factors, are shown in the attached list, "Comparison of Tunneling Data," which has been compiled from the data of mines with massive deposits in Japan (1974).

Though the consumption rate depends much on the nature of the ground, sectional size of tunnel, metheods of drilling and blasting (namely, number of free faces), quality of explosives, etc., it can be understood that the nature of the rock at Sabedaung is generally favourabe for blasting purposes, showing a good blastability.

On the other hand, in the case of wall/roof cutting and open channel cutting, the consumption rates were 239 g/t and 73 g/t, respectively.

From these quite different values, it may be understood that the consumption rate of explosives depends greatly on the blasting method, especially on the choice of the free face in blasting, as well as the depth of the location of blasting from the original ground level.

In consideration of both the size of the broken ore and the greater depth of the coming blasting site, it would be reasonable to assume the consumption at 100 g/t on average for the proposed open pit mining at Sabedaung.

7-5 Mucking and Haulage Table I-7-3., I-7-4.

The data on Mucking by shovel loader, together with Ore Haulage by dump-truck, both concerning the underground mining, are shown in Table I-7-3. and I-7-4., respectively.

The data can be summarized as follows,

	Hauled Ore	No. of Trips	No. of Cars	Ore/Trip	Trip/Car
Shovel loader	3,788 t	3,505	143	1.08 t	24.5
Dump truck	2,738 t	484	61	5.66 t	7.9

The performance of the shovel loader per man-shift was found to be 14 to 15 ton at most, because the loader was usually operated by the driver and his assistant operater, and ran in switch back trips underground owing to the design of the tunnel. In the case of the dump truck, the performance varied from day to day on account of its multiple usage, not only for the ore haulage but also for the transportation of materials, personnel, etc.

It depended also on the requirement of ore at the mill, for the testing of both quality and quantity. Thus, on some days, there was no ore transportation to the mill, and on other days 2 trucks were engaged in the haulage of the ore.

7-6 Oxidation of Ore

Chalcocite, the copper ore, is apt to change its surface colour from the original black into pale greenish blue by oxidation, shortly after it has been mined. Thus, it was often observed that the wall of the tunnel, or the surface of the broken ore changed colour even within one or two months after excavation.

Sometimes the water remaining on the floor of the tunnel, was also observed to have a blue colour because the copper ion of the ore is soluble in water.

Due to the above facts, it is expected that in future the extraction or recovering of the copper ion will be achieved by leaching.

On the other hand, it is recommended not to expose the ore body for a long time after stripping, in order to restrict unnecessary oxidation of the mineable ore.

Table I-7-1 Data on Drilling

	Channel Cut						Tunneling						Wall Cutting						Total						Remarks
		Drilling						Drilling						Drilling						Drilling					
	Tonnage (t)	Drills	Holes	Length	Bit	Rod	Advance (m)	Drills	Holes	Length	Bit	Rod	Tonnage (t)	Drills	Holes	Length	Bit	Rod	Advance of Work (t/m)	Drills	Holes	Length	Bit	Rod	
1975 June	2,163	14	210	346.5	20	20	-	-	-	-	-	-	-	-	-	-	-	-	2,163/-	14	210	346.5	20	20	31 bits and 25 rods were put in use.
July	662	13	242	392.2	38(4)	38(6)	15.4	28	491	772.3	33(1)	33(3)	-	-	-	-	-	-	662/15.4	40	773	1,164.5	71(5)	71(9)	
Aug.	19	2	19	28.5	2	2	16.9	40	584	889.9	35	35(1)	-	-	-	-	-	-	19/16.9	42	603	918.4	37	37(1)	( )... damaged
Sep.	-	-	-	-	-	-	22.6	44	669	1,047.1	33	33(2)	65	2	34	49.6	3	3	65/22.6	46	703	1,096.7	36	36(2)	
Oct.	-	-	-	-	-	-	12.4	30	486	793.6	20	20(3)	-	14	406	661.6	22	22	- /12.4	44	892	1,455.2	"16" 42	42	" " (3)... Regrinding
Nov.	-	-	-	-	-	-	21.8	29	341	550.2	18(1)	18(2)	54	4	79	180.0	10	10	54/21.8	33	420	730.2	28(1)	28(2)	
Dec.	-	-	-	-	-	-	-	-	-	-	-	-	1,172	17	175	297.2	17	17	1,172/-	17	175	297.2	17	17	
1976 Jan.	-	-	-	-	-	-	2.6	2	69	110.4	3	3	794	12	188	298.8	13	13	794/2.6	15	257	409.2	16	16	
Feb.	-	-	-	-	-	-	-	-	-	-	-	-	-	1	23	36.8	1	1	-	1	23	36.8	1	1	
Total	2,844	28	471	767.2	60(4)	60(6)	91.7	173	2,640	4,163.5	142(2)	142(11)	2,085	51	905	1,524.0	66	66	4,929/91.7	252	4,016	6,454.7	268(6)	268(17)	

Table I-7-2 Consumption of Explosives

	Channel Cut					Tunneling					Wall Cutting					Total	
	Tonnage	Explosive		Detonator		Advance	Explosive		Detonator		Tonnage	Explosive		Detonator		Explosive	Detonator
	t	kg/t	kg	pc/t	pcs	m	kg/t	kg	pc/t	pcs	t	kg/t	kg	pc/t	pcs	kg	pcs
1975 June	2,163	0.043	93.5	0.098	212	-	-	-	-	-	-	-	-	-	-	93.53	212
July	662	0.160	105.94	0.380	252	15.4	15.80	243.37	32.53	501	-	-	-	-	-	349.31	753
Aug.	19	0.408	7.75	1.105	21	16.9	17.36	293.39	40.05	687	-	-	-	-	-	301.14	708
Sep.	-	-	-	-	-	22.6	16.27	367.63	37.17	840	65	0.239	15.41	0.589	38	383.04	878
Oct.	-	-	-	-	-	12.4	19.23	238.43	39.60	491	-	-	-	-	-	238.43	491
Nov.	-	-	-	-	-	21.8	13.26	289.12	22.11	482	54	0.239	12.99	0.478	26	302.11	508
Dec.	-	-	-	-	-	-	-	-	-	-	1,172	0.239	279.82	0.330	387	279.82	387
1976 Jan.	-	-	-	-	-	2.6	14.20	36.93	26.54	69	794	0.239	189.48	0.413	328	226.41	397
Feb.	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total	2,844	0.073	207.22	0.171	485	91.7	16.02	1,468.87	33.48	3,070	2,085	0.239	497.70	0.374	779	2,173.79	4,334

[Reference] Comparison of Tunneling Data

Items \ Name of Mine		Shakanai	Monywa (Sabedaung)	Kosaka	Kamaishi	Kamioka (Tochibora)
Geology	-	Tuff (Kuroko type)	Altered porphyry	Ryolite, Tuff-breccia (Kuroko type)	Porphirite, Diorite, Slate (Skarn type)	Gneiss, Hedenbergite (Skarn type)
Dimension of tunnel	m	2.6 x 2.4 & 3.2 x 2.8	2.0 x 2.0 & 3.0 x 3.0	2.8 x 2.5	2.1 x 1.8 & 3.0 x 2.5	2.2 x 2.2 & 3.0 x 4.2
No. of drilling	holes	20 - 45	23 & 42	22 - 28	23	35 & 42
Length of a hole	m	0.8 - 1.3	1.6	1.10 - 1.20	1.35	1.5 - 2.5
Bit gauge	m/m	38	32 & 34 (tapered) cross type	38 can type	36 can type	27 - 32 can & 45 cross
Rod gauge	m/m	22 hexagonal	22 hexagonal	22 hexagonal	22 hexagonal	19, 22, 32 hexagonal
Drill in use	-	ASD-F7	ASD-317	TY-85 & 76	TY-24 LD	R-70, F-12, TY-90
Pneumatic pressure	kg/cm^2	5.0	5 - 6	5 - 6	5 - 7.5	8.2
Drilling speed	cm/min	80	55 - 66	40	20 - 30	100
Blasting method	-	Pyramid cut, (elec.)	Burnt cut (elec.)	V cut (elec.)	Pyramid cut (fuse)	Burnt cut (elec.)
Explosives	-	Kiri No. 3, AN-FO	Gelignite (NG 42%)	Kiri No. 3, AN-FO	Enoki No. 2 & No. 3	Sugi & AN-FO
Consumed explosive	kg/m	14.3	13.4 & 17.0 (Av. 16.0)	20.1	25.4	33.6
Main performance	m/MS	1.20	0.15	0.29	0.47	0.85
Total performance	m/MS	0.32	0.06	0.26	0.47	0.57

MS .... man shift

Table I-7-3 Shovel Loader Performance

Period	Calender day	Advance of Tunnel	Shovel Loader				Hauled ore/month
			No. of S.L.	Trips	Sub-total	ton/trip	
1975 7/13-19	7	5.2 m	4	122			
20-26	7	4.4	4	103	331	1.00	331 t
27-31	5	4.6	4	106			
8/1 - 2	2	1.5	2	35			
3 - 9	7	3.9	4	91			
10-16	7	6.5	5	151	421	0.90	379 "
17-23	7	3.9	5	91			
24-30	7	2.3	6	53			
8/31	1	-	-	-			
9/1 - 6	6	4.1	5	90			
7 - 13	7	4.9	5	84			
14-20	7	6.5	6	123	439	1.00	439 "
21-27	7	4.7	5	142			
28-30	3	2.4	2	0			
10/1 - 4	4	-	2	81			
5 - 11	7	6.0	6	100			
12-18	7	3.9	5	136	396	0.95	376 "
19-25	7	2.5	3	79			
26-31	6	-	-	0			

Period	Calender day	Advance of Tunnel	Shovel Loader				Hauled ore/month
			No. of S.L.	Trip	Sub-total	ton/trip	
11/1	1	--	1	40			
2 - 8	7	5.4	5	80			
9 - 15	7	9.9	6	110	416	0.75	312 t
16-22	7	3.9	4	45			
23-29	7	2.6	5	141			
11/30	1	0	0	0			
12/1 - 6	6	-	6	113			
7 - 13	7	-	5	206			
14-20	7	-	5	141	688	1.30	894 "
21-27	7	-	4	144			
28-31	4	-	3	84			
1976 1/1 - 3	3	-	-	-			
4 - 10	7	-	6	212			
11-17	7	-	6	245	632	1.30	822 "
18-24	7	2.6	4	145			
25-31	7	-	4	30			
2/1 - 7	7	-	6	182			
8 - 10	3	-	-	-	182	1.29	235 "
Total	days 213	m 91.7	cars 143	3.505	3.505	1.081	3.788 t

Table 1-7-4 Dump Truck Performance

Period	Calender day	No. of cars	Trips	Monthly total	ton/car	Monthly hauled ore	Remarks
1975							Used cars Ton/car
10/20-25	6	1	2	2	5.65	11 t	1 11.3
26-31	6	-	-				
11/1	1	1	8				
11/2 - 8	7 7	1	5				
9 - 15	7	2	12	50	5.66	283 "	11 25.7 (car / day car/day) (50 / 11 = 4.6)
16-22	7	4	10				
23-29	7	3	15				
30	1	-	-				
12/1 - 6	6	4	78				
7 - 13	7	4	34	213	5.66	1,205 "	17 70.9 (car / day car/day) (213 / 17 = 12.5)
14-20	7	2	21				
21-27	7	4	40				
28-31	4	3	40				
1976							
1/1 - 3	3	-	-				
4 - 10	7	6	61				
11-17	7	5	47	153	5.66	866 "	16 54.1 (car / day car/day) (153 / 16 = 9.6)
18-24	7	3	30				
25-31	7	2	15				
2/1 - 7	7	4	66		5.66	373 "	4 93.4 (car / day car/day) (66 / 4 = 16.5)
Total	111	49	car 484		5.66	2,738 t	49 55.9 (car / day car/day) (484 / 49 = 9.9)

Chapter 8 Geological Survey

8-1 Outline of Geological Survey

8-1-1 Underground Geological Survey

The underground geology was sketched in a scale of 1/100 for the whole tunnel, consisting of an exploratory drift of 91.7 m and an enlargement of 166 m^2. The data was then compiled into an underground geological map of 1/500 (PL I-8-1).

8-1-2 Underground Sampling

A total of 275 samples were assayed for copper (Cu). Of these 209 were collected from the side walls and another 66 from the back of the tunnel. In addition to Cu, some samples were assayed for gold (Au), silver (Ag), iron (Fe), sulphur (S), arsenic (As), zinc (Zn), and lead (Pb). Total number of analyzed elements amounted to 397.

8-1-3 Period of Survey

From November, 1975 to February, 1971.

8-1-4 Data Analysis

1) Thin sections of the principal rock types were made for microscopic examination to observe their petrographical natures and modes of alteration.

2) Polished sections of representative ores were made to examine microscopically their mineral compositions and textures.

3) The modes of ore emplacement were studied by preparing underground geological and assay maps of 1/100, which were supplemented by a geological plan of the Sabedaung Ore Deposit on the level of 110.m.S.L., together with its geological sections of 1/500 also prepared for this purpose.

8-2 Ore Deposits

8-2-1 General Remarks

A description of the geology of the Monywa District will be omitted here, as it was given already in the previous report Vol. I, and vol.II, the First and Second Year Phases.

The ore deposits of Monywa consist of the three major ones of Sabedaung, Kyisindaung, and Letpadaung, which have developed in and around the volcanic domes scattered in the Monywa Basin.

All of these domes are lava domes of biotite porphyry formed by the volcanism during the Pliocene, Thertiary Period, and it was established through geological surveys in the First and Second Year Phases, that the ore deposits are epithermal, network and disseminated deposits of copper, accompanied by rhyolite dykes intruding in and around the lava domes.

Pyrite and secondary chalcocite derived from supergene enrichment are the principal minerals of the deposits, which lie more or less flat in lenticular forms, and are overlain by leached zones ranging from 10 to 100 m deep below the surface.

8-2-2 Mineralization

Geological surveys performed from the First Year Phase to the Third Year Phase have established the relation of mineralization to the volcanism, and the chronological succession of events can be explained as follows;

1) activity of hornblende-biotite porphyry at the stage of upper Magygon Formation,

2) eruption of biotite porphyry to form the domes,

3) intrusion of rhyolite dykes in and around the lava domes and the associated mineralization, and

4) post-mineralization activity of rhyolite.

Surrounding the bunches of rhyolite dykes intruding into the lava domes, alteration haloes have been formed which consist of such alternating zones of silicification, alunitization, and argillization from the inner towards the outer zones, and the ore is concentrated where the silicification is intense. As mentioned above, the copper mineral most widely recognized is chalcocite, but some pyrite includes minute grains of unsolved chalcopyrite, which justifies the assumption that chalcopyite is the primary copper mineral.

8-3 Sabedaung Deposit

This deposit is emplaced in a lava dome of biotite porphyry, penetrating and overlying the upper tuff of Magygon Formation. The dome is one of a series of lava domes of Kyisindaung, Letpadaung, etc., derived from the same volcanism.

The Sabedaung lava dome has a dimension of about 400 m in east-westerly width, north-southward elongation of about 600 m, and relative height of 80 m above the flat land. The deposit has an elongation of 500 m in north-south, east-westerly width of 350 m, average thickness of 60 m, and leached zone of average thickness of 26 m.

The ore reserves of this deposit have been calculated at 25.7 million tons with a grade of 1.01 % Cu, based upon the data of diamond boring.

Although 53 holes in all have beendrilled, thereserves should be classified as probable ore, because the deposit has been explored by diamond boring only.

Several dykes of rhyolite, containing locally the breccias of biotite porpyry and penetrating the dome of biotite porphyry, are recognized in the Sabedaung Ore Deposit. It has been established through the previous surveys, and even more through the current survey, that in the directions of dykes, 3 systems of N 40°E, N 30°W, and N 70°E are outstanding. The major one of 20 m wide is located nearly in the center of the dome, in association with minor ones of about 1 m wide, with the strikes of N 30°W and N 70°E, and the mineralization has spread from these dykes.

8-4 Underground Geology

8-4-1 System of Underground Geological Survey

An underground map of scale 1/100 was made beforehand by surveying the tunnel with transit-compass, measuring tape, and automatic level (self-adjusting level). Geological sketches of the back and side walls of the tunnel were drawn on this underground map, which was then compliled into an underground map of scale 1/500.

8-4-2 Underground Geology

The rocks observed in the tunnel are dome forming biotite porphyry with dyke forming rhyolite penetrating the former. Some parts of the rhyolite dykes contain the breccias of biotite porphyry.

1) Biotite Porphyry: Principal minerals are the phenocrysts of brown biotite of less than 4 mm in diameter, corroded quartz, and plagioclase, which has been replaced to an advanced stage by clay minerals (mostly kaoline). Groundmass consists of minute crystals of quartz, plagioclase, and clay minerals.

2) Rhyolite Dykes: Two of the dykes are found in the tunnel. One is 11 m inside from the portal, having a width of 1 m and being controlled by the fissures of the N 75°W and N-S systems. The other is at 35 m from the portal, striking in N 40°E and dipping steeply to the west, with a width of up to 10 meters. The rock is intensely altered; plagioclase is mostly replaced by clay minerals and quartz, and the groundmass is altered into the equigranular aggregates of quartz by silicification. Under the microscope, chlorite, epidote, and sericite

(0.03 - 0.2 mm in size) are recognized as the secondary minerals.

3) Breccia-Bearing Rhyolite Dykes: They are vertical dykes having 2 systems of strikes of N 70°E and N 30°W. Width varies from 10 cm to 3 meters. The breccias contained are the invaded biotite porphyry, in which plagioclase is at an advanced stage of alteration by argillization, yielding such clay minerals as kaoline, alunite, etc. The breccias are less than 10 cm in sizes and are contained in the dykes in irregular shapes. The matrix is so intensely silicified that it has changed into an aggregate of minute quartz grains, scarcely retaining its original texture.

8-4-3 Ore Minerals and Leached Zone

The ore minerals principally observed in the tunnel are pyrite, chalcopyrite, chalcocite, hematite, and green oxide copper minerals, and under the microscope, chalcopyrite is recognized, occasionally, as unsolved inclusion in chalcocite.

1) Pyrite occurs in network veinlets and in the disseminated form, presenting various modes such as corroded, crushed, and minute granular aggregation, etc. Sizes of pyrite vary from coarse (reaching to 5 mm) to minute (less than one micron), with an average of 0.3 mm.

2) Chalcocite fills up the cracks in pyrite, covers some pyrite as a thin film, or replaces pyrite in veinlets. It is also found disseminated in the host rocks or associated as stringers in quartz veinlets.

3) Green oxide copper minerals are seen to be produced when the ore is exposed for several days after it is mined, resulting in green coloration on the surface of ore. This shows the alteration of unstable chalcocite to the green oxide copper minerals. Such difference in ore due to the varying grades of alteration has to be kept in mind in the future development of the deposit.

4) Chalcopyrite is hardly recognized megascopically in the tunnel, but under the microscope, minute grains from 5 to 50 microns of unaltered, primary chalcopyrite of irregular shapes are recognized occasionally as being contained in the chalcocite.

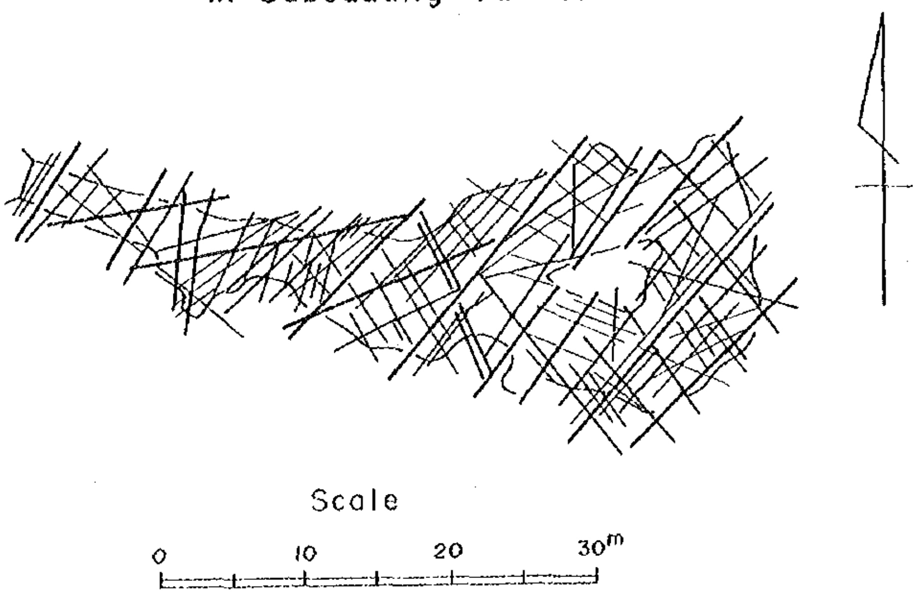
5) Hematite is the secondary product of pyrite, occurring disseminated or in veinlets in the leached zone. The boundary between the leached zone and the enriched ore zone more or less follows the ups and downs of surface topography, but it reaches deeper like a wedge due to the advanced oxidation along the fissures.

8-4-4 Distribution of Network Veinlets

Almost all the systems of fractures are found in the tunnel in the forms of fissures or network veinlets, which is well explained by Fig. I-8-1. Among them, those of N 40°E appear most frequently, those of N 70°E and N 45°W come next, and numerous network fractures containing pyrite are intermingled as well. This may cleary explain universal distribution of these fractures in the tunnel, too, which offered passages for migration of the dissolved copper, which was subsequently redeposited to form the secondary enrichment zone. At the same time, the possible lateral continuation of the deposit may have been justified.

It may safely be anticipated that the Kyisindaung Deposit has a similar tendency in its fracture distribution.

Fig. I-8-1 Fissure & Ore Veinlet Map in Sabedaung Tunnel



8-5 Underground Assays

8-5-1 Underground Sampling Method

In collecting underground assay data, the order of the procedures followed was sample collection, preparation of assay samples, and chemical analysis.

1) Underground Sampling: The samples were collected from the back and side walls of tunnel after the surfaces had been washed by splashing water on them to remove the adhering impurities. The side wall was sampled continuously at intervals of one meter along the entire sampled length. Each sample was collected by digging with chisel and hammer a channel of 1 m long, 5 cm wide, and 2 cm deep, at an average height of 1 m from the floor, from which the entire materials were caught by a canvas about the size of 2 m x 3 m. Quantity of each sample taken was about 2.5kg/m.

Samples from the back of the tunnel were taken crosswise at one meter intervals, and each sample was collected by digging a channel of 50 cm long, 5 cm wide, and 2 cm deep, from which all the materials were collected by a similar procedure to that used in the side wall sampling.

2) Preparation of Assay Samples and Chemical Analysis: The samples collected were smashed into grains of under 100 mesh by repeated crushing and quartering, to be assayed for total Cu by the Monywa Chimical Laboratory. Some of the samples were brought to Japan to be assayed by MESCO for total Cu, Au, Ag, S, Fe, As, and Zn.

The methods of chemical analysis in Burma and Japan are shown on the next page. Total samples collected from the tunnel number 275, of which 209 were from side walls and 66 from the back.

Refer to the attached diagram

Comparison of the Methods of Chemical Analysis by MMDC of Burma and by MESCO in Japan.

Comparison of the Methods of Chemical Analysis
by MMDC of Burma and by MESCO in Japan

MMDC

Sample 0.5 - 1.0 gr
- HCl (15-20 ml)
- $HClO_4$ (15 ml)

Evaporation for Consolidation followed by Natural Cooling
- H_2O (50 ml)

Heating and then Natural Cooling

Filtration
- Residue

Filtered Solution

Heating
- NH_4OH
- CH_2COOH

Natural Cooling
- NaF

Volume Arrangement
(adding H_2O up to total 50 ml)
- KI (2 gr)

Shaking

Titration $Na_2S_2O_3$

Cu %

MESCO

Sample 0.5 - 1.0 gr
- HCl (10 ml)
- HNO_3 (10 ml)

Heating for Solution

Evaporation for Consolidation
- HCl (5 ml)

Re-drying up
- $HCl + H_2O$ (1:1 10 ml)

Filtration, Cleaning
- Residue

Filtered Solution

Volume Arrangement
(adding H_2O up to total 100 ml)

Atomic Absorption

Cu %

* Assay for total Cu.

8-5-2 Results of Chemical Analysis

1) All the assay results were plotted on the underground assay map of scale 1/100 (PL.1-8-2).

2) Test samples for metallurgical treatment was collected from the deeper area, more than 12 m from the portal, where the effects of the leached zone were thought to be less. The grade of the ore zone was 0.81% Cu in arithmetic average of those samples from strictly within the ore zone as shown by the assay map of 1/100. It was showed 0.61% Cu when the leached zone was included in averaging.

This difference is due to the closeness of the level of tunnel to the boundary between the leached and enrichment zones, where the proportion of leached zone is more exposed. The boundary between the leached and enriched zones does not lie smoothly, but has many ups and downs. The grades will improve and be more stable at levels lower than the present tunnel or in the eastern extension, where the ore zone is at a greater depth from the surface.

3) Diamond drill hole JS-9 interesects the tunnel. The relation of assays of drill cores and tunnel at this intersection is shown on Fig.1-8-2.

4) The results of chemical analysis made in Japan on some of the underground samples have shown that Au, Ag, Zn and As are all traces through the analysis of 7 elements including Cu, Fe and S. This may suggest that the ore can be treated as a simple ore of which the major copper mineral is chalcocite.

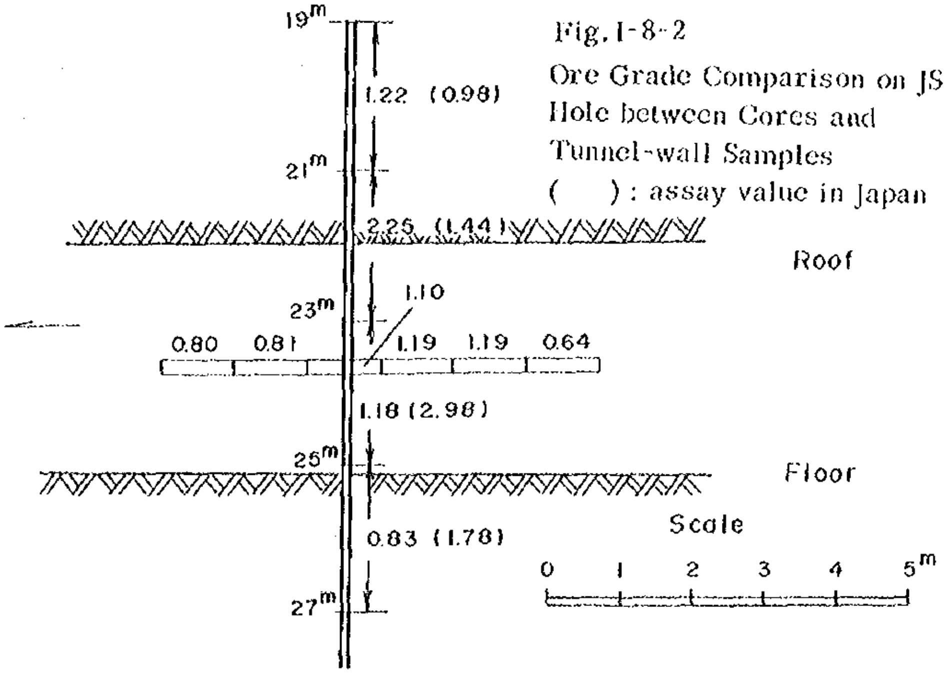


Fig. I-8-2
Ore Grade Comparison on JS-9 Hole between Cores and Tunnel-wall Samples
( ) : assay value in Japan

Table I-8-1 Chemical Analysis of Ore Sample in Sabedaung Tunnel

No.	Sample No.	Chemical Analysis							
		Burma	Japan						
		T-Cu	T-Cu	Fe	S	As	Zn	Au	Ag
		%	%	%	%	%	%	%	%
1	AR-29	1.19	1.41	3.83	3.87	0.00	0.00	tr	tr
2	ST-177	0.82	0.96	4.78	4.73	0.00	0.00	tr	tr
3	ST-184	2.41	2.14	6.00	6.21	0.00	0.00	tr	tr
4	AR-13	0.69	0.51						
5	AR-17	1.12	0.87						
6	AR-21	0.88	0.61						
7	AR-25	0.60	0.31						
8	AR-33	0.49	0.33						
9	AR-37	0.30	0.15						
10	AR-41	2.47	1.96						
11	AR-45	0.10	0.07						
12	ST-121	0.88	0.59						
13	ST-189	0.55	0.49						
14	ST-103	0.78	0.45						
15	ST-165	1.30	1.23						
16	ST-169	0.80	0.98						
17	BL - 5	1.02	0.82						
18	BL - 11	1.95	1.72						
19	BB - 9	0.97	0.86						
20	ST-108	0.49	0.31						
	Total	19.81	16.77						
	Average	0.99	0.84						

Table I-8-2 Microphotographs

No. 1

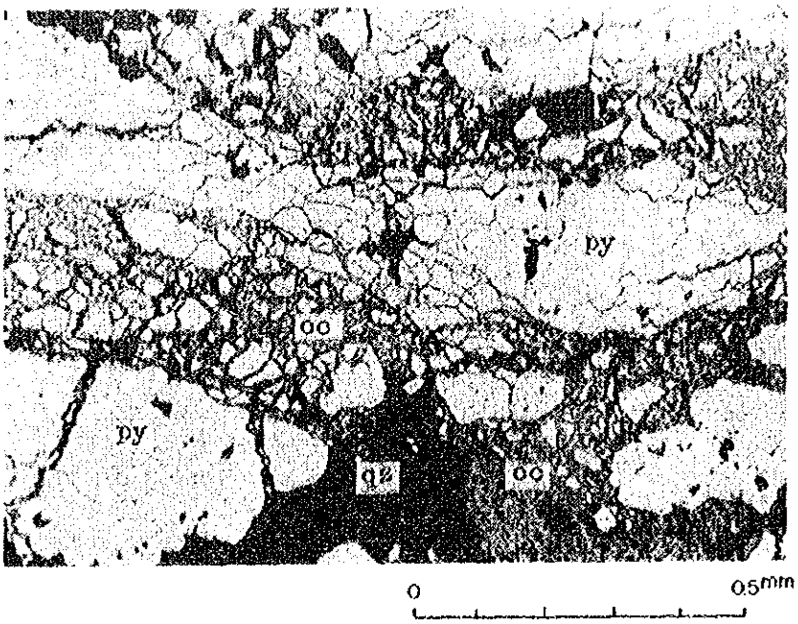
Sample No. 4

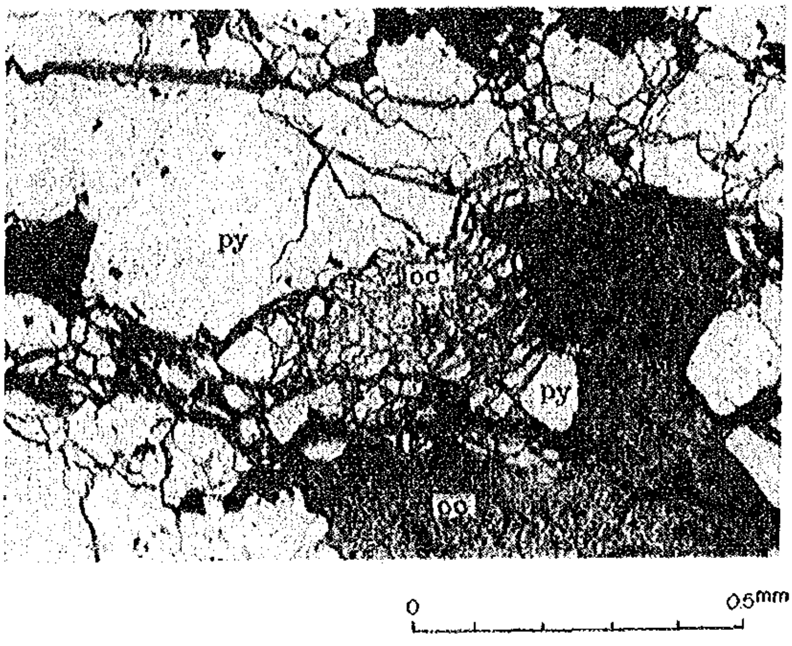
Location : Sabedaung Tunnel

py : Pyrite

cc : Chalcocite

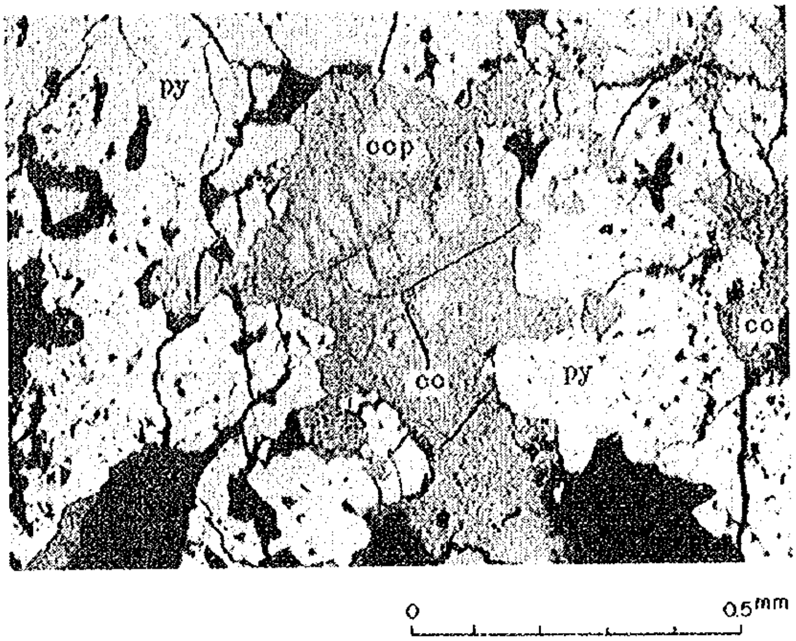
qz : Quartz





Sample No. 7

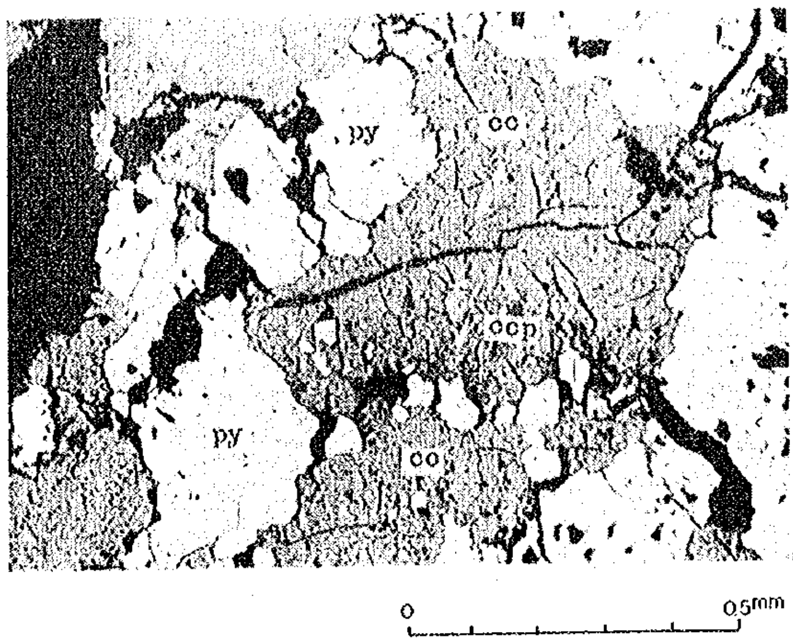
Location : Sabedaung Tunnel



py : Pyrite

cc : Chalcocite

ccp: Chalcopyrite

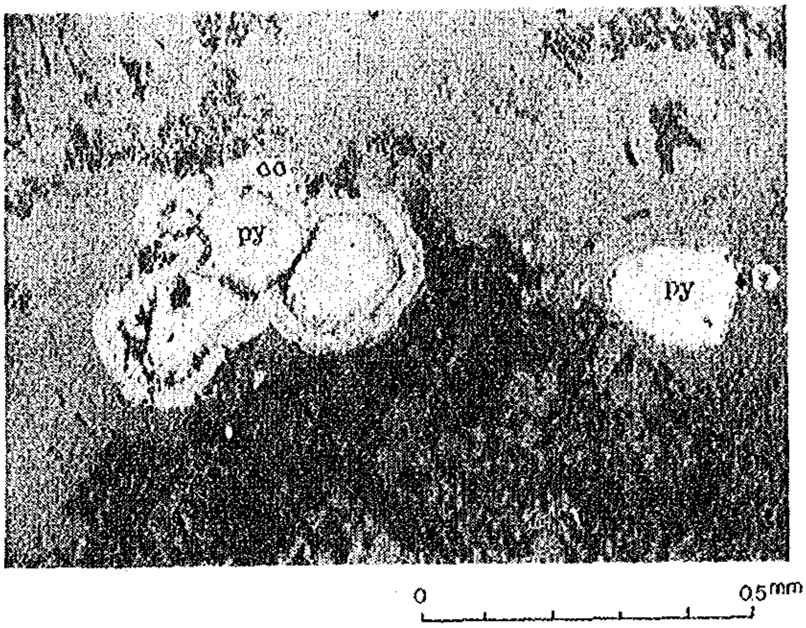


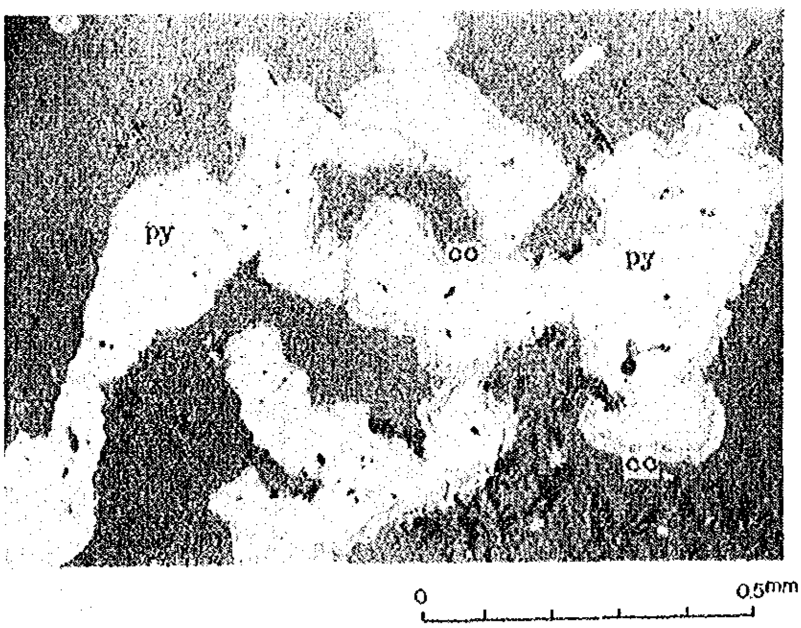
Sample No. 8

Location : Sabedaung Tunnel

py : Pyrite

cc : Chalcocite





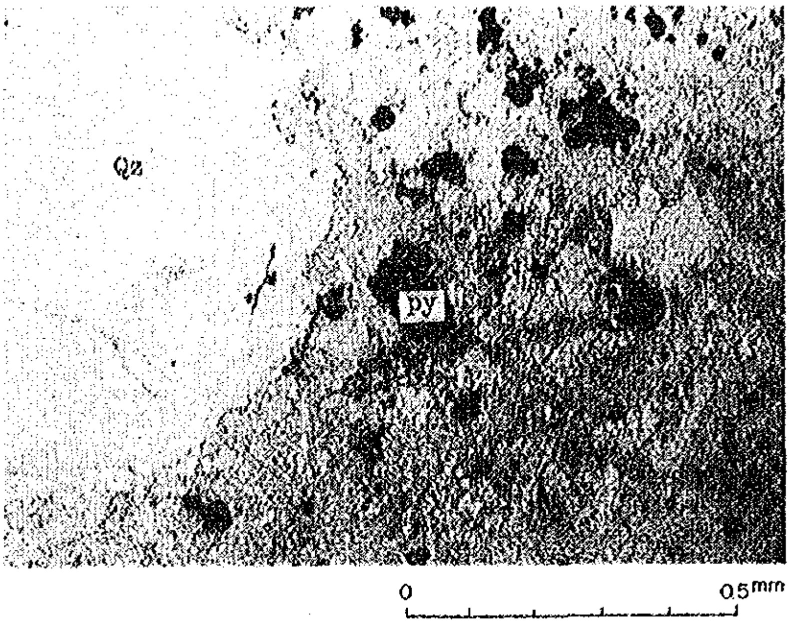
No. 4

Sample No. 7

Location : Sabedaung Tunnel

Rock Name :

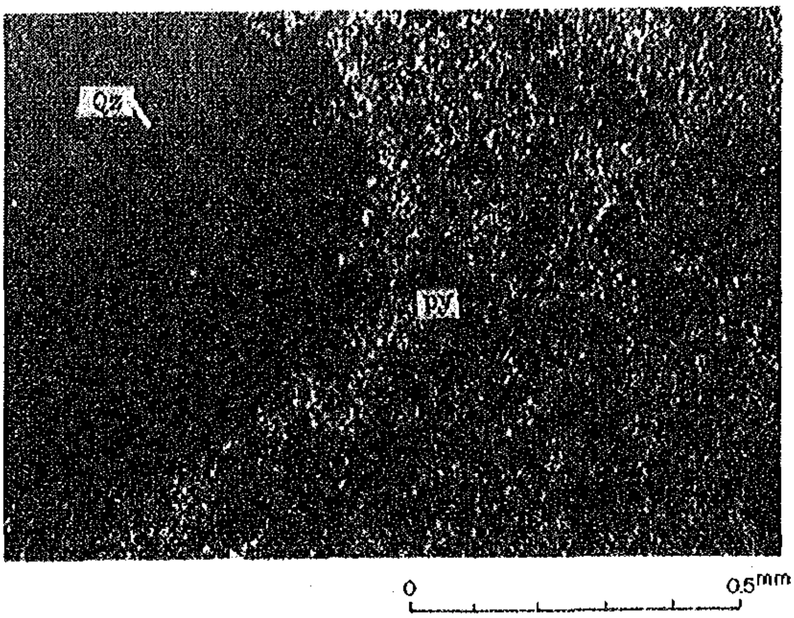
Altered Porphyry



Open Nicol

Qz : Quartz

py : Pyrite



Crossed Nicols

The rock shows porphyritic texture.

Groundmass is a very fine-grained equigranular rock, which consists mainly of quartz chlorite and sphene as secondary minerals, and pyrite as opaque mineral

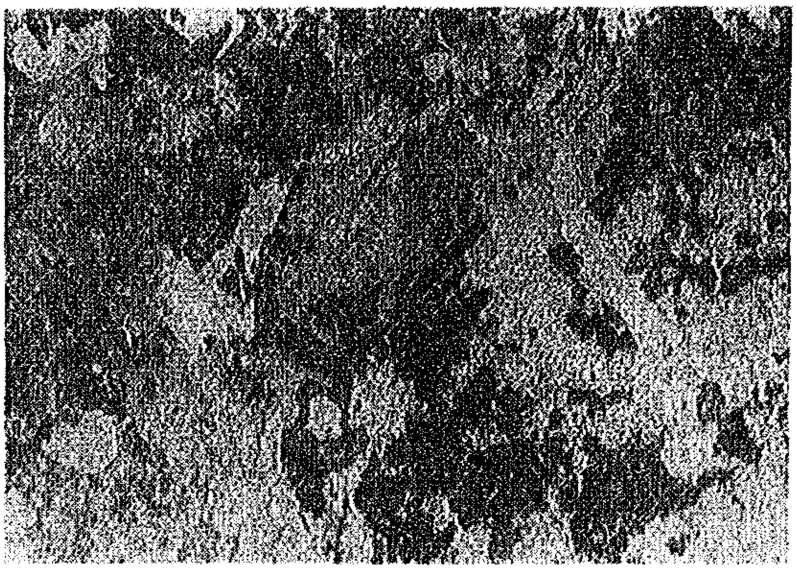
No. 5

Sample No. 9

Location : Sabedaung Tunnel

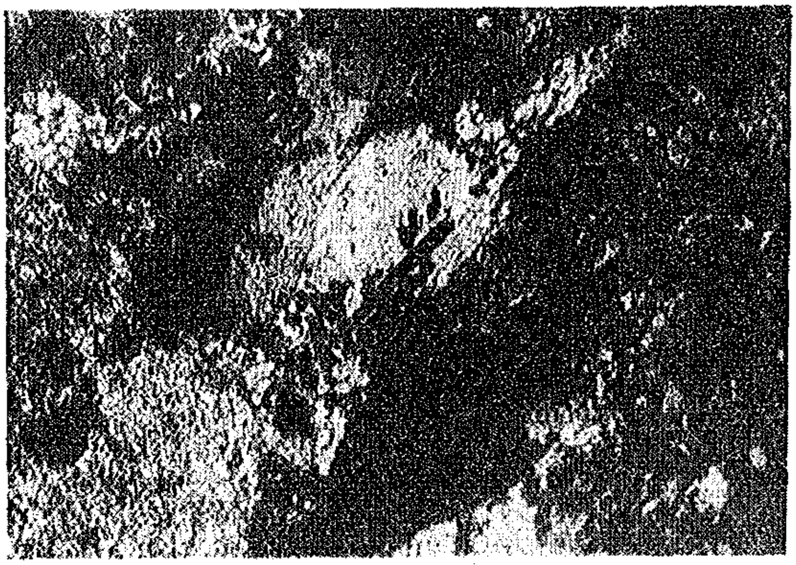
Rock Name :

Sltered Rhyolite



Open Nicol

0 0.5mm



Crossed Nicols

0 0.5mm

It consists of quartz chlorite, sericite, epidote, and sphene as secondary minerals.

Table I-8-3 List of Rock and Ore Sample at Sabedaung Tunnel

Sample No.	Location	Rock and Ore Name	Thin Section	Polished Section	Remarks
1	Sabedaung Tunnel	Pyrite-chalcocite ore in brecciated rock			Brecciated rock is porphyry
2	" "	Pyrite-chalcocite ore in altered rhyolite			Pyrite and chalcocite occurs in net-work veinlets and as disseminated
3	" "	ditto			ditto
4	" "	"		O	"
5	" "	"			"
6	" "	"			"
7	" "	Pyrite-chalcocite ore in altered porphyry	O	O	"
8	" "	ditto		O	"
9	" "	Altered rhyolite with pyrite	O		Weathered, silicified rhyolite with iron oxide
10	" "	Pyrite-chalcocite ore in brecciated rock			
11	" "	ditto			

Chapter 9 Conclusion

In Phase III, overburden removal was carried out to prepare for mining the ore in the vicinity of JS-9 hole in the southwestern part of the Sabedaung deposit. After the removal of 9, 100 m^3, the plan was changed into an Open Channel Cut(resulted in 1, 138 m^3 or 2, 844 t) to speed up the mining of the ore body. This was both in consideration of the total time period available for carrying out all the work and also of the difficulties in obtaining additional supplies of explosives and detonators.

To deal with the urgent necessity of feeding the requested ore for the mill test at the pilot plant, which was expected to start in September, the tunneling toward JS-9 hole from the end of the aforementioned channel was initiated in the middle of July as a part of Phase IV activities.

In driving the tunnel for a total length of 91.7 m, copper ore of 1,758 t was obtained, and additionally 2, 085 t of ore was mined by Wall/Roof cutting.

Out of those 3, 843 t in total, 3,788 t was hauled to the surface, and later the selected amount of 2,738 t was transferred to the pilot plant.

Throughout these operations, various factors on the development of Sabedaung deposit, i.e.: conditions of excavation, working efficiency of the excavation and transportation, consumption rate of explosives and detonators, requirement of bits and rods, and the distribution of ore grades underground were clarified. Furthermore, the ore deposit and the mineralized zone were studied geologically at the site.

As a result of those survey activities in Phase III and IV, the following items have been confirmed;

1) At Sabedaung deposit, the surface soil is about 2 m in thickness and the underlying weathered soft rock about 3 m in thickness.

Both of these layers can be efficiently removed by large earth-moving machinery such as bulldozers with rippers.

2) The nature of the ore deposit and its surrounding rock is less resistant to both drilling and blasting, and yet it is firm enough to support itself without timbering, not only in large scale tunneling but also in cutting walls up to about ten meters in width.

All these factor assure a highly efficient excavation requiring less explosives and smaller numbers of bit and rod.

3) Regarding mucking and haulage of the mined ore, there would be no problem even during the rainy season if only some reinforcement, together with drainage installations, and precise arrangement of the primary ore storage are provided at the mine site.

4) By means of underground geological survey, ore sampling and assay, the shape of the oxidized leached zone in the Sabedaung ore deposit, the horizontal continuity of the deposit, and the actual distribution of the ore grade have been directly ascertained.

Hereafter, in the future development of the mine, attention should be paid to the deeper extension of the low grade oxidized/leached zone along the fissures which prevails in the top portion of the deposit, rather close to surface.

5) The broken ore, has been observed to be very thinly coated with oxidized green minerals after being left even for a few days.

This kind of coating may have occurred due to the existence of unstable chalcocite.

It will be necessary to pay due consideration to the changes with time in the copper mineral which are dependent on this kind of weathering velocity.

In closing the report on mining, the following themes are to be recommended:

1. Continuation of Tunneling on a Deeper Level At Sabedaung

Deeper level tunneling is expected to be carried out to check the following: continuity of the ore deposit among the drilled holes, the extension of the oxidized leached zone, the ore grade distibution in the unknown portions of the deposit, clarification of the nature of ore, and also the completion of the mining programme. The captioned tunneling is expected to be carried out successively.

2. The main ore mineral, chalcocite, has been observed to dissolve in the underground water rather easily. A study should therefore be made on whether the recovery of copper ion from the lower grade ore can be carried out by means of in-place leaching, etc.

Part II

Construction of Pilot Plant

Part II Construction of Pilot Plant

Contents

Chapter 1 Introduction

1-1 Purpose

The purpose of the operations is to construct a pilot plant, together with its auxiliary installations, in order to carry out the concentrating test of the copper ore close at the mine site, in Monywa area, under the joint cooperation-survey program agreed between the goverments of Burma and Japan.

1-2 Outline

On the northeastern skirt of Kyisindaung, along the main mine road, the site for the plant had been selected.

The gentle slope was then partly cut to form several large areas of flat ground, chiefly for machinery foundations. Preliminary bench cut, pit digging, and ferro-concrete setting were carried out successively, according to the plans. Then, the concentrating plant and a diesel generator, which had been all brought in from Japan, were set and fixed on the prepared concrete foundations, and the necessary pipe and wire fitting work was carried out. Later, wooden buildings were constructed for the storage of the necessary machinery and equipment.

Auxiliary installations including the water supply and delivery system, a water reservoir, a fuel tank, and temporary housing for construction workers were constructed.

In addition, the banking for the tailing disposal pond, as well as construction work on the road were also carried out.

According to the specifications, the capacity of the pilot plant was designed to be 50 t/day on a 24 hour-operation basis.

The construction work started at the mill site on April, 8, 1975, and was completed on October, 28. Then, the test rounning of the pilot plant started accordingly.

1-3 Members

Akira Shimizu	MESCO	Civil engineer
Hiroshi Hashizumi	"	Metallurgist
Shoichi Uegaki	"	"
Kenichi Maeda	"	"
Yoshiharu Nakagawa	"	"
Toshio Tsujimoto	"	Mechanical engineer
Minoru Matsuhashi	"	"
Kazuhiko Momo	"	Electric engineer
U Than Maung	MMDC	Deputy director
U Taung Sein	"	Chief Engineer
U Htun Aung Zaw	"	Mechanical engineer
U Kyi	"	Officer in Charge
U Ko Ko	"	" "
U Saw Linn	"	Civil engineer
U Ba Ohn	"	"
U Myint Lwin	"	"

Chapter 2 Principal Machinery and Installations

2-1 Chief Machines and Equipment

Principally, the following machines and equipment are used in the pilot plant:

Main Machinery useed Pilot Plant

Name of Machinery	Type/ Specification	Q'ty	Manufacturer
Single Toggle Crusher	F 209, 18.5 KW	1	Otsuka Iron Worer
Single Toggle Crusher	F 158, 11.0 KW	1	-ditto-
Tube Mill	T-1515, 37.0 KW	1	-ditto-
Tube Mill	T-912, 18.5 KW	1	-ditto-
Conditioner tip	3' ø × 3', 2.2 KW	1	Futaba Manufactory
Conditioner	4' ø × 4', 3.7 KW	1	-ditto-
Spiral Classifier	400ø×5,000 L, 0.75 KW	1	-ditto-
Negaclon	HN-3	2	Rasa Traduing Co., Ltd.
Negaclon	HN-6	2	-ditto-
Flotator	FW# 15, 3.7 KW/2 Cells	14	Kawaguchi Marufactory
Flotator	FW# 12, 2.2 KW/2 Cells	8	-ditto-
Surge Tank	2.5mø	1	-ditto-
Thickener	5mø × 2.45m, 0.75 KW	1	-ditto-
Thickener	3mø × 2.40m, 0.75 KW	1	Futaba Marufactory
Centrifuge	T-26, 660ø, 2.2 KW	1	Nippon- Asahi Kiko
Vibrating Screen	2' ×4', 0.75 KW	1	Futaba Manufactory
Belt Feeder	350W × 4,000 L, 0.4 KW	1	-ditto-
Grizzly	1 m× 1.2 m× 20 mm	1	-ditto-
Belt Conveyor	KMR 350 mm ×7 m, 1 KW	6	Koyo Machirery
Turbin Pump	GME-CH, 3.7 KW	2	Hitachi, Ltd.
Warman Pump	3/2 EG-R/L 4V, 3.7 KW	2	Rasa Trading Co. Ltd.
Warman Pump	1 3/4 EG-R/L 3VR, 2.2KW	7	-ditto-
Smooth Autofeeder	CF-103, 0.2 KW	2	Taisei Kogyo
Miscellaneous Scale	1,000 Kg, 50 Kg, 5 Kg, 100gr	4	-
Diesel Generator	NPU-300, 250 KVA, 50Hz, 200V	1	Tokyo Shibaura Electric Corp. Ltd.
Attached Diesed Engine	380 PS	1	Catapillar Tractor Co.

2-2 Auxiliary Installations

Auxiliary installations of the pilot plant are mainly those shown in the following list:

Name of Installation	Specifications	Quantity
Water Tank	100 m^3 concrete made	1
Tube Well	350 m^3 in north of Plant	2
Delivery Pipeline	3 inch in diameter	450m
Fuel Tank for Generator	capacity 10 t, concrete & mortar	1
Storage for Crushed ore	concrete base, wooden housing 3.3 m x 3.3 m x 4.7 m^H	1
Main building	wooden structures 33.5 m x 8 m x 5 m 14 m x 7 m x 7 m	1 1
Concrete Retaining Wall	width 7 m, partly in bricks	3 stages
Attached Housing	wooden made, office, sitting room, etc.	3
Tailing Pond	35,000 m^2, encircled by 2.5~3 m^H earth banking	1
Access Road	over 6 m in width, with ballast embedded	1
Drainage ditch	inside of the plant site	1

Chapter 3 Review of Construction

The progress of the overall construction of the pilot plant is shown in simplified form, in the following table:

Items of Construction	Apr.	May	June	July	Aug.	Sep.	Oct.	Remarks
Procurement in Japan	— 15							Time loss for ship 1.5 months
Shipment		31 —	—	— 3				
Domestic transportation				19 —	— 16			
Ground arrangement	8 —	— 3						Time loss for custom clearance 0.5 month
Access road construction	11 — 22							
Tailing Pond construction		7 — 10						
Water Supply arrangement		5 — 18						
Machine Foundations		12 —	—	—	— 15			
Setting of Machines					4 — 28			
Housing Construction		7 — 11 22 — 31			1 — 31			
Pipe setting					—	—	— 20	
Electric work					—	—	— 27	
Miscellaneous Work					—	—	— 28	
No load Test running							28	

Chapter 4 Operations Carried Out

4-1 Selection of Plant Site

In selecting the site for the Pilot Plant, the following factors were carefully examined;

1) The location, which should preferably be as close as possible to the mine site, and yet out of the zone of influence of the blasting.

2) Convenient area for the handling of material and transportation, where the mine road passes close by.

3) Sloped topography which is suitable for the mill layout, with sufficient space for waste tailing disposal in the vicinity.

4) Convenience of water supply, with no danger of submergence even in flood seasons.

5) Stable ground with strength compatible to the buildings and equipment. Loose ground should be avoided.

Based on consideration of these criteria, the gentle slope along the mine road on the north eastern foot of Kyisindaung was selected for the pilot plant site. This location is close to the mine camp, and at the same time within a distance of about 0.7 km from the mine site.

There is a suitable space for the tailing pond along the lower side of the mine road, and the distance to the Yama stream is 0.8 km, with abundant underground water in between the stream and the mill site.

4-2 Ground Arrangement and Access Road

4-2-1 Ground Arrangement

On the selected slope along the hill side of the mine road, which originally had average inclination of 12 degree, a block of 5,500 m^2 was first marked. Then, the bulldozer D-80 started dozing to prepare the necessary space for the mill by cutting 5 benches as shown in the following list;

Name of Bench	Height above Mine Road	Area (with cut slope)
No.1 (Basal floor)	1.00 m	1,580 m^2
No.2 (No.2 Crusher level)	4.80 "	590 "
No.3 (No.1 " ")	5.40 "	385 "
No.4 (Upper Access road level)	8.90 "	1,580 "
No.5 (Water Reservoir level)	12.90 "	1,365 "

The ground surface was covered by weathered soil of 2 m in thickness, with underlying weathered soft rock.

The soil and rock were poor in permeability, and caused a muddy, unstable ground condition in rain.

Excess volume of earth, coming from the ground preparation, was mostly dozed westward into a small valley.

4-2-2 Access Road

An access road was constructed from the mine road on the MMDC camp side to the 4th bench at the pilot plant by a D-80 bulldozer.

It is 600 m in length, and 6 m in width.

Near the end of thr road at the plant site, the width is widened to 20 m to provide space for handling and storage of the crude ore. A drainage ditch was dug along the elongated wide space on the hillside to prepare for the coming rainy season.

The surface of the road was also covered with ballast, and completed using the motor grader and the road roller.

For No.2, 3 and 5 benches shown in the list, branches of access road were also constructed.

4-3 Foundation Work

The foundations for the concentrating machinery, reservoirs, retaining walls, buildings and other attached installation were made of ferro-concrete.

After digging the soft rock ground to a depth of 4 to 6 feet depending on the weight of the load, selected lumps of hard rock of 6 to 8 inches in size were laid on the bottom of these pits.

Then, primary concrete beds were made to form a stable base for the ferro-concrete foundations.

Following is a list of the work carried out and the respective materials used;

Total Volume of Concrete Work	516.1 m^3 (1:2:4)
Cement	170.0 t
Sand	258.0 m^3
Ballast	310.0 m^3
Iron bars	17,058.0 kg
Area of Forms	609.0 m^2
Mortar Accomplishment	408.5 m^2
No. of Bricks used	49,860 pcs

Cement, sand, ballast, bricks and materials for forms were supplied by MMDC, and the major amount of iron bars was offered from Japan.

Regarding the foundation work, the following factors caused difficulties and delay in the completion of the foundations: the delay in delivery of the materials, inferior quality of materials (especially the sand), lack of binding iron wire for iron bar structures, planks which were too thick for forms (especially for hollow type structures), and insufficient experience or skill of both technicians and workers.

Further delay was also caused by some partial re-concreting work which had to be carried out.

Local workers were employed by contractors and worked for 8 hours daily, principally in 2 shifts a day under the supervision of the civil engineers of MMDC. The mixing of concrete was done by means of a 0.3 m^3 concrete mixer.

All of the handling and transportation were undertaken by mannual labour at the plant site.

The average rate of mixing and pouring concrete was found to be 4.72 m^3/shift.

4-4 Transportation Shipments of Materials

Main Shipments of various concentrating machinery, together with auxiliary materials, were transported to the pilot plant site according to the following itenerary;

April 15, 1975	Ready for shipping at Yokohama Port (waiting for the ship)
May 31, "	Left Yokohama (by Five Star Liner, Kalewa)
July 3, "	Arrived in Rangoon Port

Custom Clearance and Import Procedures

July 19	Left Rangoon Port by 200 t Z-craft	August 2	Left Rangoon Station by freight train
July 31	Arrived at Monywa	August 13	Arrived at Monywa Station
August 4	All River Cargos arrived at Plant Site	August 16	All Rail Cargos arrived at Plant Site

The time required for some stages of the transportation eg. stand-by at Yokohama Port, Custom clearances & Import Procedures at Rangoon Port, Domestic transportation (especially, by rail), was much longer than expected. As a result the machinery, equipment and materials arrived at the plant site after a long delay.

Most of the delay and trouble occured as a result of the complicated and time-consuming administrative system.

This was due to the multiple corporations involved, especially in the acceptance, clearance and local transportation of shipments from abroad.

The haulage of cargo to the plant site, after arrival at Monywa, was carried out by using the following facilities;

200 t	Z-craft (As a ferry on the Chindwin river)	1
35 t	Truck with 12 t crane (Scrammel)	1
3.5 t	Caterpillared crane	1
6.5 t	Trucks	4

Owing to the preliminary repair and reinforcement of the mine road, the above transportation was successfully carried out even during seasonal showers.

During the transportation, some of the cargos were purposely damaged, and the contents were stolen part by part. Most of the vinyl sheets covering the attached motors etc. were completely pulled out and lost, thus causing the contents, including the motors, to be drenched by rain water.

Those lost materials were partly re-supplied later, mostly from Japan.

4-5 Setting and Arrangement

After opening the cargos at each location the piece of machinery was set and fixed tight on the foundations designed for it by means of 3.5 t crane and D-85 bulldozer. In setting up the equipment, welding work was undertaken mostly by Japanese technicians, because of the lack of the experienced welders at the plant site.

The acethylene and oxygen gas used was supplied from Rangoon.

Parallel to this work, the setting up of the 250 KVA (200V) generator with its diesel engine was also carried out, together with all the necessary wire and pipe fitting work.

4-6 Miscellaneous Auxiliary Work

4-6-1 Water Supply facilities

The daily requirement of water supply was calculated at about 300 t in full running of the pilot plant. (3 shift/day, 50 t/d)

To fulfill the requirement, the following operations were carried out successive-ly;

1) Drilling tube wells

On the flat farm 350 m north of the pilot mill, 2 numbers of tube wells were drilled by an auger type drill, and a sufficient amount of subsurface water, to be used at the pilot plant was obtained. The tube wells consist of one with a depth of 22 m and a diameter of 100 m/m, and another one of 19 m in depth and 150 m/m in diameter.

2) A water reservoir of 100 t capacity was constructed on the top level (No.5) of the plant.

3) A pipe line of 3 inches in diameter was set for a length of 450 m between the tube well and the aforementioned water reservoir.

A pump station was also constructed.

4-6-2 Waste Tailing Pond

The lower level waste land, including some farm land a total area of 35,000 m^2 along the mine road to the east of the pilot plant, was expropriated by MMDC, this was made into a Waste Tailing Pond by dozing the internal earth to build up a bank around its circumference.

The height of the banking was set at 2.5 m, and some of the surplus amount of earth was supplied from the ground arrangement work at the plant site.

The bottom of the tailing pond was mostly compact soil of weathered laterite of a non-permeable nature.

4-6-3 Other auxiliary installations

1) Fuel tank for the diesel generator

Capacity of 10 t, made of ferro-concrete with internal mortar accomplishment, with an attached lid.

The reservoir was set on the second bench, and installed with a drainage hole at the bottom to separate the contaminating water.

2) Ore bin for crushed ore

The bin, made of wood and with a roof, was built on a base of ferro-concrete. It was 3.3 m by 3.3 m at the bottom and had a height of 4.7 m.

3) Housing of the pilot plant

On the main ground area, a housing of 33.5 m x 8 m x 5 m (height at eaves) was built.

On No.2 and 3 benches, a second of 14.0 m x 7 m x 7 m (height st eaves) was also constructed. All the materials, such as columns, planks and roofing sheets, were locally supplied.
Later the floors of these buildings were partly to be paved, using either concrete or bricks.

On the top of the ore bin, the highest building at the plant, a lightning rod was installed.

4) Concrete retaining wall

3 retaining walls each with width of 7 m, were constructed to support the ore receiving yard, and the two crushers.

The walls were built between benches No.4 and No.3, No.3 and No.2, No.2 and No.1, successively. The walls were made of ferro-concrete at the central portions, and the both-sides were constructed of superficial bricks with internal concrete.

5) Auxiliary housing

Superviser's office, worker's room, small work shop and temporary store rooms were accomodated in 3 auxiliary buildings. These were wooden structures with woven sheets of bamboo bark according to the local specifications.

6) Others

Rooms for simple assay, measuring, metallurgical tests, minor repairs and for the watchman etc., were also prepared at the plant site, together with necessary instruments and equipment.

A telephone line was temporarily set for communications between the pilot plant and the base camp of the survey team.

Part Ⅲ

Concentrating Test

Part III Concentrating Test

Contents

List of Figures

List of Tables

Attached Plate

Chapter 1 Introduction

1-1 Purpose

The purpose of the concentrating test was to ascertain the Batch Test Results on the copper ore, which was described in Phase II Report for Manywa Copper Project (vol. No.2), utilizing the 50 t/d pilot plant built at the Northeastern foot of Kyisindaung hill in the Monywa Area. At the same time, the test was expected to supply the technical information necessary for the Feasibility Report on the development of the Monywa Copper Mine.

1-2 Outline

After the construction of the pilot plant, as well as the auxiliary installations which had been started in April, 1975, was completed, the test running and the adjustment of each mechanical, device and facility were carried out during the period from October, 28th to November, 10th.

Following this, on November 11, 1975, the operating test was started and was continued until February 14, 1976. During the operating test, the following points were studied and clarified·

1) Size distribution of Feed ore and Products
2) Grinding Work Index
3) Consumption of Bll and Liner of the Ball Mill
4) Operating Condition of Grinding and Flotation

In addition, 200kg of the produced copper concentrate was sent to Japan for tests on filtering and drying.

1-3 Members

Choki Okura	MESCO	Chief

Genshiro Sakai	MESCO	Metallurgist
Hiroshi Hashizumi	"	"
Shoichi Uegaki	"	"
Kenichi Maeda	"	"
Yoshiharu Nakagawa	"	"
U Than Maung	MMDC	Deputy Director
U Saw Ettrict San Hoo	"	Metallurgist
U Sin Kyin	"	"
U Kyaw Myint	"	"
U Nyunt Htay	"	"
U Myint Thien	"	"
U Than Nyunt	"	"
U Zaw Win	"	"
U Than Aung	"	Mechanical engineer
U Ko Ko	"	" "
U David	"	Electrical "

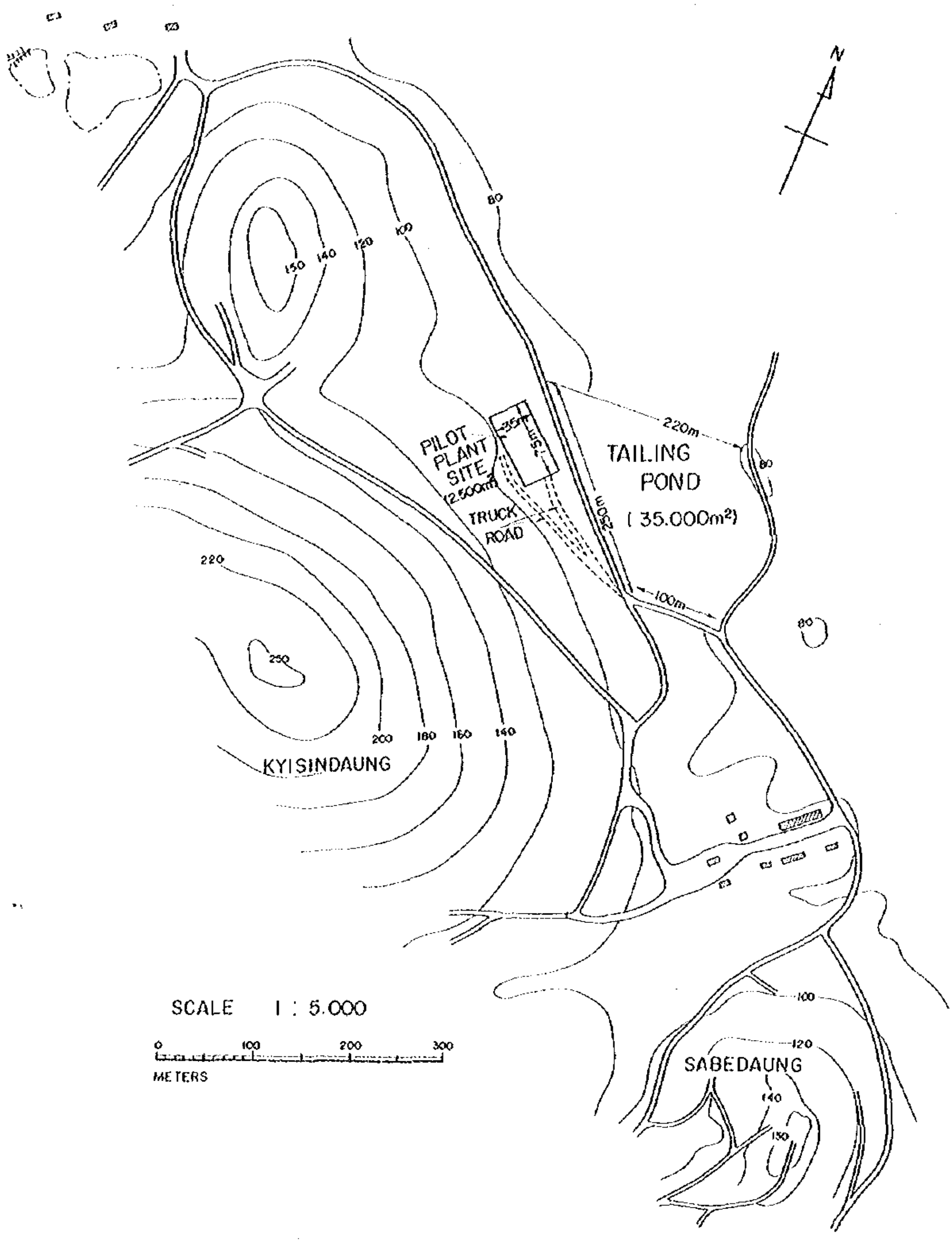
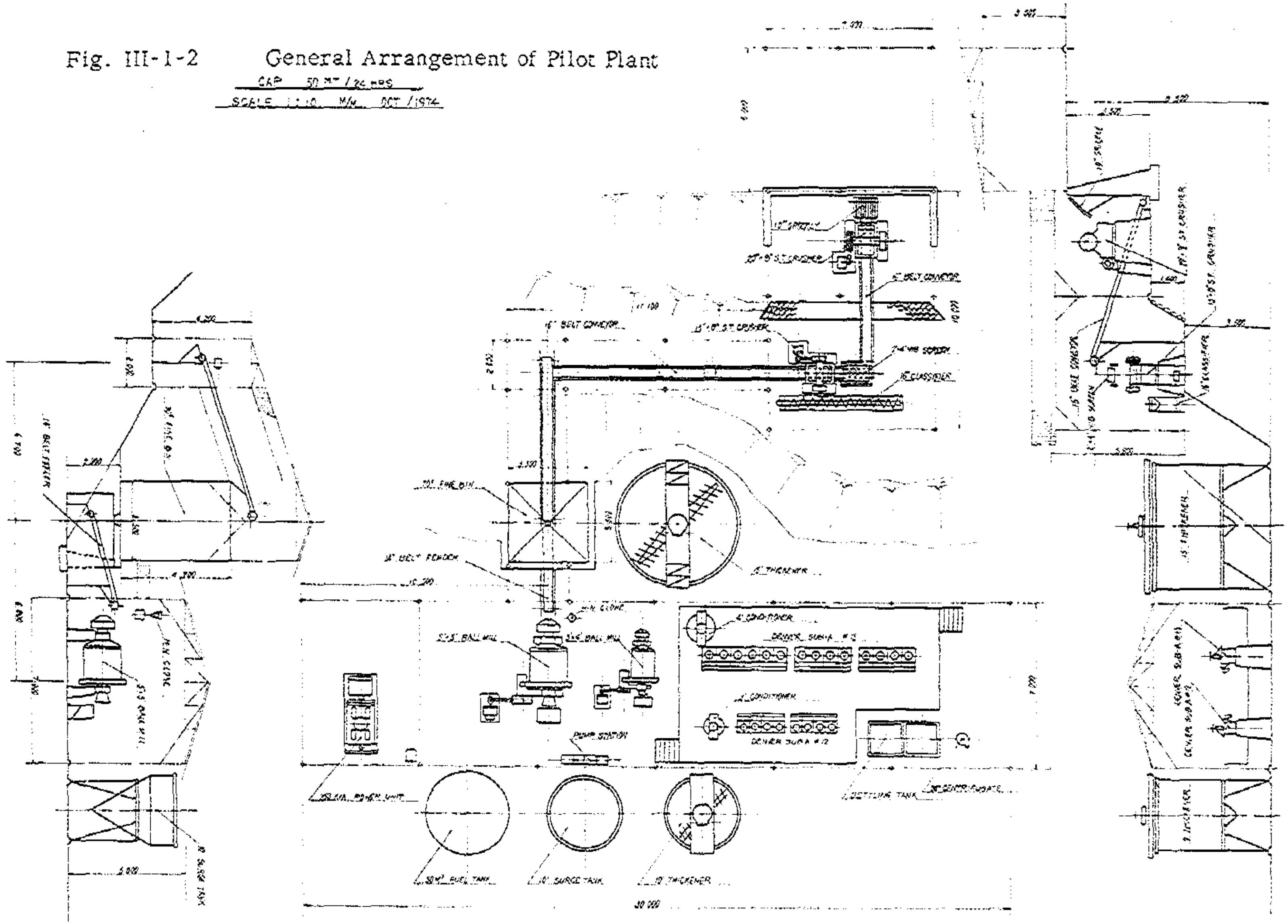


Fig. III-1-1 Location Map of Pilot Plant

Fig. III-1-2 General Arrangement of Pilot Plant

CAP 50 MT / 24 HRS

SCALE [illegible] M/M OCT /1974



Chapter 2 Outline of Pilot Plant

2-1 Flowsheet and Machinery

The flowsheet of the pilot plant and the specifications of the machinery used, both from the time of completion of the plant construction, are shown in Fig.III-2-1 and Table III-2-1.

2-2 Water Supply Installation

At the farm, 350m to the north of the pilot plant, 2 tube wells, one of 100 m/m dia. in 22m and the other of 150 m/m dia. in 19m were drilled by machine.

The water obtained from both of the wells was sent, by a turbine pump through a pipeline of 80 m/m dia., to the water reservoir. The reservoir, with a capacity of $100m^3$ was located on the hill slope of Kyisindaung at a level of 10m higher than the primary ore feed level, above the No. 1 crusher of the poiot plant.

2-3 Power Supply Facility

A diesel generator with 250 KVA capacity was installed in the main building of the plant.

All of the necessary power for the plant was supplied by the generator.

2-4 Waste Tailing Pond

To the north of the plant, on the lower flat ground beside the mine road, a waste tailing pond with an area of $35,000m^2$ was constructed, encircled by an earth banking of 2.5m in height.

Fig.Ⅲ-2-1 Flowsheet for 50tpd Pilot Plant

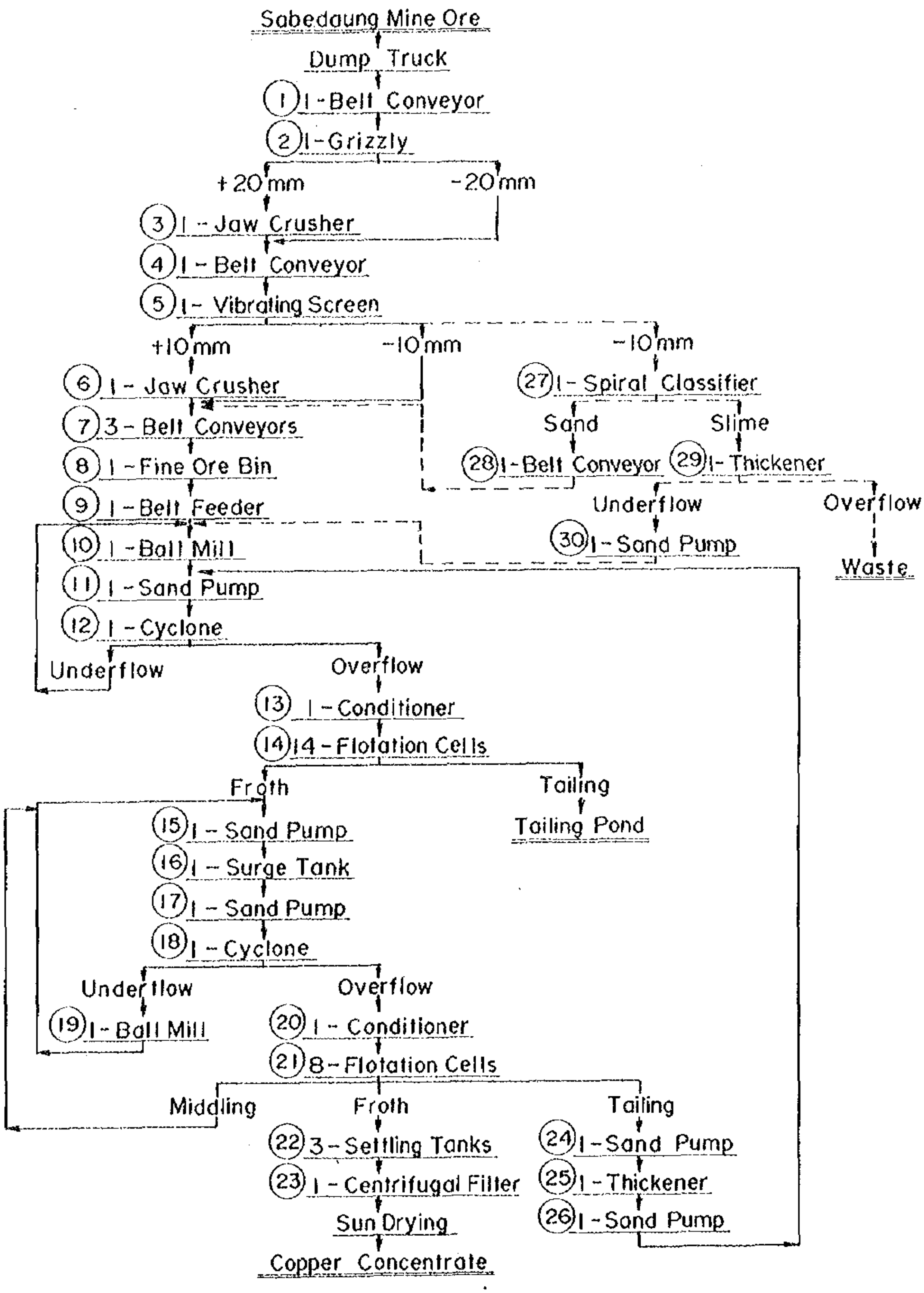


Table III-2-1 Specifications of Machineries

Item No.	Machinery and Equipment	Dimension, mm, or Capacity	RPM or Speed	Motor, Kw	Q'ty
1	Belt Conveyor	W 350 x L 7,000	38 M/Min	1	1
2	Grizzly	W 800 x L 1,200			1
3	Jaw Crusher	510 x 230	300	18.5	1
4	Belt Conveyor	W 350 x L 7,000	38 M/Min	1	1
5	Vibrating Screen	W 600 x L 1,200	900	0.75	1
6	Jaw Crusher	380 x 200	300	11	1
7	Belt Conveyor	W 350 x L 7,000	38 M/Min	1	3
8	Fine Ore Bin	70-ton			1
9	Belt Feeder	W 350 x L 4,000	10 M/Min	0.4	1
10	Ball Mill	Diam 1,500 x L 1,500	28	37	1
11	Sand Pump	Diam 75 x 50	1,560	3.7	1
12	Cyclone	Diam 150			1
13	Conditioner	Diam 1,200 x 1,200	500 M/Min	3.7	1
14	Flotation Cell	W 610 x L 610 x H 900	500 M/Min	3.7 per 2 cells	14
15	Sand Pump	Diam 25 x 20	1,990	2.2	1
16	Surge Tank	Diam 2,430 x H 4,115 with Mixer. Diam 450 x L 2,500	360	2.2	1
17	Sand Pump	Diam 25 x 20	1,990	2.2	1
18	Cyclone	Diam 75			1
19	Ball Mill	Diam 900 x L 1,200	38	18.5	1
20	Conditioner	Diam 900 x H 900	500 M/Min	2.2	1
21	Flotation Cells	W 560 x L 560 x H 900	500 M/Min	2.2 per 2 cells	8
22	Settling Tank	0.5 M^3			3
23	Centrifugal Filter	Diam 660 x H 330	1,000	2.2	1
24	Sand Pump	Diam 25 x 20	1,990	2.2	1
25	Thickener	Diam 3,000 x H 2,400	0.57	0.75	1
26	Sand Pump	Daim 25 x 20	1,990	2.2	1

- Continued -

Item No.	Machinery and Equipment	Dimension, mm, or Capacity	RPM or Speed	Motor, Kw	Q'ty
27	Spiral Classifier	Diam 400 x L 5,000	12	0. 75	1
28	Belt Conveyor	W 350 x L 5,000	38 M/Min	1	1
29	Thickener	Diam 5,000 x H 2,450	0. 26	0. 75	1
30	Sand Pump	Diam 25 x 20	1,990	2. 2	1
	Platform Scale	1,000 kg			1
	Platform Scale	50 kg			1
	Steel Yard Scale	5 kg			1
	Pulp Density Scale	1,000 ml			1
	PH Meter	Model D-5			1
	Dry Reagent Feeder	1,800 ml/Hr		0. 2	2
	Milky Lime Feeder with Sand Pump	Tank Vol. 0. 8 M^3 Diam 25 x 20	1,990	2. 2	1
	Submersible Pump	Diam 40	3,000	0. 25	1
	Handcart	0. 11 M^3			1
	Stand-by:				
	Belt Conveyor	W 350 x L 7,000	38 M/Min	1	1
	Belt Conveyor	W 350 x L 5,000	38 M/Min	1	1
	Sand Pump	Diam 75 x 50	1,560	3. 7	1
	Sand Pump	Diam 25 x 20	1,990	2. 2	1
	Cyclone	Diam 150			1
	Cyclone	Diam 75			1
	Turbine Pump	Diam 40 x 8 stages	1,500	3. 7	2
	Water Tank	100 M^3			1
	Flotation Test Machine	MS-type, 100 g and 200 g		0. 2	1
	Flotation Test Machine	FW-type, 500 g		0. 2	1
	Testing Sieve	26. 7mm, 18. 8mm ----- 44 and 37			set
	Fuel Tank	4 M^3			2
	Diesel Generator	250 kvA	1,500		1

Chapter 3 Review of Test

3-1 Sequence of Operations

The sequence of the operations carried out in the construction and operating of the 50 t/d pilot plant at Kyisindaung, is shown as follows:

April 8, 1975	Start of Plant Construction (Ground Arrangement)
May 3, "	Completion of Ground Arrangement
May 12 to Aug. 15	Machinery Fountation Works
July 19 to Aug. 16	Cargo Transportation (Rangoon to Plant site)
Aug. 5 to Sep. 20	Setting of Major pieces of Machinery
Aug. 1 to Oct. 27	Housing Construction
Sep. 2 to Oct. 27	Setting of Water Delivery Pipes
Sep. 8 to Oct. 18	Setting of Miscellaneous Installation
Sep. 11 to Oct. 27	Electric Engineering Works
Sep. 28	Completion of Plant Construction
Oct. 28 to Nov. 10	Running Test and Adjustment
Nov. 11	Start of Operating Mill Test
Nov. 11 to Dec. 20	Operating Test (in 1 shift, 8 hrs/d)
Dec. 22 to Dec. 24	- ditto - (in 2 shifts, 16 hrs/d)
Dec. 26 to Jan. 20	- ditto - (in 3 shifts, 24 hrs/d)
Jan. 22 to Jan. 25	Rearrangement of Flotation Circuit and Maintenance Work
Jan. 26 to Jan. 31	Operating Test (in 2 shifts, 16 hrs/d)
Feb. 2 to Feb. 7	- ditto - (in 3 shifts, 24 hrs/d)
Feb. 9 to Feb. 11	Measurement of Ball/Liner Consumption
Feb. 13 to Feb. 14	Operating Test (in 2 shifts, 16 hrs/d)

Feb. 14	Completion of Operating Test

3-2 Personnel Arrangement

The personnel arrangement for the operation and maintenance of the pilot plant is shown as follows;

(Operation)	No. of man	Shift	man-shift/d
Chief Metallurgist	1	1	1
Metallurgist	1	3	3
Assistant Metallurgist	1	3	3
Chief Operator	1	3	3
Operator for Crushing	3	3	9
Operator for Grinding	1	3	3
Operator for Flotation	1	3	3
Sampler	1	3	3
Helper for Dewatering			4
Total			32
(Maintenance)			
Mechanical Engineer	1	1	1
Assistant Mechanical Engineer	1	1	1
Chief Operator for Generator & Pump	1	1	1
Operator for Diesel Generator	1	3	3
Operator for Water Pump	1	3	3
Mechanic for Repair (Welding)	4	1	4
Electrician	1	3	3
Watchman	1	3	3
Total			19
Grand total			51

3-3 Considerations for Safety

Sufficient attention was paid to the operating safety of the pilot plant, as well as to the necessary devices for safety in its construction.

The chief items concerned were as follows:

1) Safety patrol for the facilities and for the surroundings at the pilot plant was carried out more than once during each operating shift. In the patrol, the following points were checked, and notes were made to be handed to the responsible person of the next shift;

(1) Whether or not there is any change in the machinery, instruments, and installations concerning safety.

(2) Daily details of every operation, maintenance, repair and stoppage at the plant

(3) Actions taken to ensure Safety and their Results

(4) Other comments for next shift, if any.

2) To prevent misfire or spontaneous fires, smoking and the use of fire were forbidden in and around the pilot plant, except in the smoking corner and the areas where necessary welding work was carried out.

In areas with a high possibility of spontaneous fires, a daily check on the temperature and humidity was carried out and the results were recorded with careful attention being paid to changes.

3) Concerning the storage of containers used for gas welding, special measures were taken to prevent them from rolling or falling, and to protect them from sudden shocks.

Attention was also paid that the temperature of the containers was kept below 35 degree Centigrade. Safety spectacles and gloves were delivered and their use was requested during gas and electric arc welding work.

4) The storage area for powerful drugs and poisonous chemicals, both of which were to be used in either the flotation or the chemical laboratory, was locked carefully to prevent accidints and to insure against theft.

5) All the iron platforms and housings of the installed electric facilities at the plant were grounded for safety. Circles or Fences were built around the high voltage transformers with warning signboards, "Danger".

Also, a lightning conductor was set on the top of the highest building.

6) Necessary coverings, fences and other safety devices were installed for those areas of the pilot plant and other facilities where the machinery could be considered as dangerous.

7) Handrails were put on necessary stairways and scaffolding in the pilot plant.

Checkered iron plate was especially used for both the stairway steps and the floors of the passages, and platforms of the plant, to reduce the danger of slippage.

Chapter 4 Result of Tests

The number of days of operation of the pilot plant and the amount of ore treated, etc. during the period between November, 11, 1975 and February, 14, 1976, are shown as follows;

Period	Operating Day	Operating Hour		Ore	
		Crushing	Concen-tration	Treated, Tons	Assay % Cu
		H M	H M		
Nov., 1975	10	50:00	71:30	150. 1	0. 77
Dec.	29	280:30	347:30	739. 7	0. 71
Jan., 1976	21	321:55	422:40	890. 8	0. 70
Feb.	8	740:25	107:25	357. 6	0. 62
Total	68	792:50	1,009:05	2, 138. 2	0. 70

The result of the tests is described separately for each test item in the following articles:

4-1 Physical Nature of Crushed Ore

4-1-1 Size Distribution

The result of the study on the size distribution of feed ore, primary crushed ore, and secondary crushed ore is shown in Table III-4-1.

Table III-4-1 Sizing Test on Crushing Products

Size			Distribution, %		
mm	inch	mesh	Crude Ore	Primary Crushed (Screen Feed)	Secondary Crushed (Mill Bin Feed)
+152	+6		41. 8		
+12. 7	+1/2		49. 1		
+0. 21		+65	7. 6		
+0. 074		+200	0. 3		
-0. 074		-200	1. 2		
+19. 1	+3/4			38. 3	3. 2
+12. 7	+1/2			18. 0	24. 2
+9. 52	+3/8			8. 3	13. 9
+6. 73		+3		7. 4	12. 3
+4. 76		+4		5. 7	10. 9
-4. 76		-4		22. 3	35. 5
Total			100. 0	100. 0	100. 0
Nos. of Measurement			7	1	2

4-1-2 Apparent Specific Gravity

Feed ore of under 152 mm in size had a weight of 214.3 kg for a volume of 0.115 m^3, when it was tightly packed on a hand cart, with water content of 0.8% of the ore weight. Thus, the apparent specific gravity was calculated as 1.85, namely;

$$214.3 \text{ kg} \times 0.992 \div 115 \text{ l} = 1.85$$

Similarly, the apparent specific gravity of the secondary crushed ore, in 3 measurements was found, to be 1.683, 1.636, and 1.706. The average of the three is 1.68, and therefore we summarize as follows;

-152 m/m	size Ore	1.85 t/m^3
-19.1 m/m	size Ore	1.68 "

4-2 Necessity of Washing

As already shown in Table III-4-1, the water content of the crude ore of under 200 mesh was found to be as low as 1.2% on average. For this reason, it was not felt necessary to wash the ore to get rid of the primary slime in the feed. Thus, washing tests were not applied to the ore at all.

4-3 Size Distributions of Ground Ore

4-3-1 Primary Grinding

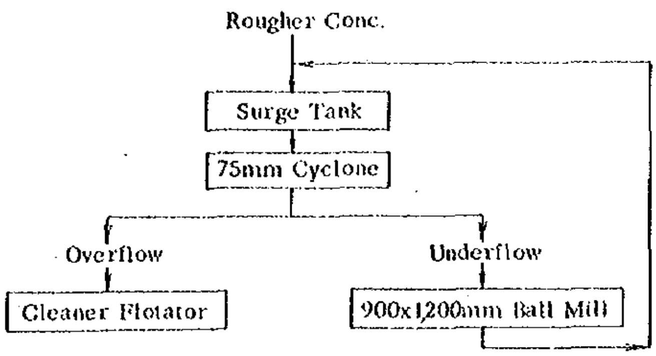
The size distribution of each product under normal operation was found to be as shown in the following Table III-4-2:

Table III-4-2 Sizing Test on Primary Grinding Products

Size		Distribution, %			
mm	Inch	1,500 x 1,500mm Ball Mill		150mm Cyclone	
		Feed	Discharge	Overflow	Underflow
+19. 1		2. 9			
+12. 7		20. 5			
+9. 52		15. 7			
+6. 73	+3	14. 4			
+4. 76	+4	10. 0			
-4. 76	-4	36. 5			
+1. 19	+14		1. 2		1. 3
+0. 59	+28		3. 6		3. 6
+0. 42	+35		4. 0		4. 3
+0. 297	+48		6. 7		6. 4
+0. 210	+65		9. 7	1. 1	13. 2
+0. 149	+100		18. 5	3. 6	20. 4
+0. 105	+150		13. 6	9. 0	14. 8
+0. 074	+200		9. 9	7. 8	11. 5
-0. 074	-200		32. 8	78. 5	24. 5
Total		100. 0	100. 0	100. 0	100. 0
Nos. of Measurements		7	7	16	7

4-3-2 Regrinding (Before January, 10, 1975)

During the period from the start of the operating test until January, 10, 1976, a ball mill of 900 x 1200 mm in size, together with 75 m/m cyclon, was used in the closed circuit of the following flow sheet:



The result of the size distribution measurement of the reground products in the said circuit is shown in the following Table III-4-3:

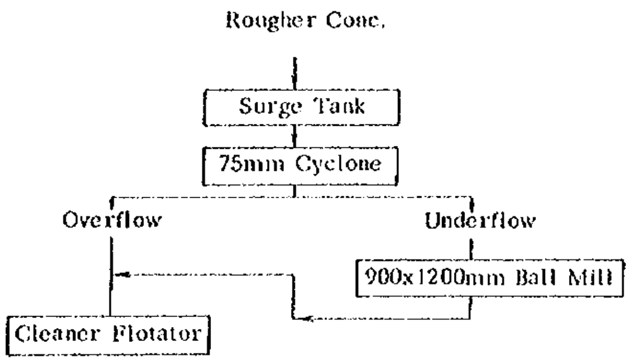
Table III-4-3 Sizing Test on Regrinding Products (Before 10th, Jan.)

Size		Distribution, %		
micron	mesh	Cyclone Feed	Cyclone Overflow	Cyclone U'flow (Mill Feed)
+149	+100	2.3	-	2.0
+105	+150	2.5	-	6.8
+74	+200	7.3	1.0	6.5
+53	+270	3.4	1.2	5.1
+44	+325	9.3	2.0	17.6
-44	-325	75.2	95.8	62.0
Total		100.0	100.0	100.0
Nos. of Measurements		3	5	3

4-3-3 Regrinding (After January, 13, 1976)

After stopping the ore grinding under the closed circuit on Jan., 10, a new circuit was adopted to improve the grinding.

In the new circuit, the underflow of the 75 mm cyclon is reground a second time then fed to the cleaner together with the overflow of the cyclon, as shown in the following illustration.



The size distribution of the products in the afore-mentioned circuit is as shown in Table III-4-4.

Table III-4-4 Sizing Test on Regrinding Products (After 13th Jan.)

Size		Distribution, %	
micron	mesh	Cyclone Overflow plus Mill Discharge	Cyclone Underflow (Ball Mill Feed)
+149	+100	-	2.3
+105	+150	-	2.9
+74	+200	1.6	4.4
+53	+270	2.5	6.5
+44	+325	2.9	10.0
-44	-325	93.0	73.9
Total		100.0	100.0
Nos. of Measurements		2	3

4-4 Grinding Work Index*

The result of the measurement of the Operating Work Index of a 1500 x 1500 mm Ball Mill is shown in Table III-4-5 as follows;

Table III-4-5 Operating Work Index on 1500 x 1500mm Ball Mill

Period	Ore Treated, ton	Energy Input		80% Passing Size		Operating Work Index, Kw-hr/short ton
		Kw-hr	Kw-hr short ton	Feed, micron	Product, micron	
1st Dec. to 30th Dec.	564.0	9,261.1	14.90	13,400	91	14.33
5th Jan. to 7th Feb.	1,077.4	19,028.4	16.02	13,750	81	16.44

* Index which is based on "The Third Theory of Comminution" by F.C. Bond. This index primarily expresses the resistivity of the smashed particle, which can be defined in the following formula;

$$W = Wi\left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right)$$

where

W : The motor input required for the grind in the kilowatt hours per short ton.

Wi: The work index of the material. It is proportional to the kilowatt hours per short ton required to grind a theoretically infinite size particle to 80 percent passing 100 microns.

P : The 80 percent passing point of product screen analysis in microns.

F : The 80 percent passing point of feed screen analysis in microns.

The standard work index, mentioned by, F.C.Bond, is originally derived from the value which has been obtained through the feed ore under the size of $4{,}000\sqrt{\frac{13}{W}}$ microns by means of a ball mill of 8 feet in diameter.

Accordingly, the operating work index mentioned above had to be corrected on both the mill diameter and the oversize feed.

Correspondingly, the correction was made, and the result is shown in Table III-4-6. The table shows that the work index for the Monywa (Sabedaung) copper ore comes to between 11.76 and 13.14 Kilowatt hours per short ton.

Table III-4-6 Grinding Work Index

Period	Operating Work Index (1)	Factor		Work Index kw-hr/short ton (1) x (2) x (3)
		for Mill Size (2)	for Over-size Feed (3)	
1st to 30th, Dec.	14.33	0.901	0.911	11.76
5th Jan. to 7th Feb.	16.44	0.901	0.887	13.14
				Average 12.45

4-5 Consumption of Balls and Liners

The consumption of balls and liners in each ball mill is shown in Tables III-4-7 andIII-4-8,respectively. On average, the consumption was 764 g/t of balls and 215g/t of liners for the 1500 x 1500 mm ball mill. On the other hand, an average of 212 g/t of balls and 67 g/t of liners, respectively were consumed in the 900 x 1,200 mm smaller ball mill.

4-6 Relation of Feed Rate to Performance

The result of the study on the relation between the feed amount for the ball mill and the grinding performance, under the closed circuit operation by means of 1500 x 1500 mm ball mill and 150 mm cyclon, is shown in Table III-4-9.

In short, the grain size of the cyclone overflow (rougher feed) becomes finest when the feed rate for the ball mill is set at between 2.1 and 2.2 t/hr.

Table III-4-7 Wear of Grinding Ball

Period : 11th Nov., 1975 to 7th Deb., 1976
Ore treated : 1,855.4 ton
Material of Ball : Cast alloy iron

Diameter of Ball, mm	Weight, kg			Consumption, g/t
	Charge	Remain	Wear	
1,500x1,500mm Ball Mill				
75	1,857.6	1,436.5	421.1	227
50	2;542.4	2,345.5	996.9	537
40	800.0			
Total	5,200.0	3,782.0	1,418.0	764
900x1,200mm Ball Mill				
40	351.2	249.0	102.2	55
30	411.6	288.0	123.6	67
25	400.0	232.0	168.0	90
Total	1,162.8	769.0	393.8	212

Table III-4-8 Wear of Ball Mill Liner

Period : 11th Nov., 1975 to 7th Feb., 1976
Ore treated : 1,855.4 ton
Material of Liner : Manganese-steel

Type of Liner	No. of Set	Unit wt., kg		Total wt., kg		Consumption	
		New	Used	New	Used	Kg	g/t
1,500x1,500mm Ball Mill							
End A	4	44.0	39.5	176.0	158.0	18.0	10
" A'	4	44.0	39.5	176 0	158.0	18.0	10
" B	8	43.0	40.2	344.0	321.6	22.4	12
" B'	8	43.0	40.7	344.0	325.6	18.4	10
Shell C	12	71.0	63.0	852.0	756.0	96.0	52
" D	12	71.0	62.2	852.0	746.4	105.6	57
" E	12	70.0	60.0	840.0	720.0	120.0	64
Total	60	-	-	3,584.0	3,185.6	398.4	215
900x1,200mm Ball Mill							
End A	4	13.0	12.0	52.0	48.0	4.0	2
" A'	4	13.0	12.1	52.0	48.4	3.6	2
" B	4	22.0	20.8	88.0	83.2	4.8	3
" B'	4	22.0	20.9	88.0	83.6	4.4	2
Shell C	8	36.0	32.0	288.0	256.0	32.0	17
" D	4	36.0	31.6	144.0	126.4	17.6	10
" E	8	55.0	47.7	440.0	381.6	58.4	31
Total	36	-	-	1,152.0	1,027.2	124.8	67

Table III-4-9 Effect of Feed Rate on Performance of 1,500x1,500mm Ball Mill

Date		5th Dec.		6th Dec.		4th Dec.		3rd Dec.		8th Dec.	
Feed Rate t/Hr	8 a.m.	1,700		1,898		2,203		2,066		2,544	
	9 a.m.	1,574		1,748		2,120		2,538		2,874	
	10 a.m.	1,696		1,619		1,622		2,382		2,379	
	11 a.m.	1,532		2,166		1,697		2,330		2,439	
	12 a.m.	1,625		2,301		2,210		2,108		2,881	
	1 p.m.	1,565		1,682		2,915		2,520		2,809	
	2 p.m.	1,811		1,703		2,252		2,412		2,586	
	Average	1,643		1,874		2,146		2,337		2,645	
Screen Analyses		Weight, %									
Mesh		MD	CO	MD	CO	MD	CO	MD	CO	MD	CO
+65		18.3	4.5	22.8	4.1	26.4	1.9	30.7	2.3	31.9	3.3
+100		13.1	7.6	17.8	6.0	17.6	4.3	16.9	7.8	17.7	6.2
+150		8.6	8.8	11.3	8.4	13.1	7.4	10.5	7.8	11.3	8.3
+200		9.5	11.6	12.1	11.5	7.4	10.4	12.9	11.1	13.1	10.2
-200		50.5	67.5	36.0	70.0	35.5	76.0	29.0	71.0	26.0	72.0
Pulp Density, % solid		49.3	30.7	50.7	28.5	59.0	26.8	60.5	26.5	59.4	24.5
Circulating Load, %		118		670		785		848		1263	
Power Consumption, kwh/t		20.52		18.39		15.81		14.77		13.03	
-200 mesh Tons Produced per kwh		0.033		0.038		0.048		0.048		0.055	

MD = Mill Discharge CO = Cyclone Overflow

4-7 Flotation Circuit and Grade of Products

4-7-1 Ordinary Flotation Circuit

Two types of flotation circuit were adopted.

They are the types which were introduced in Fig. III-4-1 and Fig. III-4-2, respectively.

The latter differs from the former in the way of drawing the tailing in the later half of the cleaning process. In the latter circuit, the tailing is drawn out from each cell.

4-7-2 Scavenger-Cleaner type Flotation Circuit

In the ordinary circuits of Fig. II-4-1 and Fig. II-4-2, the tailing of the cleaner is fed back to the primary ball mill, and this often adversely affected the flotation performance.

In this case, the bad condition of the cleaner circuit directly affected the rougher.

To solve this problem, instead of returning the middling to the primary ball mill, an exclusive circuit for retreatment of the middling in flotation was contrived to stabilize the operation. Using this new circuit, the flotation test was investigated.

This circuit is shown in Fig. III-4-3, and has 2 cleaning sub-circuits combined with a ball mill for the necessary regrinding.

Of the 2 sub-circuits, one is used as a rougher-cleaner only to treat the froth from the rougher. The other is used as a scavenger-cleaner, which deals separately with both the primary scavenger froth from the cleaner and the tailing from the rougher cleaner.

Fig. III-4-1 Schematic Flotation Circuit (at the Beginning)

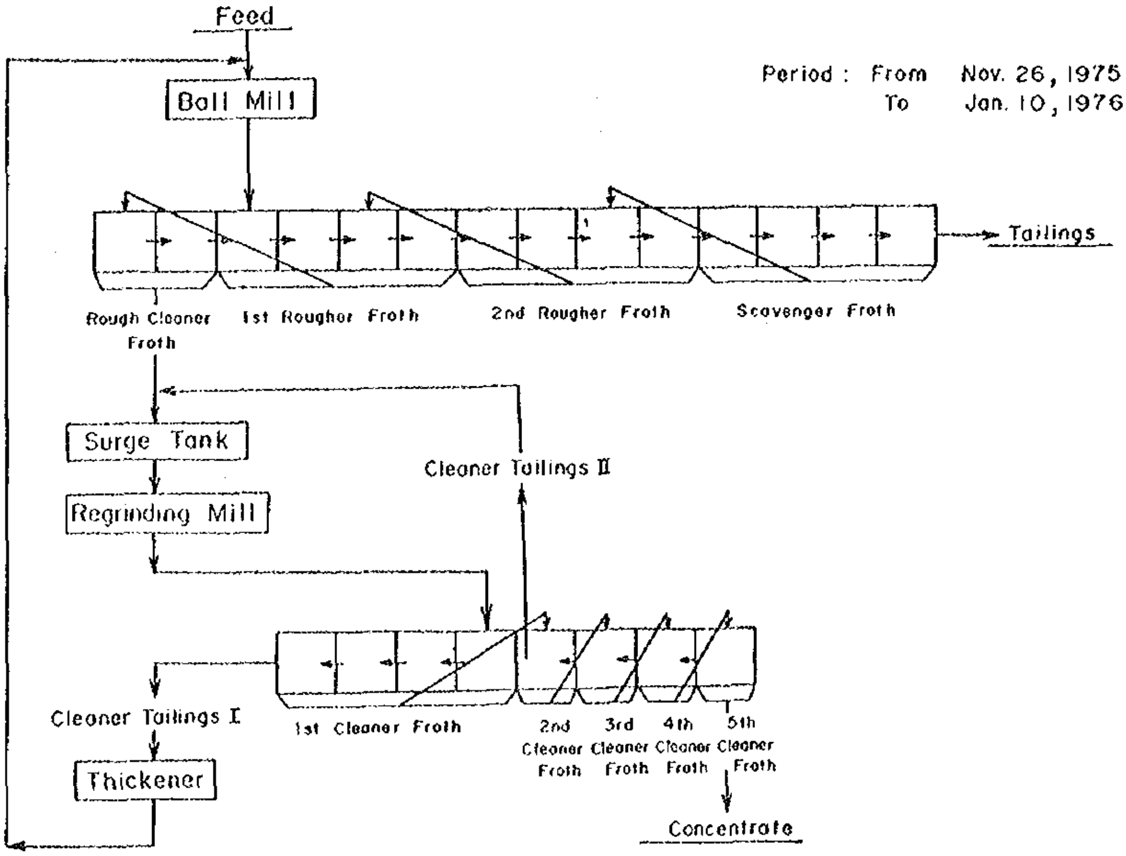


Fig. II-4-2 Schematic Flotation Circuit (with Modified Cleaner)

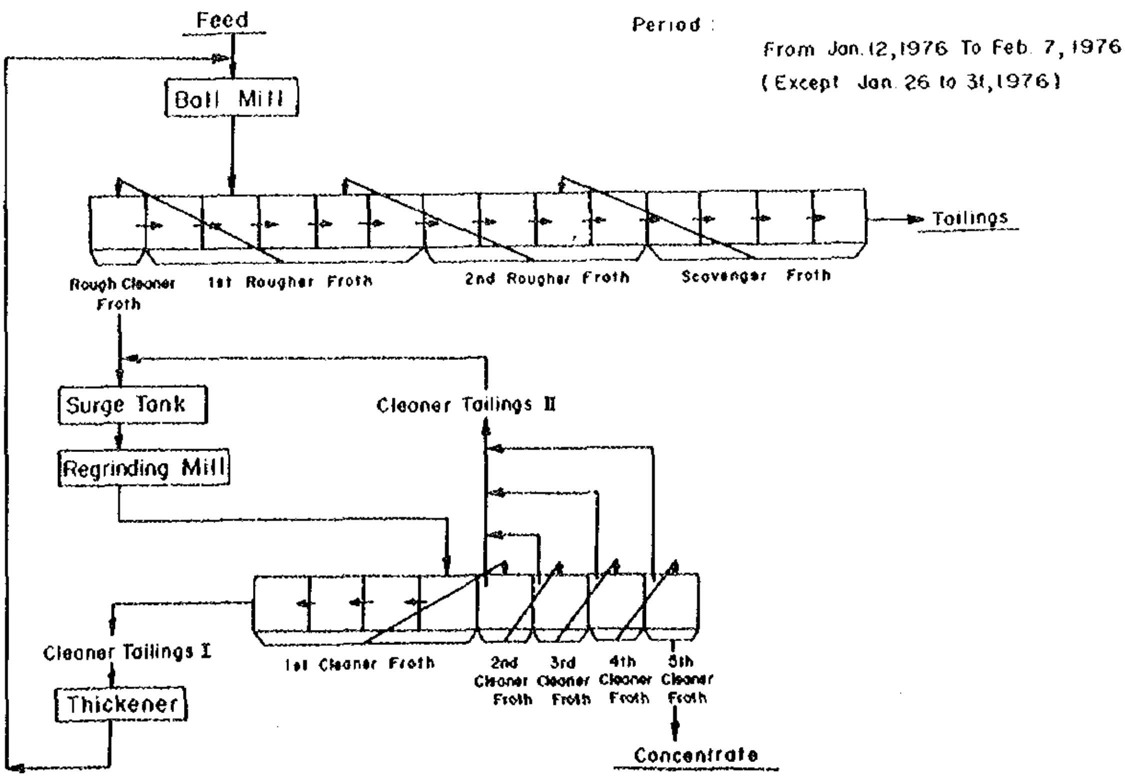
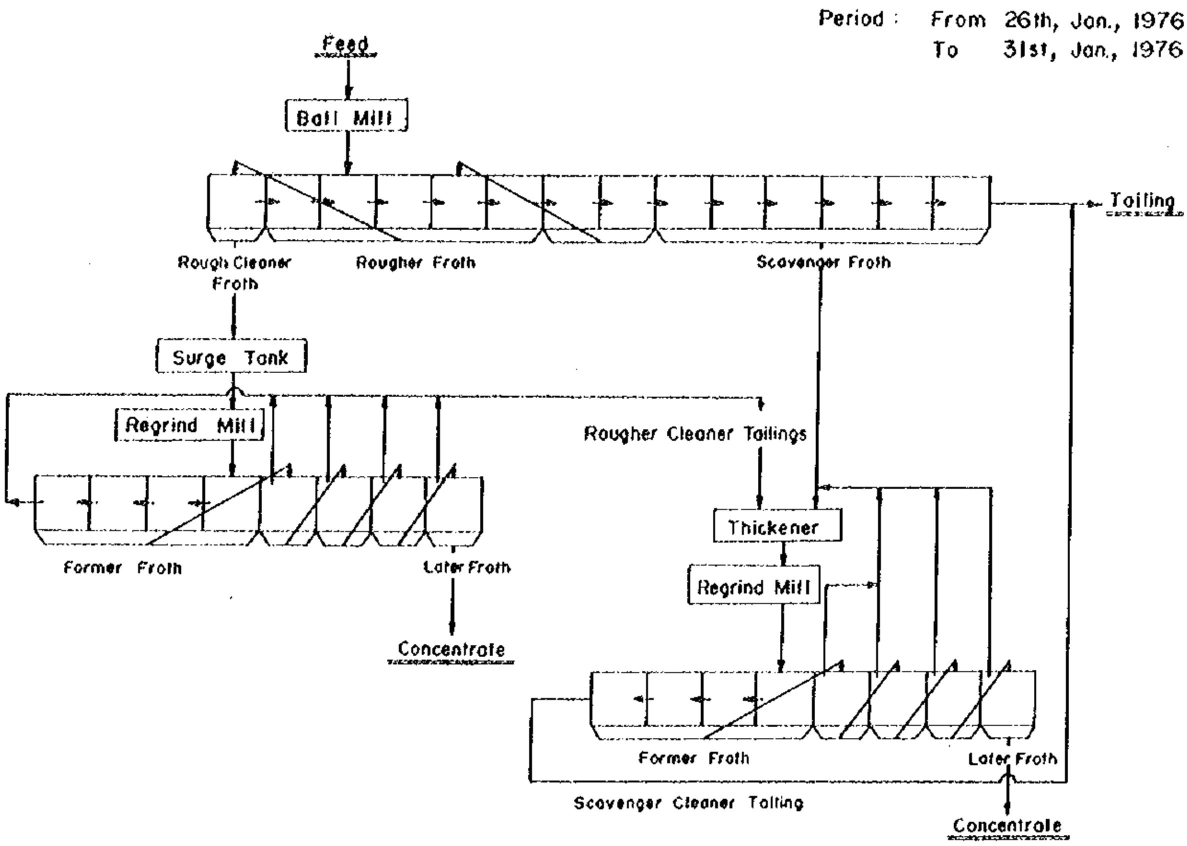


Fig. III-4-3 Schematic Flotation Circuit (Scavenger-cleaner Method)



4-7-3 Assay Result of Flotation Products

An assay of the products, which were collected in the operating test, was carried out at the assay laboratory at Kyisindaung.

There are two kinds of ore samples; a) daily routine samples (take hourly)

b) snap samples

The assay results are shown in Tables III-4-10 and III-4-11.

4-8 Counter Assay

Thable III-4-12 shows the results of the assays, which were carried out at the Central Research Laboratory of Mitsui for both total copper and soluble copper in the samples taken at the plant during the period between January, 21 and February, 2, 1976.

These samples were chosen at random from the total number of samples which were assayed at Kyisindaung.

Fig. III-4-4 illustrates the Comparison of the assay values in total copper, between Monywa and Japan.

The ratio of the soluble copper against the total copper has been proved to be much higher than was initially expected. The cause may be attributed to the fact that the sample was finely crushed by a wet method, and more than 2 months had passed between the sampling and the time when the assay was undertaken.

4-9 Flotation Results (Concentrate grade and Recovery)

The result of the comparison between the grade of feed and the concentrating performance, which are both based on the Assay data in Table III-4-10, is shown as in Fig. III-4-5. Assay data for the period between Jan., 26 and Feb., 2 is excluded.

With this result and the assay result of Table III-4-12, the concentrating recovery on each feed grade has been inferred with an assumed concentrate copper grade of 30%, and the final result can be summarized as shown in Table III-4-13. In short, from crude ore with a copper content of 0.9%, the concentrate of cu 30% can be expected to be recovered, with a recovery rate of 75%.

On the other hand, in the volume II report of Phase II, 1974, the concentrate grade was 20% in copper with a corresponding recovery of 80% according to the Laboratory Batch Test.

4-10 Consumption of Flotation Reagents

The actual consumption rates of the flotation reagents, used in the operating test, are as shown in Table III-4-14.

4-11 On Slaked Lime

On February 6 and 7, 1976, locally made slaked lime was tentatively used instead of the imported product from Japan. The operating test encountered a series of trouble caused by the lime both at the dissolving tank and at the lime feeding pump.

The troubles were mostly caused by impurities such as sand, pebbles, and immatured calcined lime.

Table III-4-15 shows the size distribution and assay result of the slaked lime made in Burma.

For reference, the analysis of the slaked lime of MESCO brand is shown in Table III-4-16.

Table III-4-10 Assay Result of Flotation Products (Daily Samples)

Date	Assay, % Cu							
	Primary Mill Feed	Rougher		Scaven-ger Tail.	Cleaner		Cleaner	
		Feed	Conc.		Feed	Conc.	Tail. I	Tail. II
Nov. 26	0. 69	0. 96	-	0. 33	15. 09	36. 25	-	-
28	0. 93	1. 20	-	0. 36	15. 30	42. 68	-	-
29	0. 67	1. 42	-	0. 46	15. 90	48. 15	-	-
Dec. 1	0. 92	1. 57	-	0. 40	17. 60	49. 95	-	-
2	0. 71	1. 53	-	0. 58	16. 40	49. 90	-	-
4	0. 61	4. 34	-	0. 89	14. 90	44. 00	-	-
5	0. 65	1. 40	-	0. 75	13. 17	37. 97		
6	0. 65	0. 85	10. 89	0. 50	9. 63	18. 12	4. 73	11. 34
8	0. 83	1. 08	9. 95	0. 38	9. 21	27. 62	4. 90	10. 54
9	0. 73	1. 48	11. 29	0. 70	8. 95	27. 68	3. 20	11. 98
10	0. 54	0. 64	6. 70	0. 19	6. 05	18. 48	2. 09	9. 80
11	1. 00	0. 71	4. 86	0. 51	6. 50	19. 48	4. 00	15. 34
12	0. 63	0. 97	4. 37	0. 32	6. 51	18. 91	1. 34	10. 40
13	0. 77	0. 74	4. 52	0. 45	3. 58	12. 83	1. 53	6. 93
15	0. 59	0. 54	5. 25	0. 69	4. 36	13. 37	2. 13	6. 98
16	0. 62	0. 66	6. 72	0. 34	7. 26	14. 50	2. 08	8. 26
17	0. 69	0. 65	5. 84	0. 11	6. 72	14. 34	2. 14	8. 10
18	0. 59	0. 69	5. 94	0. 35	4. 16	13. 12	0. 89	7. 92
19	0. 61	0. 73	7. 13	0. 22	5. 69	11. 88	1. 10	8. 12
20	0. 78	0. 70	8. 37	0. 51	5. 15	14. 50	1. 21	10. 05
22	1. 01	0. 61	5. 27	0. 27	5. 40	16. 14	1. 45	11. 14
23	0. 59	1. 00	6. 54	0. 18	4. 80	16. 28	1. 14	7. 82
24	0. 65	0. 84	8. 14	0. 12	6. 79	20. 32	1. 80	10. 04
26	0. 62	0. 82	-	0. 14	-	21. 78	-	-
27	0. 81	0. 75	-	0. 31	-	14. 21	-	-
28	0. 66	0. 80	-	0. 22	-	20. 46	-	-
29	0. 66	0. 80	-	0. 22	-	20. 46	-	-
30	0. 85	0. 90	-	0. 24	-	15. 71	-	-

- Continued -

Date	Primary Mill Feed	Rougher Feed	Rougher Conc.	Scavenger Tail.	Cleaner Feed	Cleaner Conc.	Cleaner Tail. I	Cleaner Tail. II
		Assay, % Cu						
Jan. 5	0.82	0.88	-	0.09	-	14.40	-	-
6	0.63	0.82	-	0.12	-	15.90	-	-
7	0.71	0.87	-	0.21	-	17.24	-	-
8	0.80	0.92	-	0.40	-	17.68	-	-
9	0.58	1.36	-	0.22	-	20.68	-	-
10	0.75	0.94	-	0.14	-	18.87	-	-
12	0.98	1.13	-	0.21	-	13.87	-	-
13	0.54	0.85	-	0.21	-	15.32	-	-
14	0.73	1.24	-	0.24	-	17.51	-	-
15	0.90	1.60	-	0.36	-	29.00	-	-
16	0.73	1.02	-	0.25	-	32.69	-	-
17	0.70	1.03	-	0.28	-	30.36	-	-
19	0.76	2.30	-	0.36	-	42.10	-	-
20	0.56	2.21	-	0.35	-	42.55	-	-
21	0.85	3.45	-	0.56	-	42.02	-	-
26	0.58	0.79	-	0.25	-	49.20	-	-
27	0.91	0.94	-	0.38	-	50.75	-	-
28	0.68	0.61	-	0.38	-	19.25	-	-
29	0.51	0.56	-	0.19	-	28.50	-	-
30	0.48	0.68	-	0.31	-	27.70	-	-
31	0.52	0.41	-	0.20	-	-	-	-
Feb. 2	0.56	0.85	-	0.15	-	18.35	-	-
3	0.53	0.67	-	0.25	-	29.80	-	-
4	0.83	0.72	-	0.33	-	27.70	-	-
5	0.76	0.72	-	0.66	-	33.62	-	-
6	0.43	0.68	-	0.21	-	32.76	-	-

Table III-4-11 Assay Result of Flotation Products (Snap Samples)

A. Conventional Method

Date	Assay, % Cu																					
	Rougher Feed	Rough Cleaner		1st Rougher		2nd Rougher		Scavenger		Cleaner Feed	1st Cleaner		2nd Cleaner		3rd Cleaner		4th Cleaner		5th Cleaner		Combined Tail	
		Conc.	Tail.	Froth	Tail.	Froth	Tail.	Froth	Tail.		Froth	Tail.	Froth	Tail.	Froth	Tail.	Froth	Tail.	Conc.	Tail.	3rd to 5th	2nd to 5th
Dec. 28	0.85	8.68	0.33	4.41	0.28	1.25	0.18	0.49	0.18	7.42	13.63	1.13	16.78	10.38	18.14	13.78	17.94	16.14	18.92	16.24	-	-
30	0.80	8.25	0.31	3.83	0.28	2.30	0.21	0.38	0.24	6.94	10.14	1.29	9.99	6.50	11.49	7.37	13.82	10.28	14.07	9.85	-	-
Jan. 7	0.89	9.35	0.40	4.85	0.35	1.13	0.28	0.33	0.24	8.05	10.19	1.18	11.16	9.22	10.48	10.33	11.06	10.67	11.74	10.28	-	-
8	-	-	-	-	-	-	-	-	-	9.51	12.05	2.06	15.34	11.27	15.48	16.61	14.41	15.09	18.72	15.29	-	-
9	0.76	9.50	1.20	6.58	0.68	1.25	0.40	0.52	0.26	11.90	15.68	3.28	18.72	14.50	19.99	18.13	19.89	18.82	20.97	19.40	-	-
12	0.82	6.76	0.56	2.51	0.61	1.69	0.38	1.01	0.26	8.13	11.47	2.84	11.07	8.72	10.34	11.17	10.88	9.21	11.96	10.49	-	-
13	0.63	7.99	0.45	6.27	0.26	1.79	0.22	0.75	0.28	-	9.60	-	10.58	8.82	10.00	9.65	11.76	11.76	13.57	11.61	-	-
16	1.08	11.20	-	-	1.36	-	-	-	-	15.82	25.12	8.34	26.48	22.35	-	-	-	-	32.49	-	26.53	-
17	1.38	7.96	4.85	7.72	0.66	5.58	0.40	3.50	0.30	11.83	25.22	5.14	27.60	21.97	-	-	-	-	31.59	-	23.96	-
19	1.10	12.56	11.06	12.42	0.47	8.63	0.40	2.67	0.21	16.10	39.72	14.50	38.07	32.40	-	-	-	-	40.35	-	32.11	-
20	-	-	-	-	-	-	-	-	-	19.69	33.32	11.74	35.30	31.14	-	-	-	-	41.76	-	20.96	-
21	4.15	19.22	14.00	16.44	1.86	11.42	1.20	5.07	0.47	19.96	34.16	14.30	-	-	-	-	-	-	39.52	-	-	33.86
"	2.00	11.72	9.08	10.49	0.85	6.19	0.56	4.29	0.42	28.01	37.19	9.91	38.44	32.16	-	-	-	-	39.38	-	37.33	-
Feb. 2	0.75	10.01	-	-	-	-	-	-	0.66	-	-	-	-	-	-	-	-	-	-	-	-	-
6	0.64	13.92	-	-	-	-	-	-	0.29	16.66	22.04	9.78	-	-	-	-	-	-	25.03	-	-	18.95
"	0.61	12.42	-	-	-	-	-	-	0.46	15.89	18.60	5.62	-	-	-	-	-	-	25.36	-	-	19.06
7	0.87	6.67	-	-	-	-	-	-	0.81	13.26	16.54	5.84	-	-	-	-	-	-	19.56	-	-	14.10

- Continued -

B. Scavenger Cleaner Method

Date		Assay, % Cu														
		Rougher			Scavenger		Rougher Cleaner					Scavenger Cleaner				
		Feed	Conc.	Tail.	Froth	Tail	Feed	Former		Later		Feed	Former		Later	
								Froth	Tail	Conc.	Tail.		Froth	Tail	Conc.	Tail
Jan.	26	0. 68	15. 37	0. 54	4. 05	0. 27	-	-	-	-	-	-	-	-	-	-
	27	0. 49	13. 35	0. 49	2. 80	0. 18	11. 75	33. 20	10. 65	48. 85	33. 25	-	-	-	-	-
	"	0. 89	13. 80	0. 19	4. 20	0. 14	-	-	-	-	-	-	-	-	-	-
	28	0. 36	14. 10	0. 10	2. 17	0. 14	11. 65	13. 35	3. 80	18. 70	18. 00	-	-	-	-	-
	"	0. 66	14. 00	0. 35	1. 37	0. 22	-	-	-	-	-	-	-	-	-	-
	29	*	12. 10	0. 21	*	*	-	-	-	-	-	6. 18	21. 16	5. 07	32. 47	17. 80
	"	*	*	*	*	*	-	-	-	-	-	-	-	-	-	-
	30	0. 70	12. 84	0. 28	2. 98	0. 57	-	-	-	-	-	4. 66	13. 37	1. 38	33. 15	13. 68
Feb.	2	-	-	-	-	-	10. 62	23. 45	2. 87	26. 05	17. 65	-	-	-	-	-

* Sample missed.

Table III-4-12 Result of Check Analysis on Flotation Products

Sampling Date	Name of Sample	Assay, % Cu		
		at Monywa	at Mitsui Laboratory in Japan	
			Total *	Acid Soluble **
21st Jan.	Rougher Feed	4. 15	3. 54	0. 51
	Rough Cleaner Conc.	19. 22	18. 43	1. 30
	Rough Cleaner Tail.	14. 00	12. 97	0. 89
	1st Rough Froth	16. 44	17. 66	1. 10
	1st Rougher Tail.	1. 86	1. 80	0. 38
	2nd Rougher Froth	11. 42	11. 32	1. 08
	2nd Rougher Tail.	1. 20	0. 97	0. 36
	Scavenger Froth	5. 07	4. 49	0. 57
	Scavenger Tail.	0. 47	0. 39	0. 33
	Cleaner Feed	19. 96	19. 92	1. 59
	1st Cleaner Froth	34. 16	34. 88	2. 16
	1st Cleaner Tail.	14. 30	13. 33	2. 45
	Cleaner Conc.	39. 52	39. 20	1. 50
	Cleaner Middling	33. 86	32. 85	1. 94
21st Jan.	Rougher Feed	2. 00	1. 48	0. 39
	Rough Cleaner Conc.	11. 72	11. 72	0. 83
	Rough Cleaner Tail.	9. 08	9. 83	0. 68
	1st Rougher Froth	10. 49	10. 76	0. 67
	1st Rougher Tail.	0. 85	0. 69	0. 27
	2nd Rougher Froth	6. 19	6. 60	0. 63
	2nd Rougher Tail.	0. 56	0. 41	0. 25
	Scavenger Forth	4. 29	4. 02	0. 51
	Scavenger Tail.	0. 42	0. 33	0. 23
	Cleaner Feed	28. 01	21. 69	1. 73
	1st Cleaner Froth	37. 19	37. 41	2. 32
	1st Cleaner Tail.	9. 91	10. 24	1. 88
	2nd Cleaner Froth	38. 44	39. 66	2. 18

- Continued -

Sampling Date	Name of Sample	Assay, % Cu		
		at Monywa	at Mitsui Laboratory in Japan	
			Total *	Acid Soluble **
21st Jan.	2nd Cleaner Tail.	32. 16	30. 80	1. 82
	Cleaner Conc.	39. 38	40. 31	1. 54
	Cleaner Middling	37. 33	35. 73	2. 11
26th Jan.	Rougher Feed	0. 68	0. 51	0. 22
	Rough Cleaner Conc.	15. 37	15. 36	1. 08
	Rough Cleaner Tail.	8. 83	9. 59	0. 66
	Rougher Froth	11. 32	11. 23	0. 70
	Rougher Tail.	0. 54	0. 24	0. 13
	Scavenger Froth	4. 05	4. 17	0. 53
	Scavenger Tail.	0. 27	0. 17	0. 11
27th Jan.	Rougher Feed	0. 49	0. 55	0. 24
	Rough Cleaner Conc.	13. 35	13. 24	0. 93
	Rough Cleaner Tail.	6. 47	6. 51	0. 45
	Rougher Froth	10. 60	10. 47	0. 65
	Rougher Tail.	0. 49	0. 23	0. 15
	Scavenger Froth	2. 80	3. 42	0. 44
	Scavenger Tail.	0. 18	0. 17	0. 11
	Rougher Cleaner Feed	11. 75	11. 90	0. 95
	Rougher Cleaner Former Froth	33. 20	32. 35	2. 01
	Rougher Cleaner Former Tail.	10. 65	8. 93	1. 64
	Rougher Cleaner Later Conc.	48. 85	45. 83	1. 76
	Rougher Cleaner Later Tail.	33. 25	30. 93	1. 83
	Rougher Feed	0. 89	0. 48	0. 20
	Rougher Conc.	13. 80	13. 32	0. 94
	Rougher Tail.	0. 19	0. 19	0. 12
	Scavenger Froth	4. 20	5. 00	0. 64
	Scavenger Tail.	0. 14	0. 15	0. 11

- Continued -

Sampling Date	Name of Sample	Assay, % Cu		
		at Monywa	at Mitsui Laboratory in Japan	
			Total *	Acid Soluble **
28th Jan.	Rougher Feed	0.36	0.50	0.21
	Rougher Conc.	14.10	13.57	0.96
	Rougher Tail.	0.10	0.19	0.12
	Scavenger Froth	2.17	3.02	0.39
	Scavenger Tail.	0.14	0.16	0.11
	Rougher Cleaner Feed	11.65	11.21	0.90
	Rougher Cleaner Former Froth	13.35	13.25	0.83
	Rougher Cleaner Former Tail.	3.80	3.46	0.64
	Rougher Cleaner Later Conc.	18.70	19.19	0.74
	Rougher Cleaner Later Tail.	18.00	16.75	0.99
28th Jan.	Rougher Feed	0.66	0.51	0.22
	Rougher Conc.	14.00	13.48	0.95
	Rougher Tail.	0.35	0.18	0.12
	Scavenger Froth	1.37	1.42	0.18
	Scavenger Tail.	0.22	0.15	0.11
29th Jan.	Rougher Feed	Miss	0.48	0.20
	Rougher Conc.	12.10	13.12	0.93
	Rougher Tail.	0.21	0.19	0.13
	Scavenger Froth	Miss	2.81	0.36
	Scavenger Tail.	Miss	0.15	0.11
	Scavenger Cleaner Feed	6.18	6.17	0.49
	Scavenger Cleaner Former Froth	21.16	20.89	1.30
	Scavenger Cleaner Former Tail.	5.07	5.12	0.94
	Scavenger Cleaner Later Conc.	32.47	35.28	1.35
	Scavenger Cleaner Later Tail.	17.80	16.53	0.98
29th Jan.	Rougher Feed	Miss	0.51	0.21
	Rougher Conc.	Miss	13.68	0.97

- Continued -

Sampling Date	Name of Sample	Assay, % Cu		
		at Monywa	at Mitsui Laboratory in Japan	
			Total *	Acid Soluble **
29th Jan.	Rougher Tail.	Miss	0. 22	0. 15
	Scavenger Froth	Miss	2. 52	0. 32
	Scavenger Tail.	Miss	0. 16	0. 11
30th Jan.	Rougher Feed	0. 70	0. 45	0. 19
	Rougher Conc.	12. 84	11. 18	0. 79
	Rougher Tail	0. 28	0. 21	0. 13
	Scavenger Froth	2. 98	2. 76	0. 35
	Scavenger Tail.	0. 57	0. 33	0. 24
	Scavenger Cleaner Feed	4. 66	4. 64	0. 37
	Scavenger Cleaner Former Froth	13. 37	13. 50	0. 83
	Scavenger Cleaner Former Tail.	1. 38	1. 30	0. 24
	Scavenger Cleaner Later Conc.	33. 15	35. 43	1. 36
	Scavenger Cleaner Later Tail	13. 68	13. 28	0. 78
2nd Feb.	Rougher Feed	0. 75	0. 62	0. 27
	Rougher Conc.	10. 01	10. 89	0. 77
	Scavenger Tail.	0. 66	0. 52	0. 23
	Rougher Cleaner Feed	10. 62	10. 70	0. 85
	Rougher Cleaner Former Froth	23. 45	23. 31	1. 44
	Rougher Cleaner Former Tail.	2. 87	2. 66	0. 49
	Rougher Cleaner Later Conc.	26. 05	29. 45	1. 13
	Rougher Cleaner Later Tail.	17. 65	17. 17	1. 01

* Based upon the Japanese Industrial Standard (JIS)

** 2 grams of ore samples are treated in 50 ml. of 5% solution of H_2SO_4 at the temperature of $80^{\circ}C$, kept in a water both for 20 minutes. Thus, Cu content in the filtered solution is analysed quantitatively.

Table III-4-13 Metallurgical Estimate

Grade of Crude Ore Cu %	Conventional Method				Scavenger Cleaner Method			
	Concentrate			Tailing	Concentrate			Tailing
	Wt., %	Assay % Cu	Recovery, % Cu	Assay % Cu	Wt., %	Assay % Cu	Recovery, % Cu	Assay % Cu
0.5	0.85	30	51.1	0.25	0.95	30	57.0	0.22
0.6	1.15	"	57.7	0.26	1.27	"	63.5	"
0.7	1.48	"	63.3	"	1.59	"	68.3	0.23
0.8	1.80	"	67.4	0.27	1.92	"	72.0	"
0.9	2.12	"	70.7	"	2.25	"	75.0	"
1.0	2.45	"	73.5	"	2.56	"	76.8	0.24
1.1	2.77	"	75.5	0.28	2.87	"	78.2	0.25

Table III-4-14 Consumption of Flotation Reagents

Period	Tonnage, ton	Consumption, g/t					PH	
		Lime	Z11	SFA	Pine Oil	MIBC	Rougher	Cleaner
2nd to 10th, Dec.	109.6	10,036	56	50	No data	No data	11.4	11.4
15th to 30th, Dec.	369.1	6,665	109	68	72	39	11.3	11.4
5th to 14th, Jan.	382.0	9,476	82	84	58	40	11.4	11.4
15th to 31st, Jan.	438.8	10,552	52	58	60	16	11.4	11.4
2nd to 6th, Feb.	242.4	9,468	36	45	63	-	11.3	11.3

Fig. III-4-4 Comparison of Copper Assay Values
(Monywa / Japan)

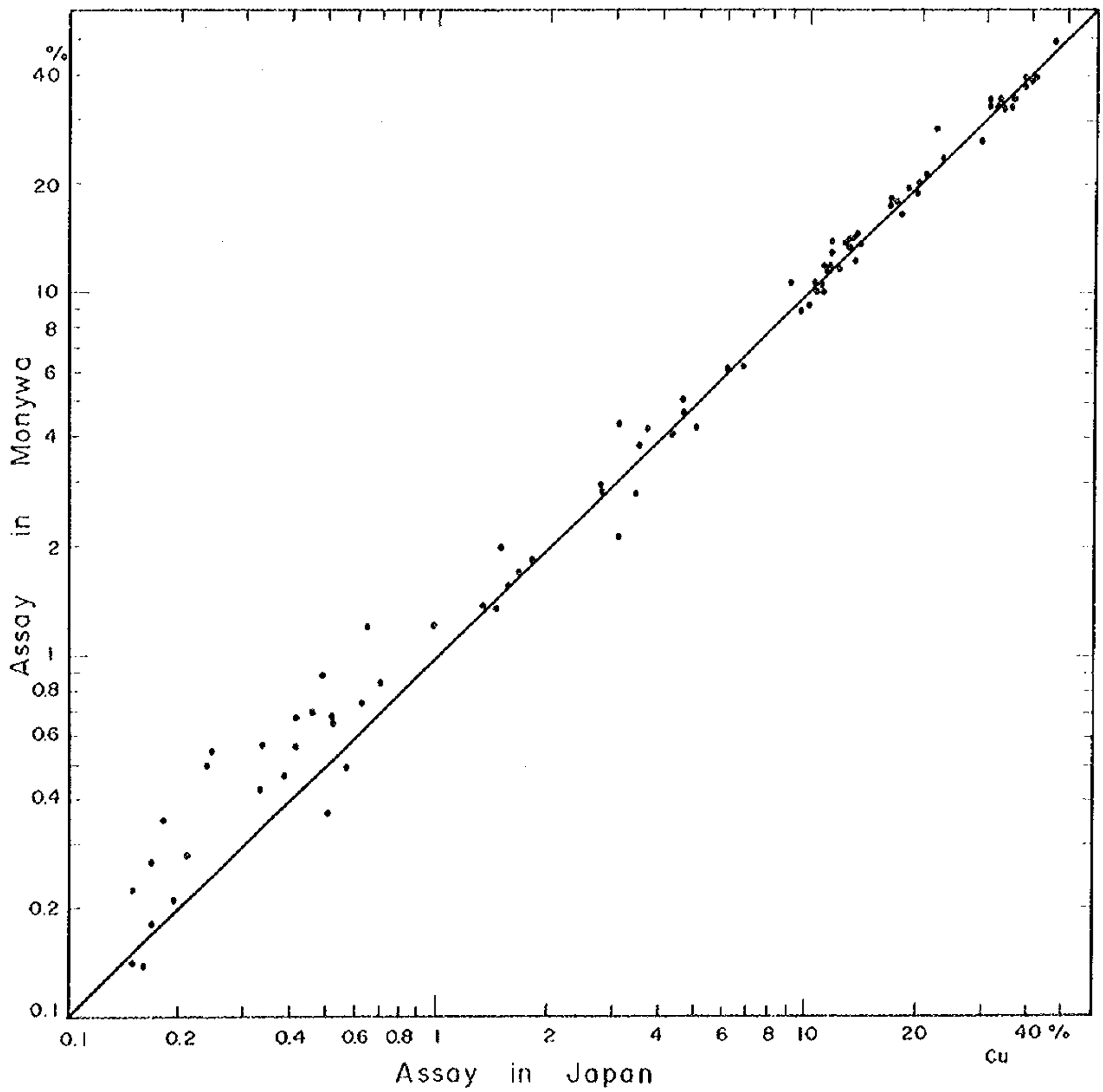


Fig. Ⅲ-4-5 Comparison of Feed Grade and Concentrate / Recovery

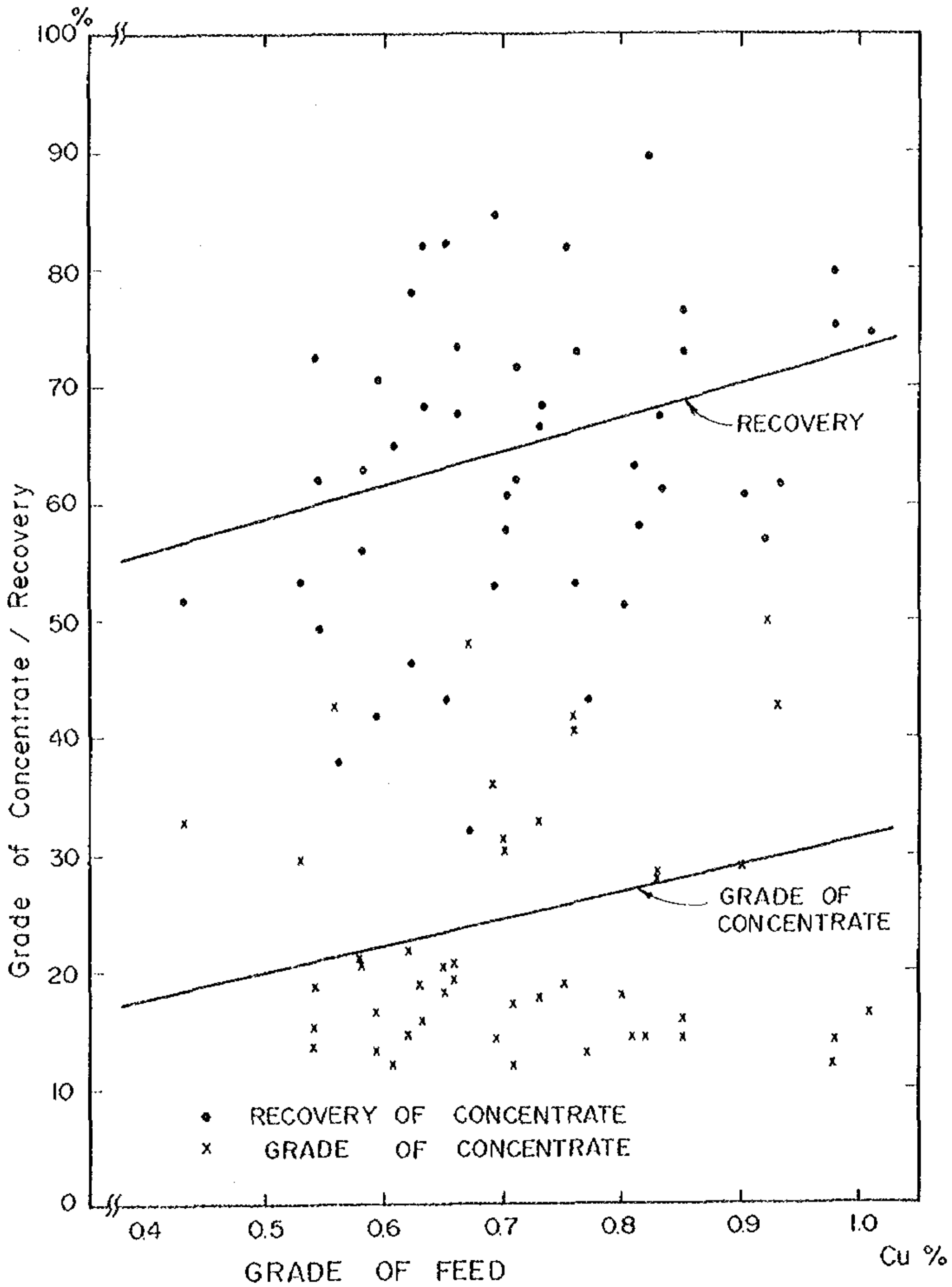


Table III-4-15 Sizing Test on Slaked Lime Produced in Burma

Size		Weight		Assay, %	
mm	mesh	gram	%	CaO	MgO
+6.73	+3	46.5	3.2	-	-
+3.36	+6	37.4	2.6	-	-
+1.68	+10	65.4	4.6	-	-
+0.84	+20	103.7	7.3	-	-
-0.84	-20	1,174.5	82.3	54.2	34.5
Total		1,427.5	100.0		

Table III-4-16 Analysis of Slaked Lime "MESCO Brand"

Assay, %					Fineness Residue	
CaO	MgO	SiO_2	Fe_2O_3 + Al_2O_3	Ignition Loss	0.59mm (28 mesh)	0.149mm (100 mesh)
72.30	2.45	0.24	0.46	23.68	0	0.39

4-12 Sizing Analysis on Rougher Feed and Scavenger Tailing

Typical examples of the sizing analysis on the rougher feed and scavenger tailling are as shown in Table III-4-17.

Table III-4-17 Typical Sizing Analysis on Rougher Feed and Scavenger Tailing

Size		Rougher Feed			Scavenger Tailing		
mm	mesh	Wt., %	Assay % Cu	Distribution, %Cu	Wt., %	Assay % Cu	Distribution, %Cu
+0.210	+65	1.1	0.20	0.3	1.4	0.28	2.3
+0.149	+100	3.6	0.19	1.0	5.0	0.17	5.0
+0.105	+150	9.0	0.27	3.5	10.2	0.14	8.3
+0.074	+200	7.8	0.45	5.0	17.3	0.11	11.1
-0.074	-200	78.5	0.80	90.2	66.1	0.19	73.3
Total		100.0	0.70	100.0	100.0	0.17	100.0

4-13 Size Distribution of Concentrate and Settling Velocity

4-13-1 Size distribution

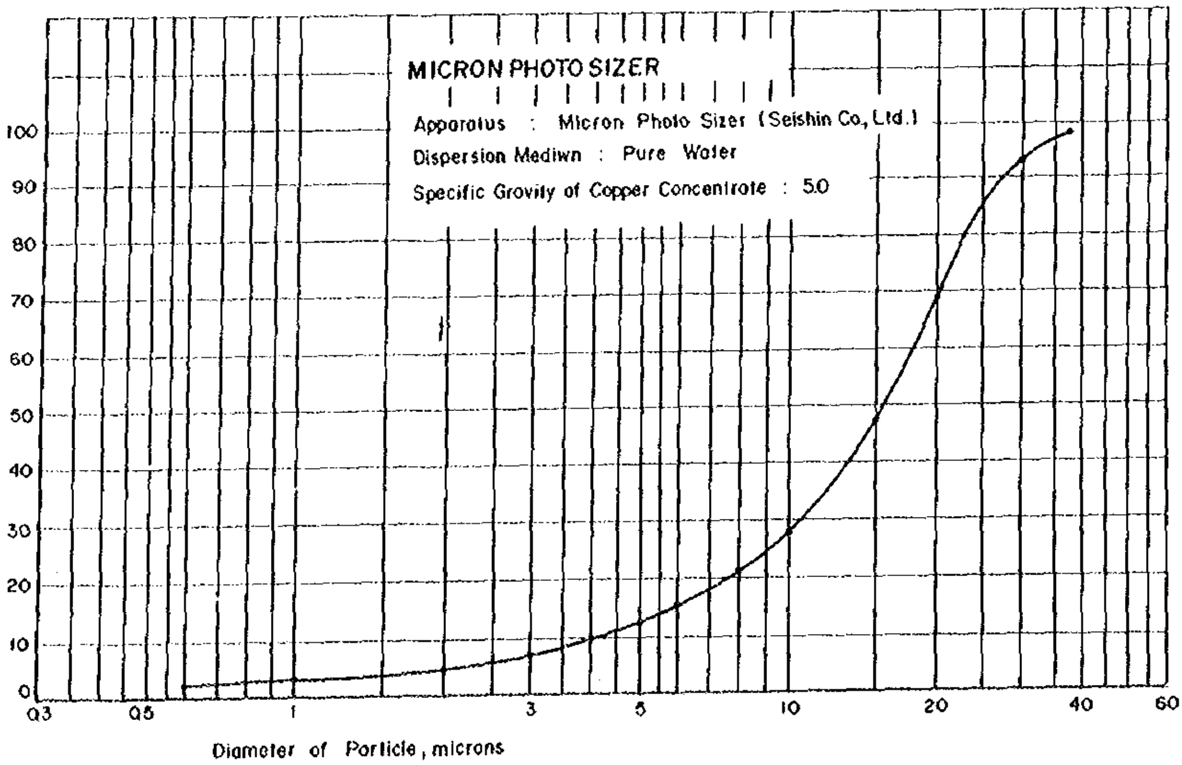
Size distribution of the copper concentrate, which was measured by Micron Photo Sizer, is as shown in Table III-4-18. The values can be illustrated as shown in Fig. III-4-6.

From the figure, it is understood that the passing size of 80% of the concentrate is approximately 23 microns.

Table III-4-18 Sizing Result on Copper Concentrate by Micron Photo Sizer

	Size Distribution											
Micron	+37	+30	+20	+15	+10	+8	+6	+3	+2	+1	+0.6	-0.6
%	2.1	5.9	22.8	21.5	19.7	5.7	5.6	9.8	1.6	1.7	0.9	2.7
Cum. %	2.1	8.0	30.8	52.3	72.0	77.7	83.3	93.1	94.7	96.4	97.3	

Fig. III-4-6 Semilogarithmic Plot of the Particle Size Distribution of Copper Concentrate



4-13-2 Settling Velocity

The settling velocities of the copper concentrate, measured in respective pulp density, are as shown in Table III-4-19.

Table III-4-19 Result of the Setting Test on Copper Concentrate

Pulp Density of Feed, %	30	35	40	50
Settling Velocity, meter/hour	0. 282	0. 215	0.201	0. 182

4-14 Chemical Analysis of Copper Concentrate

The results of the chemical analysis of each concentrate since the Batch flotation test in 1974 are shown in Table III-4-20.

Table III-4-20 Chemical Analysis of Copper Concentrate

Element	Assay					
	June, 1974 *		17th Nov., 1975	20th Nov., 1975	15th Jan., 1976	Average
	Upper Zone	Lower Zone				
Cu , %	20. 08	21. 77	36. 7	42. 8	26. 3	29. 5
Pb , %	0. 01	0. 01	< 0. 01	< 0. 01	<0. 01	<0. 01
Zn , %	0. 01	0. 01	< 0. 01	< 0. 01	< 0. 01	<0. 01
S , %	41. 1	41. 1	34. 5	31. 4	40. 2	37. 7
Fe , %	31. 5	31. 2	21. 6	18. 5	28. 0	26. 2
As , %	0. 03	0. 12	0. 02	0. 01	0. 01	<0. 04
Sb , %	0. 04	0. 04	0. 001	0. 001	0. 01	< 0. 02
Bi , %	0. 01	0. 01	0. 01	0. 01	0. 01	<0. 01
Ni , %	0. 02	0. 02	0. 005	0. 004	0. 01	< 0. 02
MoS_2 , %	< 0. 01	<0. 01	<0. 001	<0. 001	<0. 01	< 0. 01
SiO_2 , %	1. 7	0. 9	1. 5	1. 1	2. 0	1. 4
Al_2O_3, %	1. 0	0. 8	1. 0	0. 7	0. 9	0. 9
Au , g/t	1. 3	1. 3	0. 4	0. 6	0. 9	0. 9
Ag, g/t	19	17	28	35	16	23
Hg , ppm	< 0. 2	<0. 2	<0. 5	1. 0	1. 0	< 0. 6

* Assay results of the concentrates obtained through the batch flotation tests. (See Report Vol. 2, 2nd year, 1974)

There is no particular component, in the concentrate, which later causes trouble during the smelting process.

4-15 Dewatering Test of Copper Concentrate

The results of two separate dewatering tests of the copper concentrate collected at the pilot plant are described.

In one of the tests, filter leaf and dryer were used, and in the other test centrifuge was used. These tests were carried out in Japan.

4-15-1 Filtering test by Filter-Leaf

The result of the vacuum filtering test by filter-leaf is as shown in Table III-4-21.

The filtered cake with water content of 17 to 18% looks thixotropic, and the retained water seaps out soon after the vacuum is cut.

Thus, dryer-treatment is inevitably required for this kind of cake.

Table III-4-21 Result of Vacuum Leaf Test

Diam. of a Filter-Leaf Frame: 114 mm

Filter Cloth : Nyton No. 202

Pulp Density of Feed (%)	Revolution Cycle (min/rev.)	Hourly Capacity (kg/m^2)	Cake	
			Moisture (%)	Thickness (mm)
50	2	190	16.7	4
"	4	135	17.4	6
"	6	105	17.8	7
60	2	345	13.9	6
"	4	230	16.7	9
"	6	175	18.5	12
65	2	500	15.0	10
"	4	255	12.8	11
"	6	270	15.4	14

4-15-2 Filtering Test by Centrifuge

As shown in TableIII-4-22, the water content of the cake can be reduced to below 10% by means of Centrifuge. It may not be necessary to dewater the centrifuged concentrate by additional treatment with a dryer.

Table III-4-22 Result of Filtration Test by Centrifuge

Basket of Centrifuge	:	Diameter	375 mm
		Depth	230 mm
		Capacity	11.7 liters
Filter Cloth	:	Cotton No.26	
		Permeability	1.19 $cm^3/sec/cm^2$
Feed Charging	:	Overflow type	

Basket Speed, rpm	1,650		2,000	
Centrigufal Force, G	571		838	
Feed, Charging Time, min	$4\frac{1}{2}$		$5\frac{1}{3}$	
Weight, wet kg/min	22.7		20.5	
Pulp Density, % solid	30		30	
Dewatering Time, min	5	10	5	10
Cake, Weight, wet kg	-	18.9	-	29.5
Moisture, %	10.4	9.3	9.6	9.3
Thickness, mm	-	44	-	59
Filtrate, Weight, wet kg	-	15.85	-	10.9
Overflow, Weight, wet kg	-	67.6	-	68.2

4-15-3 Dryer Test of Filter Cake

Using a Fluid Bed Dryer, a dryer-test was carried out for the filter cake with adjusted water content of 18%. The result is as shown in Table III-4-23.

From the result of the test, it has been found that approximately 150,000 kcal of heat is needed to decrease the water content of one ton of the concentrate cake from 18% into 8%.

Table III-4-23 Result of Drying Test by Fluid Bed Dryer

Size of Fluid Bed : Availabel 4.8 m^2

Room Temperature, dry bulb : 17°C

wet bulb : 16°C

Feed, Weight, wet kg/Hr	300
Moisture, %	18
Apparent Specific Gravity	2.7
Temperature, °C	12
Discharge, Weight, wet kg/Hr	265
Moisture, %	7.2
Apparent Specific Gravity	2.1
Temperature, °C	48
Water evaporated, kg/Hr	35
Temperature of Hot Air, Inlet, °C	180
Exit, °C	57
Temperature of Exhoust, Dry bulb, °C	44
Wet bult, °C	38
Drying Speed, kg/m^2/Hr	240
Drying Time, min	6

4-16 Settling Test on Tailing

The result of the settling test for the tailing collected at the pilot plant is as shown in TableIII-4-24. The settling velocity was found to be very slow and unsatisfactory at 0.03 m/hr.

Table III-4-24 Result of Settling Test on Tailing

Pulp Density of Feed : 20% solid

Time of Settling, min.	15	30	45	60	120	180
Depth of Clear Water, min	8.3	16.5	23.7	30.3	47.4	60.6

4-17 Analysis of Top Water at Tailing Pond

The result of the assay of the liquid sample, which was scooped from the tailing disposal pond to the north of the pilot plant, is shown in TableIII-4-25. The content of heavier metals was found to be small, thus the top water may not cause pollution problems.

Table III-4-25 Chemical Composition of Top Water at Tailing Pond

Concentration (ppm)											pH
SS*	Cu	Pb	Cd	As	Hg	Mn	Cr	Ca	SO_4	P	
21	0.05	<0.02	<0.005	<0.02	<0.005	0.03	<0.02	31	263	0.2	6.2

* Total Dissolved Solid

4-18 Batch Flotation Test with River Water

4-18-1 With water from Yama Stream and Public supply in Rangoon (1974).

In early April, 1974, using the river water from Yama stream, Hashizumi, the metallurgist tried the Batch Flotation Test for the ore collected from Sabedaung core pieces at the Metallurgical Laboratory of MMDC in Rangoon.

Comparative flotation tests were done for the dried ore crushed under 150 mesh by both the river water from Yama stream in dry season and the public water supplied from Rangoon city.

Samples of the products concerned were delivered to Japan to be analyzed. The assay result is as shown in Table III-4-26, and demonstrates that the river water of Yama stream is better fit for the copper flotation.

Table III-4-26 Result of Batch Flotation Test by Yama Stream Water (Hashizumi)

Date tested : 8th Apr., 1974

Test Machine : 500 g Denver Type belong MMDC

Water Tested	Assay, % Cu			
	Cleaner Froth	Cleaner Tail	Scavenger Froth	Scavenger Tail
Yama Stream Water	7.65	1.26	1.72	0.20
Rangoon City Water	4.28	1.64	2.80	0.27

4-18-2 With water from Yama Stream and Chindwin River

The afore-mentioned test was undertaken only with water from Yama stream in dry season.

It was therefore planned to carry out flotation tests at a later date with water from the Chindwin river, as well as from Yama stream, both during the dry and rainy seasons.

In February, 1976, water from the said rivers and the muddy sediments from the lower river shores were collected. Then, the mud and the river water were mixed together and adjusted to give a muddy solution of 20,000 ppm in solid.

24 houres later, the top water was taken out and used for the flotation tests to be compared with the said river water of dry season.

The assumption was made that the aforementioned top water might correspond to the river water of the rainy season which had passed both the sand settling pond and the filtering pond.

The testing pulp was prepared by putting the said water into the dewatered cake of the ore, taken from the feed of the cleaner at the conditioner of 900 mm in the pilot plant.

The result of the comparative flotation test of this specially prepared feed is shown in Table III-4-27.

No remarkable differnce in performance was recognized between those two kinds of river water.

In other words, the river water from both the Chindwin and Yama is available all through the year in a quality suitable for use at the concentrating plant.

Table III-4-27 Result of Batch Flotation Test by River Water (U Sin Kyin)

Date tested : 9th Feb., 1976

Feed : Cleaner Feed sampled at Conditioner

Test Machine : 500 g FW Type

Water Tested	Product	Weight		Assay	Recovery
		Gram	%	% Cu	% Cu
Chindwin River (Rainy)	Feed	120	100.0	12.90	100.0
	Froth	30	25.0	22.34	43.3
	Tailing	90	75.0	9.75	56.7
Chindwin River (Dry)	Feed	136	100.0	13.74	100.0
	Froth	56	41.2	18.96	56.8
	Tailing	80	58.8	10.09	43.2
Yama Stream (Rainy)	Feed	130	100.0	12.54	100.0
	Froth	44	33.8	17.35	46.8
	Tailing	86	66.2	10.09	53.2
Yama Stream (Dry)	Feed	114	100.0	12.42	100.0
	Froth	40	35.1	16.27	46.0
	Tailing	74	64.9	10.34	54.0

Chapter 5 Conclusion and Proposal

5-1 Conclusion

Copper ore, excavated in Sabedaung Tunnel, was metallurgically tested at the 50 t/d pilot plant which had been constructed at the northeastern foot of Kyisindaung, in Monywa area during the period between Movember 11, 1975 and February, 14 1976.

The purpose of the concentrating test was to prove the results of the Batch Test of 1974 under the conditions of continuous operation, as well as to collect information necessary for the framing of the feasibility report.

The main items clarified by means of concentrating test are summarized as follows:

1) By improving the flow sheet, concentrate grade of 30% cu with a recovery of 75% was obtained with good prospects for future operations.

The results of the Batch test of 1974, in which concentrate grade of 20% cu was obtained from crude ore of 0.9% copper and with a recovery rate of 80% could not be reproduced.

2) Washing of the feed is not necessary, for the amount of minus 200 mesh ore is only 1.2% of the feed on average.

3) Grinding Work Index by ball mill has been found to be 12.5 on average, thus Monywa copper ore may be considered as medium in the scale of grinding resistivity.

4) In flotation, the proper size distribution of rougher feed has been found to be between 75 and 82% passing with a 200 mesh.

On the other hand, with cleaner feed it has been found sufficient to grind into 325 mesh as this allows 95 to 100% to pass.

Despite the former understanding, the comminution into minus 400 mesh by regrinding ball mill has been proved unnecessary.

5) As far as the flotation is concerned a retreating circuit with regrinding facility, which is exclusively to cope with flotation middling, should be adopted in addition to the normal cleaner circuit.

By this arrangement, stabilized operation and improvement of recovery in flotation can be expected simultaneously.

6) As far as the reagents are concerned, the following 5 kinds are expected to ensure good flotation results, namely;

Slaked lime, PH regulater

Z-11, SFA, collector

Pine Oil, MIBC, frother

However, if local slaked lime is to be used, careful attention should be paid to its quality.

7) The particle size of the concentrate has been found to be very fine. The 80% passing size has been observed at about 23 microns.

8) Through the dewatering test of the copper concentrate in Japan, the water content of the concentrate cake could be reduced to below 10% by centrifuge.

On the ogher hand, the test result by means of vacuum filter gave a water content of 17 to 18%.

Thus, in the latter case, a dryer is needed to reduce the water content to 10% or lower.

9) According to the average values of the results of 5 assays on the copper concentrate, the main ingredients have been proved as follows;

Cu 29.5%

S	37.7%
Fe	26.2"
Au	0.9 g/t
Ag	23 g/t

content in comparison with the average grade of Sabedaung deposit with an unstable aspect concerning the nature of ore.

It may have been caused by the fact that the ore come from the portion comparatively close to the top of the Sabedaung deposit, in other words, the ore being excavated from the vicinity of the boundary portion between the oxidized leached zone and the ore body. In general, there is a difference is the nature of the ore produced from different portions of the deposit i.e. from the center, bottom, or the edge of the deposit.

It is considered, therefore, that the influence of this difference on the recovery can not be neglected.

10) Consideration should be given to the prevention of ore oxidation, for it is considered that the chalcocite in the Monywa copper ore can change into sulphuric-acid-soluble copper minerals as a result of spontaneous alteration.

11) Concentrating tests have not been carried out on the ore from either the Kyisindaung or the Sabedaung South deposits.

The ore from these deposits is supposed to be of similar composition to that of the Sabedaung deposit.

However, on the other hand, there may still be some slight defference in the nature of ore.

Regarding the above points, it is proposed that the following tests be carried out, to obtain information, which will contribute to the stability of normal operations;

(1) Establishment of Standard Mill Oparation

For establishing a stabilized mill oparation, it is indispensable to maintain the pilot plant to continue the operation test for the ore from Sabedaung deposit. Fundamentally, however, it has been proved that the ore is fit for flotation by producing a concentrate of 30% in copper with a recovery of 75%.

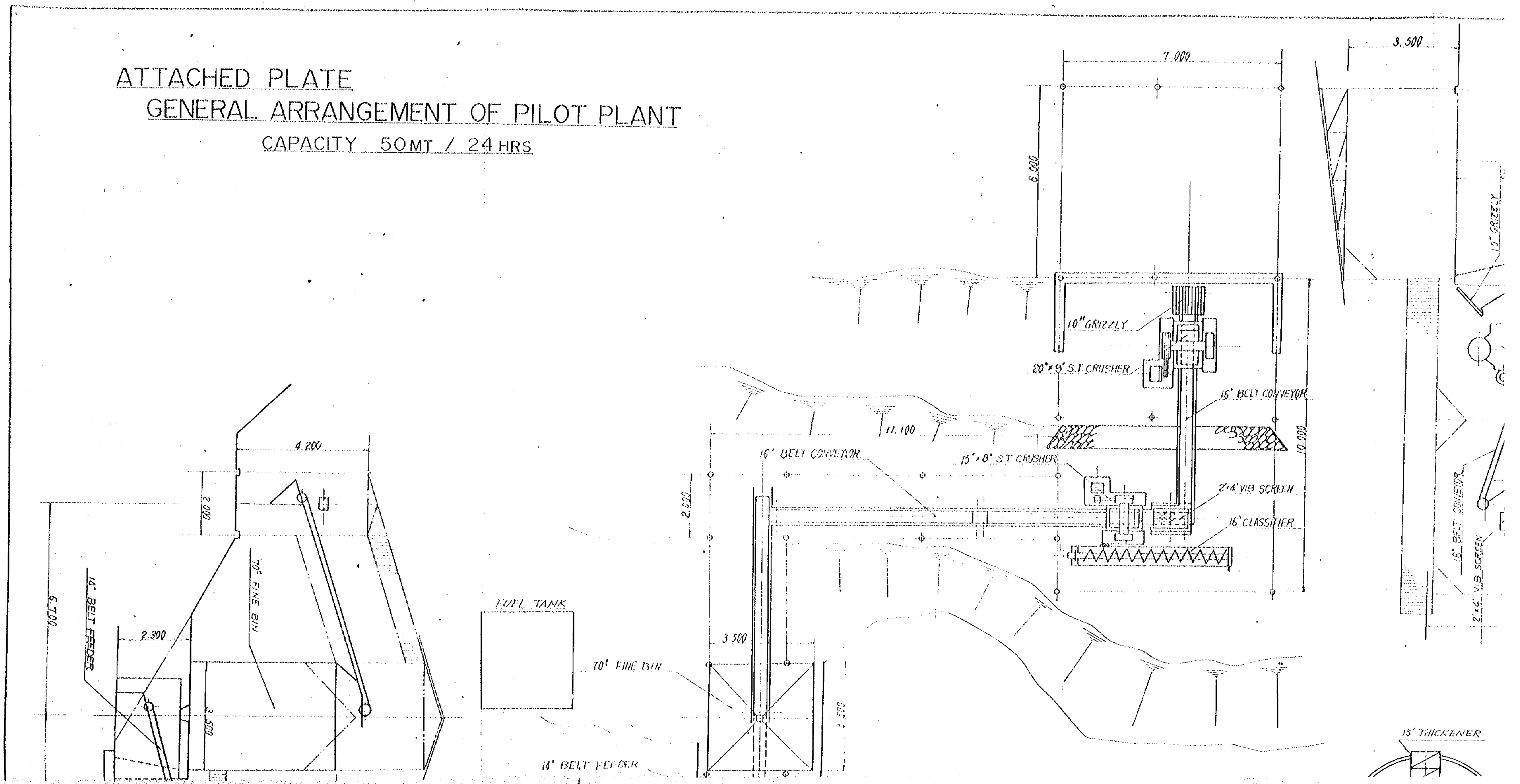
(2) Mill Test of Ore from Kyisindaung and Sabedaung South deposits

Differences in the nature of the ores from the three deposits should be clarified at an early date, by flotation tests before the inauguration of the mine.

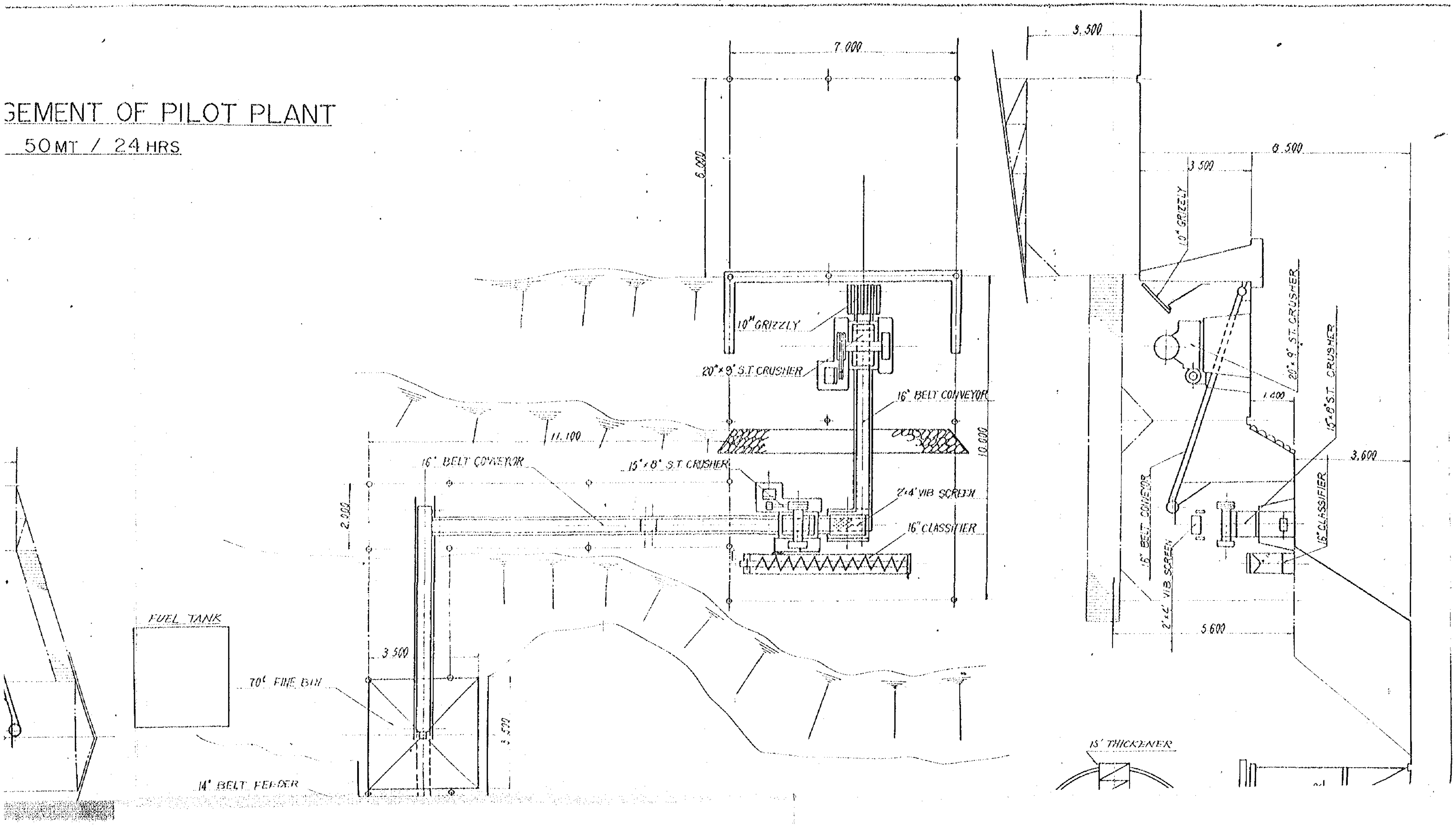
ATTACHED PLATE

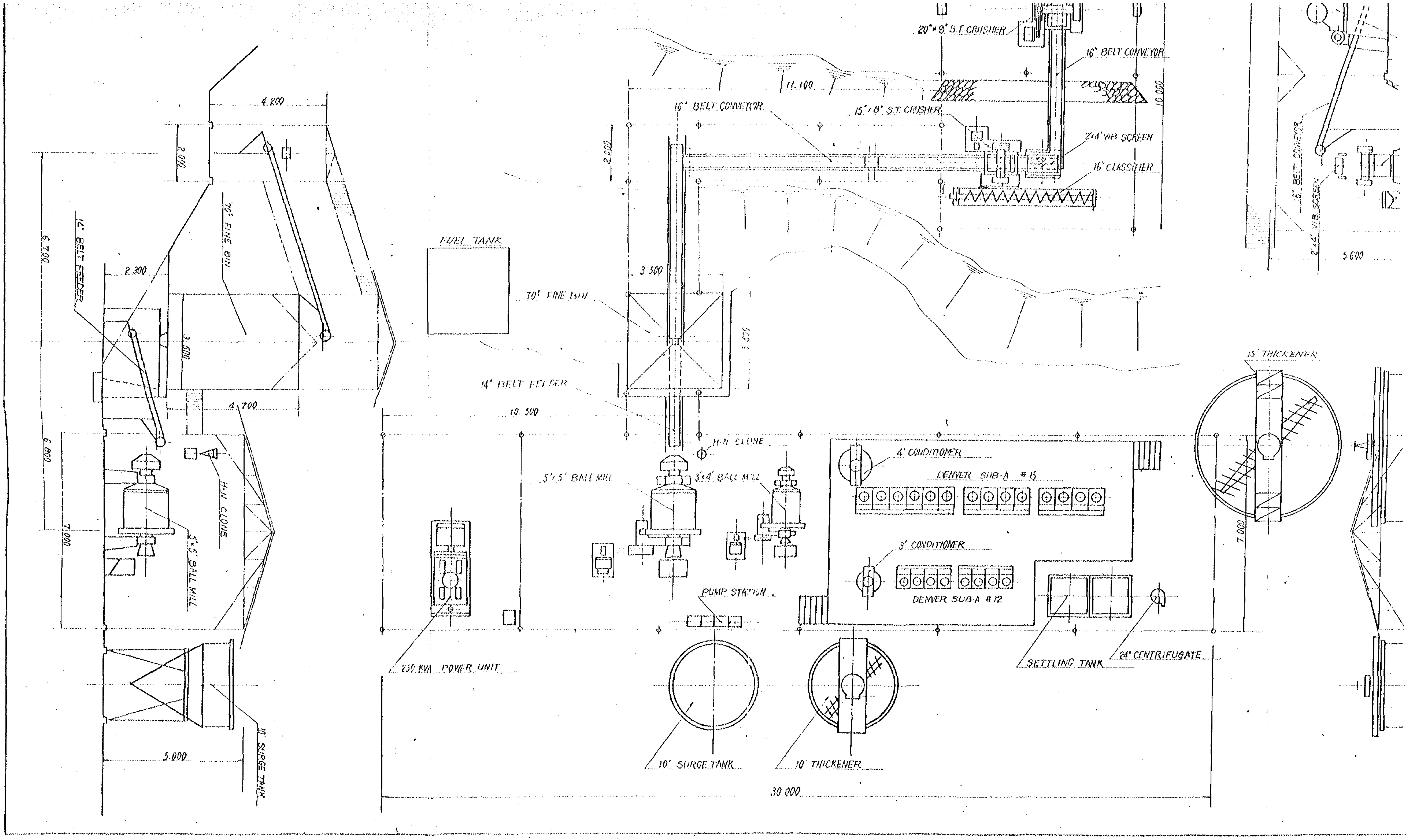
GENERAL ARRANGEMENT OF PILOT PLANT

CAPACITY 50MT / 24HRS

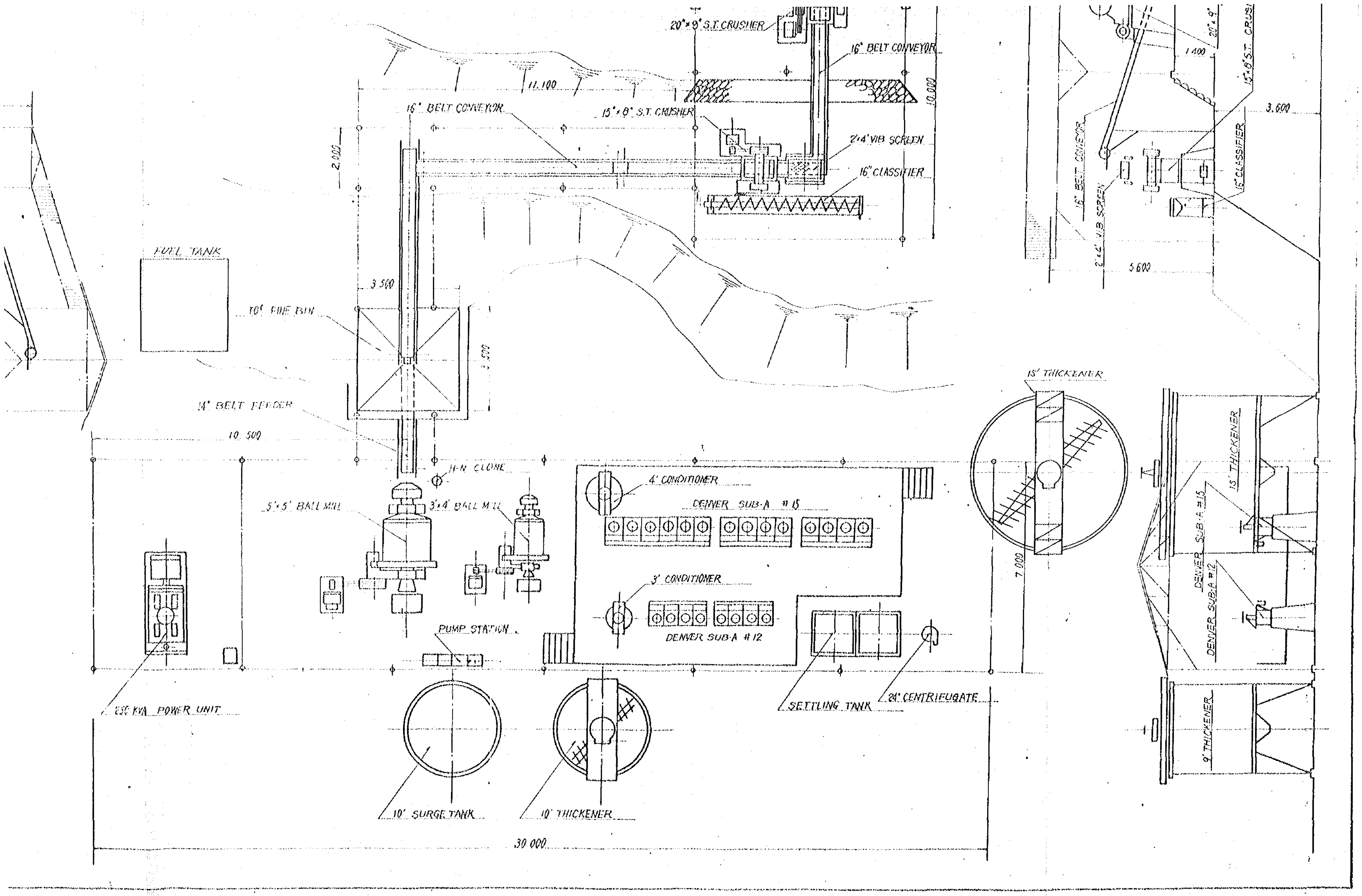


GEMENT OF PILOT PLANT
50 MT / 24 HRS

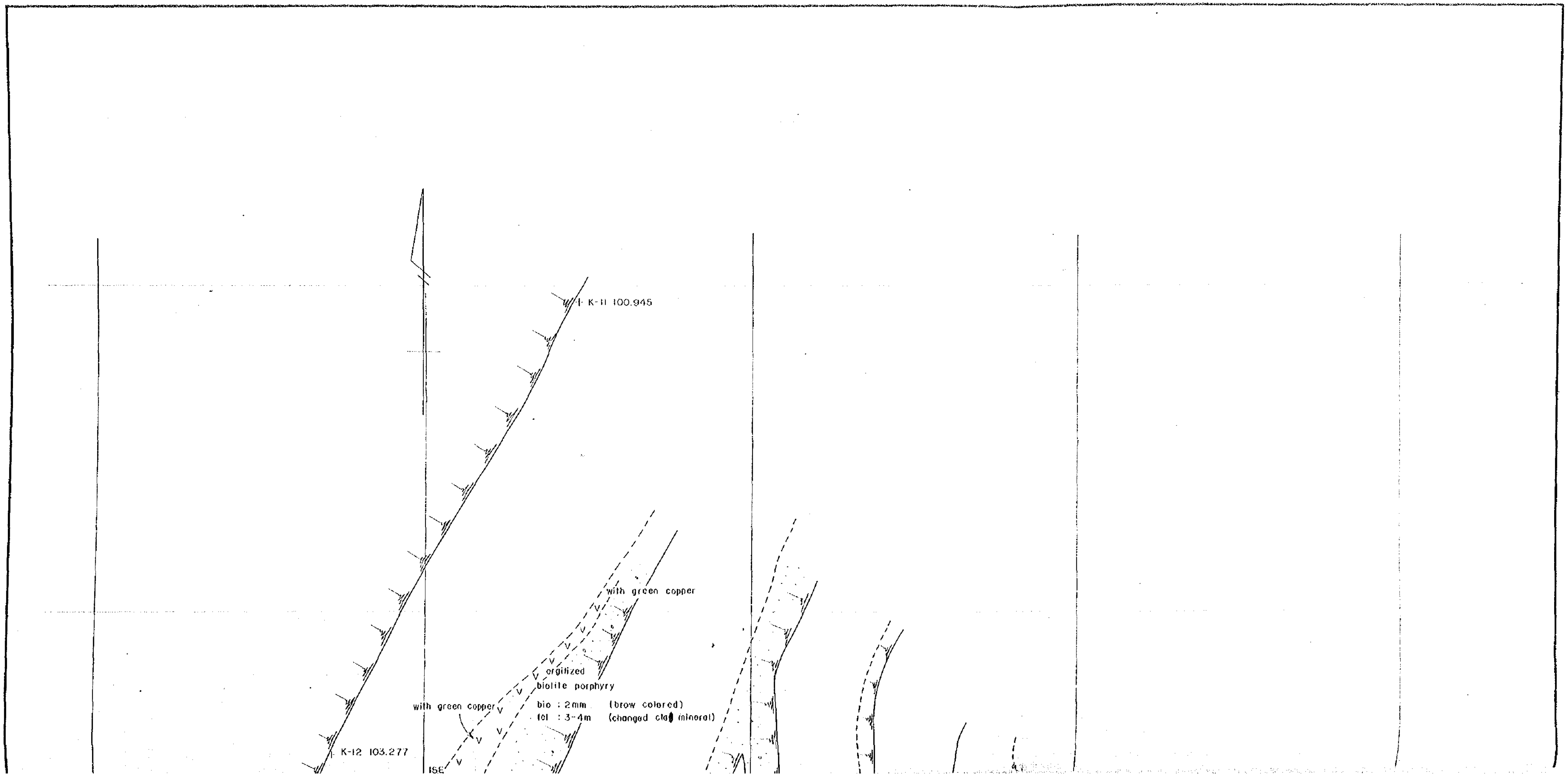


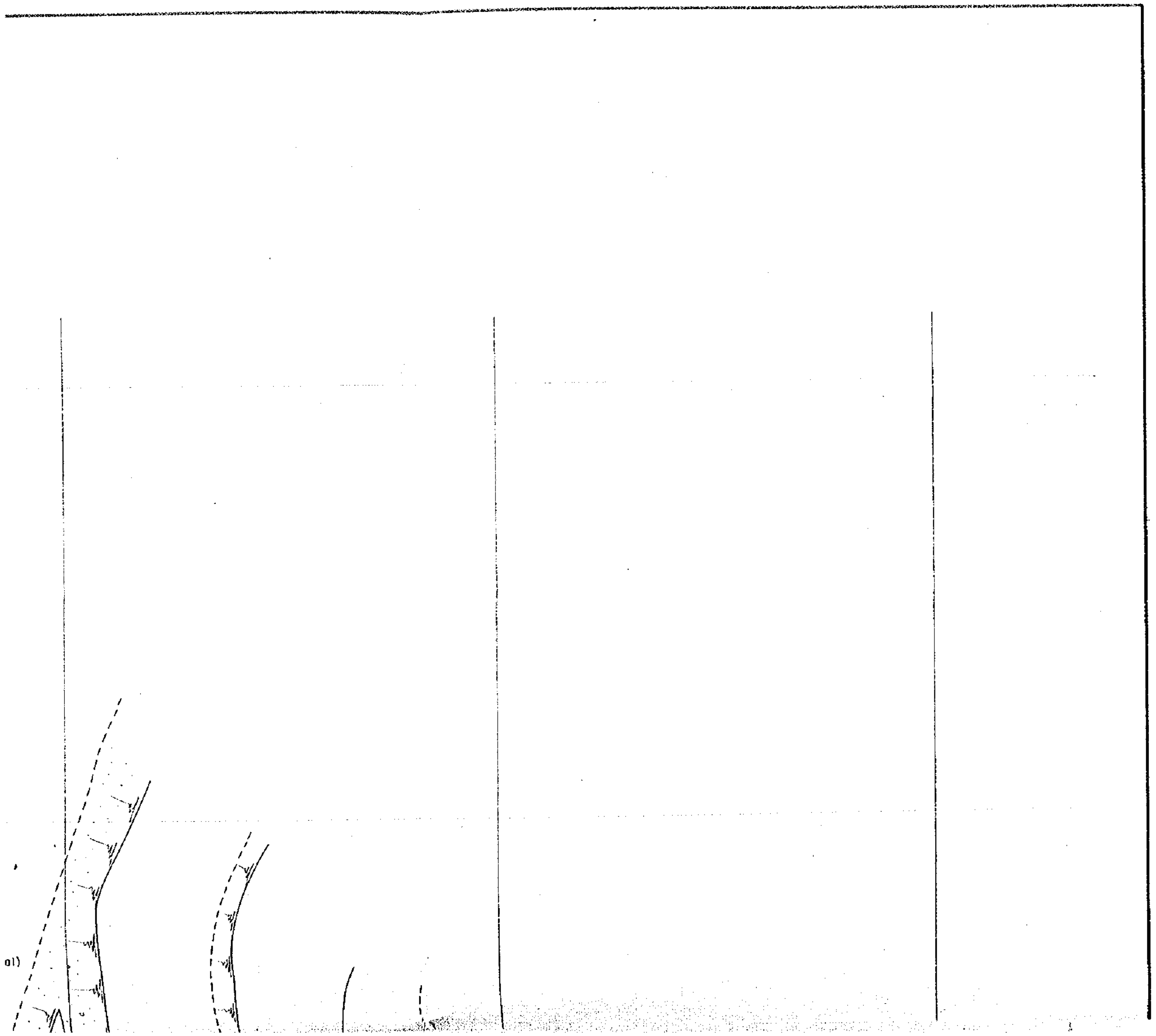


4.200
2.000
6.700
16" BELT FEEDER
70' FINE BIN
2.300
3.500
4.700
6.800
7.000
H-N CLONE
5'x5' BALL MILL
5.000
10' SURGE TANK
FUEL TANK
70' FINE BIN
14" BELT FEEDER
10.500
5'x5' BALL MILL
3'x4' BALL MILL
H-N CLONE
250 KVA POWER UNIT
PUMP STATION
10' SURGE TANK
10' THICKENER
30.000
4' CONDITIONER
DENVER SUB-A #15
3' CONDITIONER
DENVER SUB-A #12
SETTLING TANK
24" CENTRIFUGATE
15' THICKENER
7.000
20"x9" S.T. CRUSHER
16" BELT CONVEYOR
11.100
10.000
16" BELT CONVEYOR
15"x8" S.T. CRUSHER
2.600
2'x4' VIB SCREEN
16" CLASSIFIER
3.500
3.500
16" BELT CONVEYOR
2'x4' VIB SCREEN
5.600



20"×9" S.T. CRUSHER
16" BELT CONVEYOR
11.100
10.000
16" BELT CONVEYOR
15"×8" S.T. CRUSHER
2×4' VIB SCREEN
16" CLASSIFIER
2.000
FUEL TANK
3.500
10' FINE BIN
3.500
14' BELT FEEDER
10.500
H-N CLONE
5'×5' BALL MILL
3'×4' BALL MILL
4' CONDITIONER
DENVER SUB-A #15
3' CONDITIONER
DENVER SUB-A #12
PUMP STATION
250 KVA POWER UNIT
SETTLING TANK
24" CENTRIFUGATE
10' SURGE TANK
10' THICKENER
30.000
15' THICKENER
7.000
1.400
20"×9"
15"×8" S.T. CRUSHER
3.600
16" BELT CONVEYOR
2'×4' VIB SCREEN
15" CLASSIFIER
5.600
15' THICKENER
DENVER SUB-A #15
DENVER SUB-A #12
9' THICKENER


K-11 100.945
with green copper
orgilized
biolite porphyry
with green copper
bio : 2mm (brow colored)
fel : 3-4m (changed cla mineral)
K-12 103.277
ISE



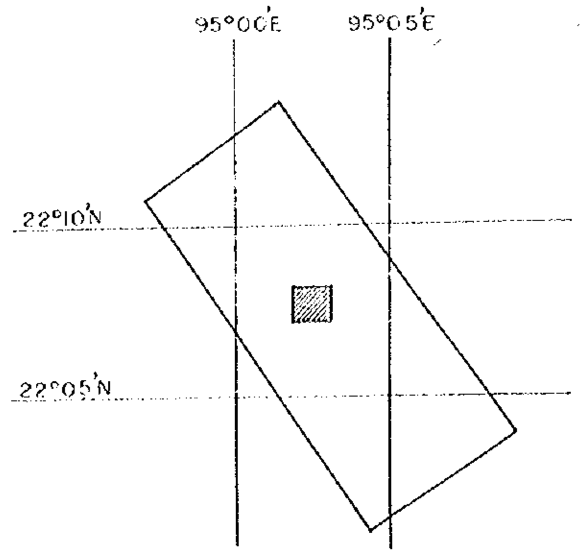
PL.I.-8-3

GEOLOGICAL SURVEY OF
MONYWA AREA UNION OF BURMA
(PHASE IV)

GEOLOGICAL SKETCH OF MINE SITE AT SABEDAUNG

Scale 1:300

5 0 5 10m



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METAL MINING AGENCY
JAPAN INTERNATIONAL COOPERATION AGENCY
GOVERNMENT OF JAPAN

JUNE 1976

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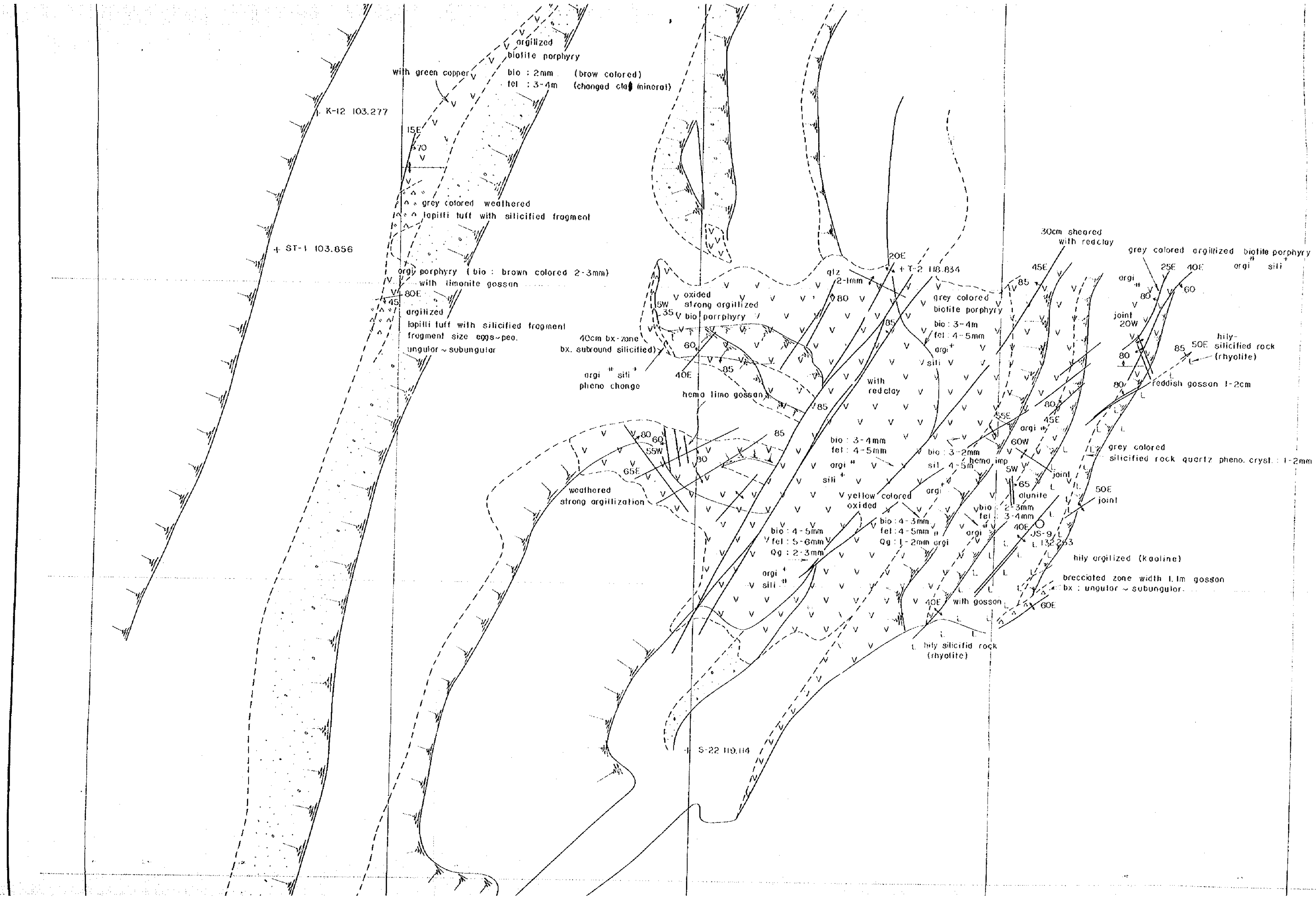
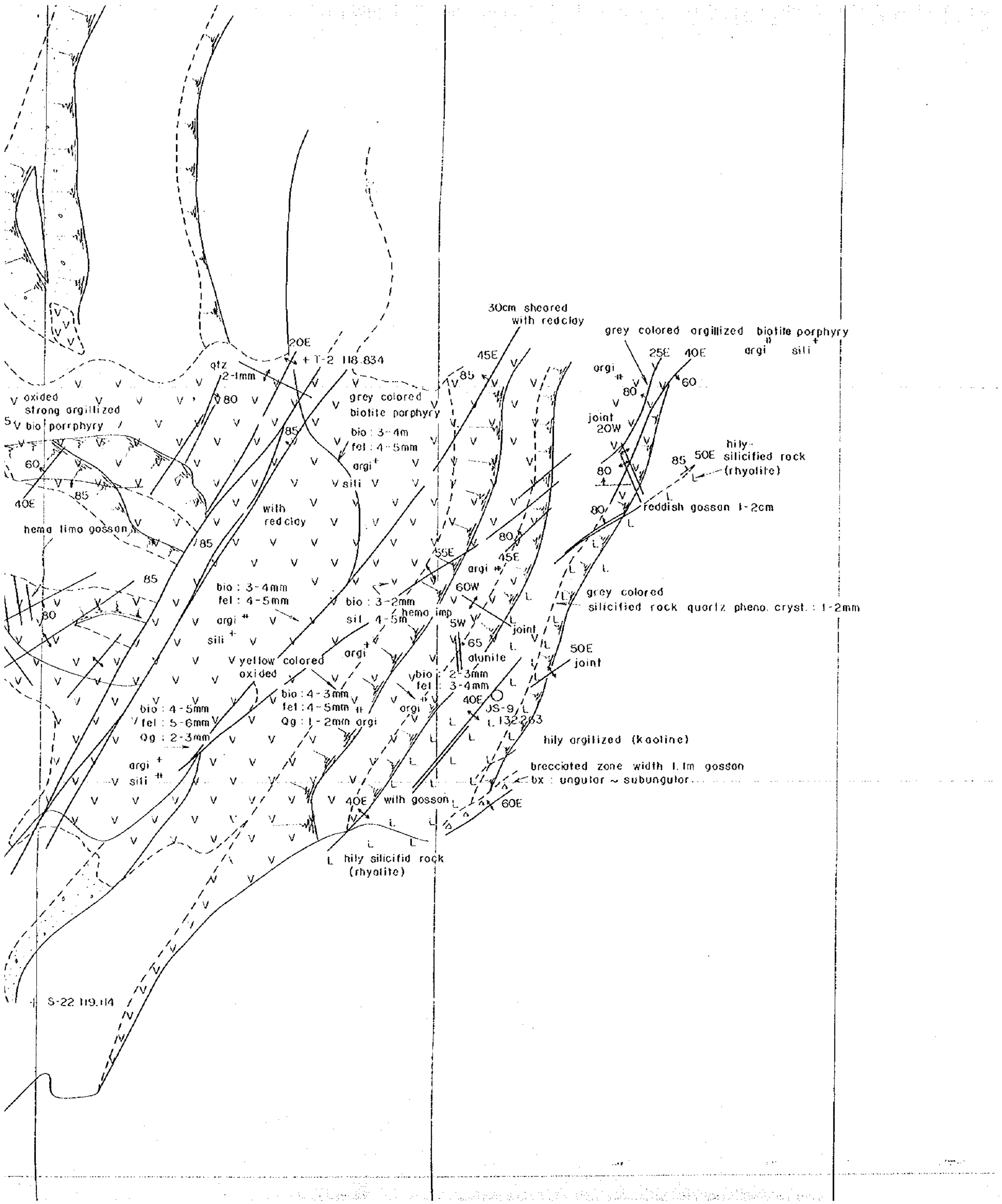

argilized
biotite porphyry
with green copper
bio : 2mm (brow colored)
fel : 3-4m (changed clay mineral)
K-12 103.277
15E
70
grey colored weathered
lapilli tuff with silicified fragment
ST-1 103.856
argi porphyry (bio : brown colored 2-3mm)
with limonite gossan
80E
45
argilized
lapilli tuff with silicified fragment
fragment size eggs~pea.
ungular ~ subungular
40cm bx-zone
bx. subround silicified)
argi# sili+
pheno change
oxided
strong argillized
bio porphyry
5W
35
60
40E
85
hema limo gossan
qtz
2-1mm
20E
T-2 118.834
80
grey colored
biotite porphyry
bio : 3-4m
fel : 4-5mm
argi+
sili
with
redclay
bio : 3-4mm
fel : 4-5mm
argi#
sili+
yellow colored
oxided
bio : 4-5mm
fel : 5-6mm
Qg : 2-3mm
argi+
sili#
bio : 4-3mm
fel : 4-5mm
Qg : 1-2mm argi
bio : 3-2mm
sil : 4-5m
hema imp
30cm sheared
with redclay
45E
85
grey colored argillized biotite porphyry
argi# sili+
25E 40E
60
joint
20W
85 50E
hily-
silicified rock
(rhyolite)
reddish gosson 1-2cm
80
55E
60W
45E
grey colored
silicified rock quartz pheno. cryst. : 1-2mm
joint
50E
joint
alunite
bio 2-3mm
fel 3-4mm
JS-9
132.63
40E
hily argilized (kaoline)
brecciated zone width 1.1m gosson
bx : ungulor ~ subungulor.
60E
with gosson
hily silicifid rock
(rhyolite)
80
60
55W
65E
30
85
weathered
strong argillization
S-22 119.114



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LEGEND

^ ^ ^	tuff, lapilli tuff
V V V	biotite feldspar - quartz porphyry
L L L	acidic rock (rhyolite)
^ ^ ^	brecciated zone (dyke)
—	fissure

Abbreviation

fel : feldspar

bio : biotite

limo : limonite

qtz : quartz

argi : argillization

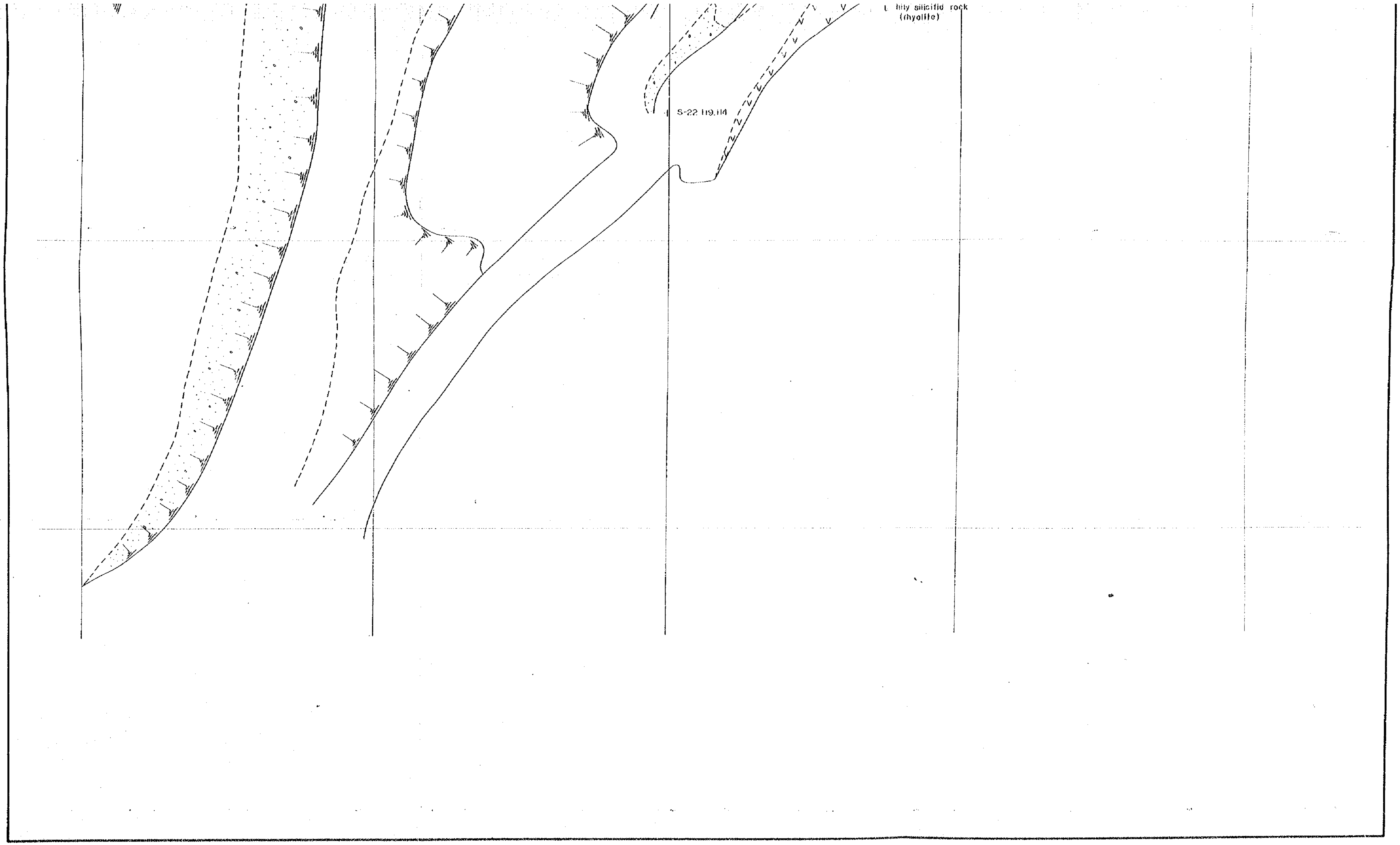
sili : silicification

hily : highly

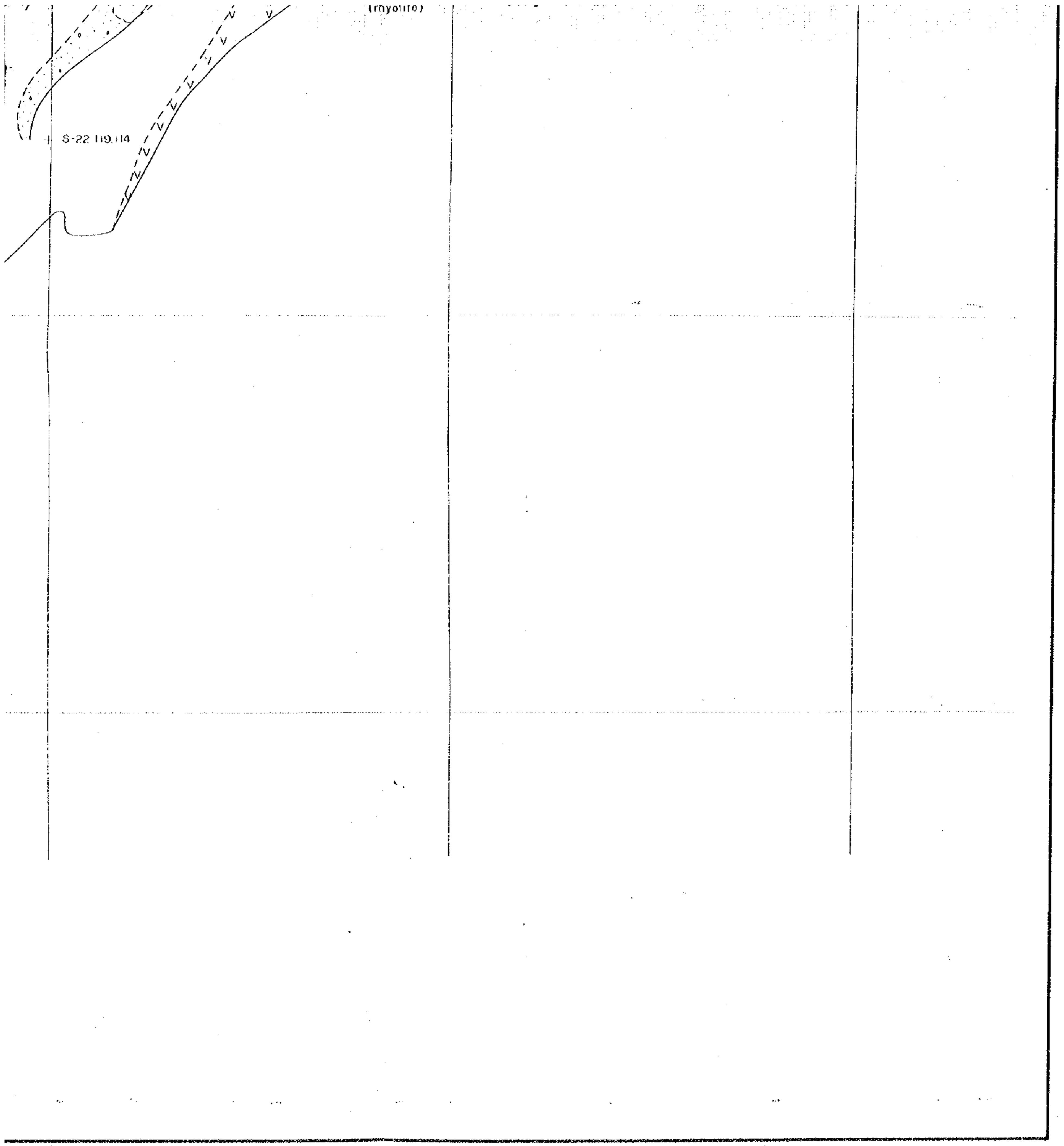
+++ strong

++ medium

+ weak



S-22 119.114
hly silicifid rock
(rhyolite)



argi : argillization

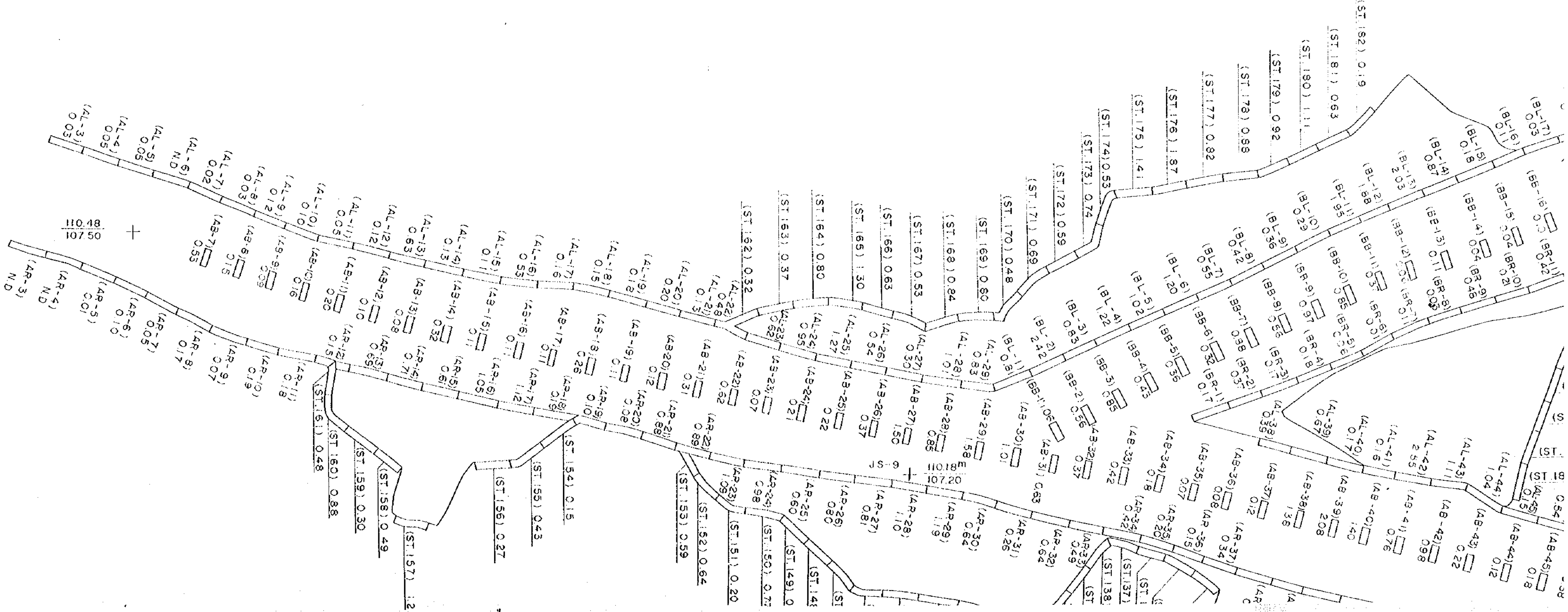
sili : silicification

hily : highly

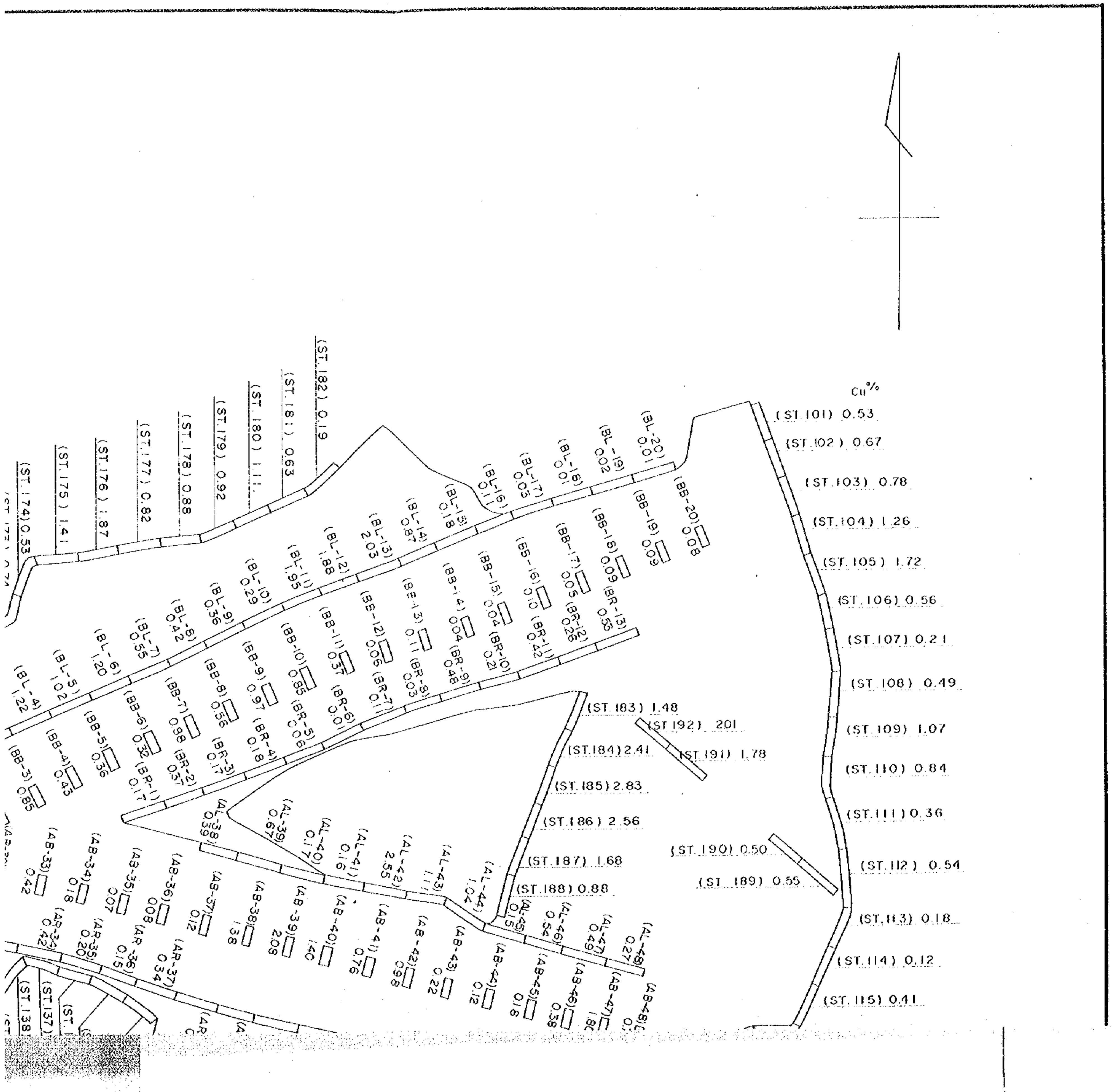
+ + + strong

+ + medium

\+ weak



110.48
107.50
JS-9
110.18
107.20



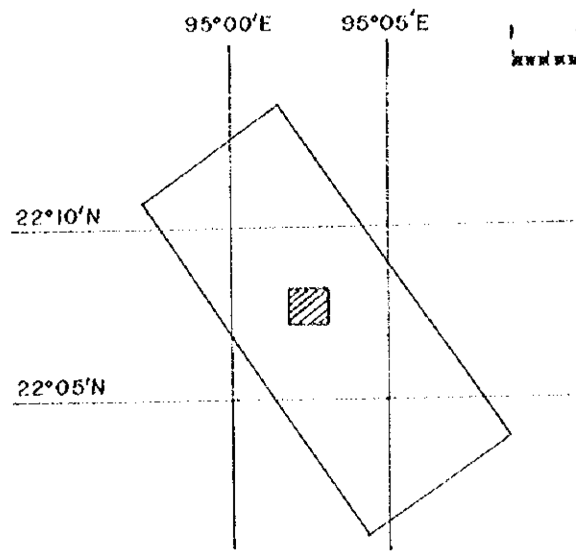
PL. I-8-2

GEOLOGICAL SURVEY OF
MONYWA AREA UNION OF BURMA
(PHASE IV)

ASSAY MAP OF SABEDAUNG TUNNEL.

Scale 1:100

0 1 2 3 4 5m



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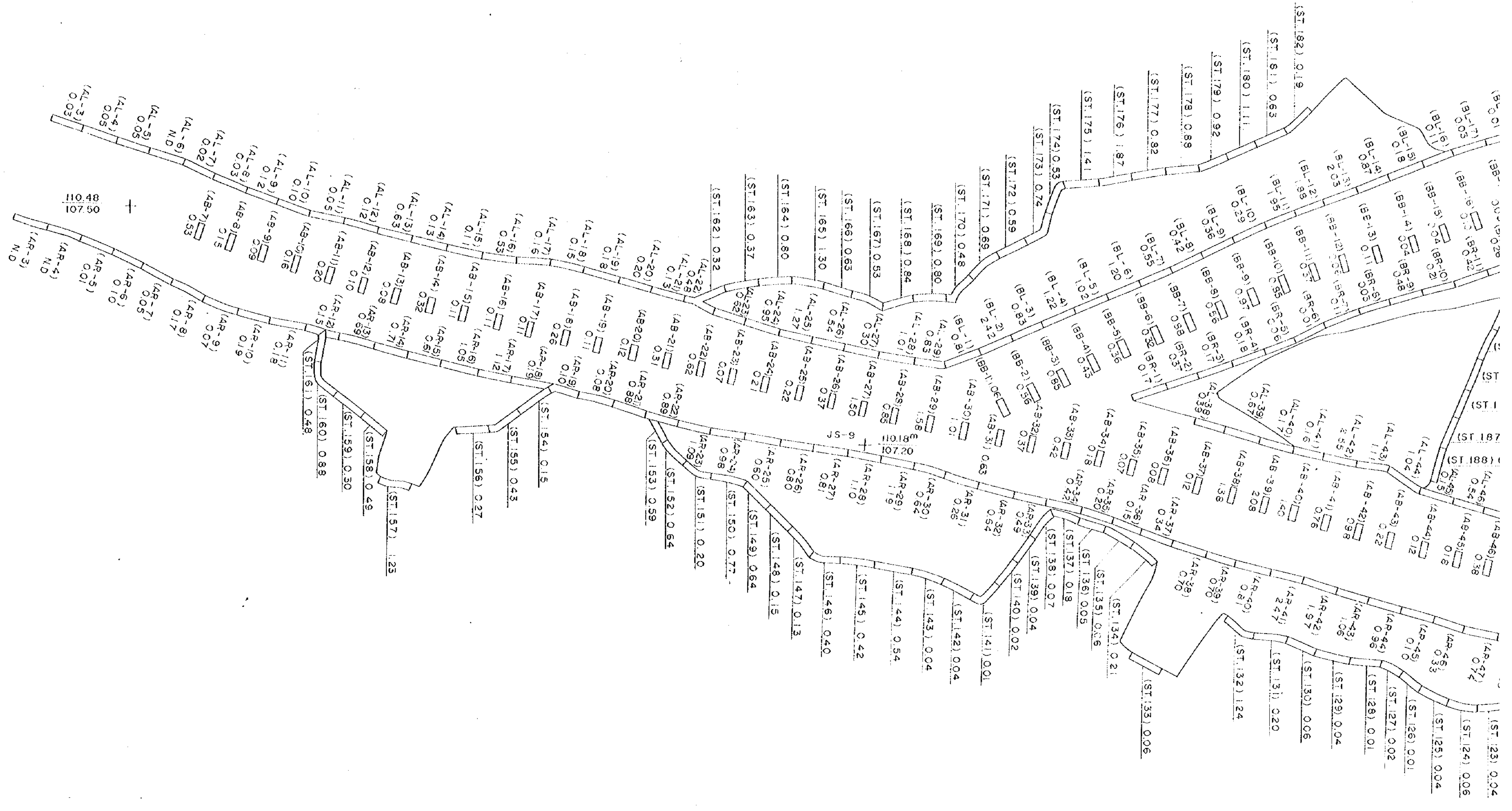
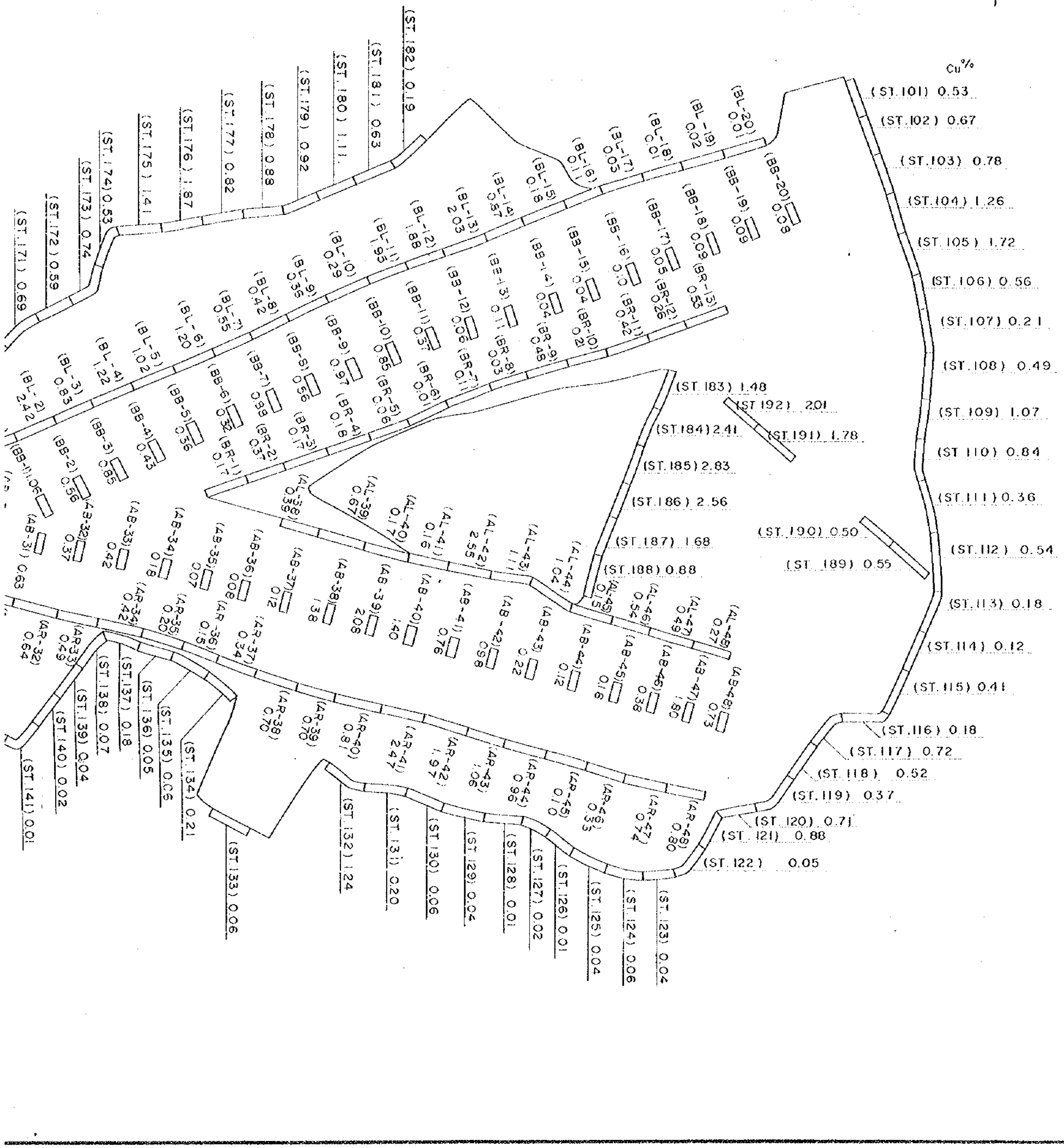
LEGEND

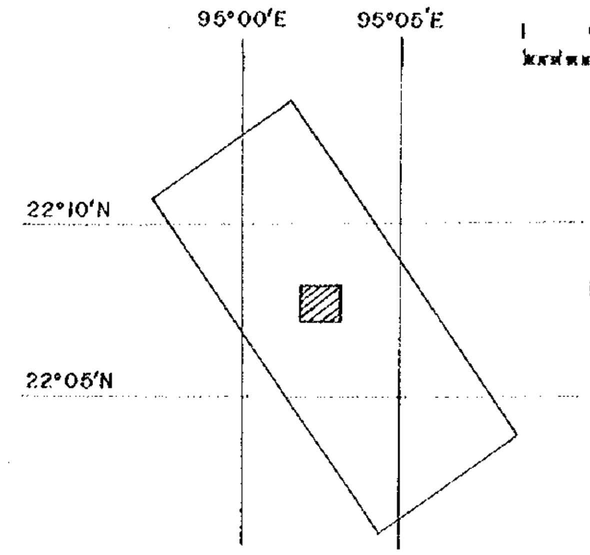

110.48
107.50
JS-9
110.18
107.20



ASSAY MAP OF SABEDAUNG TUNNEL.

Scale 1: 100

1 0 1 2 3 4 5m



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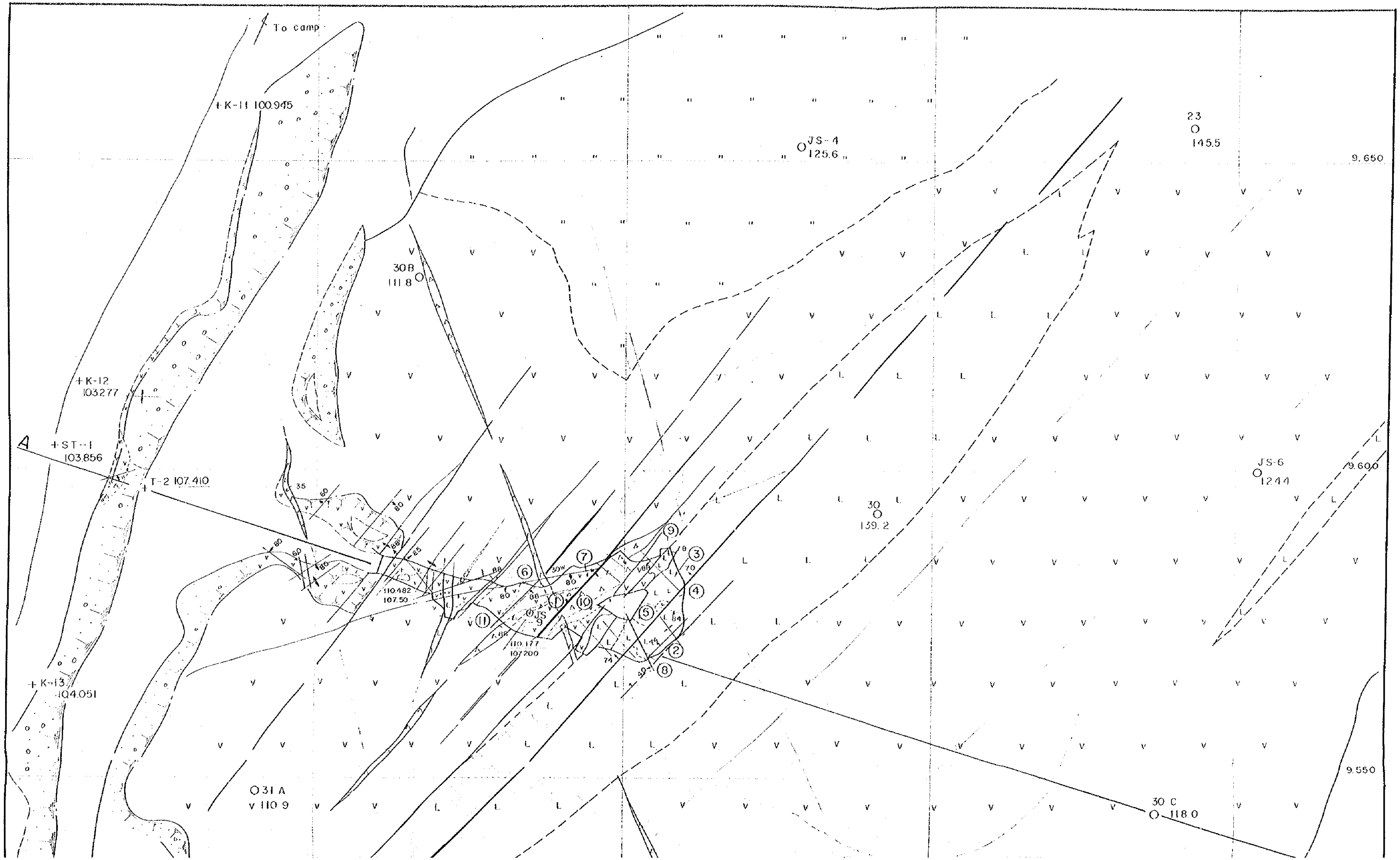
LEGEND

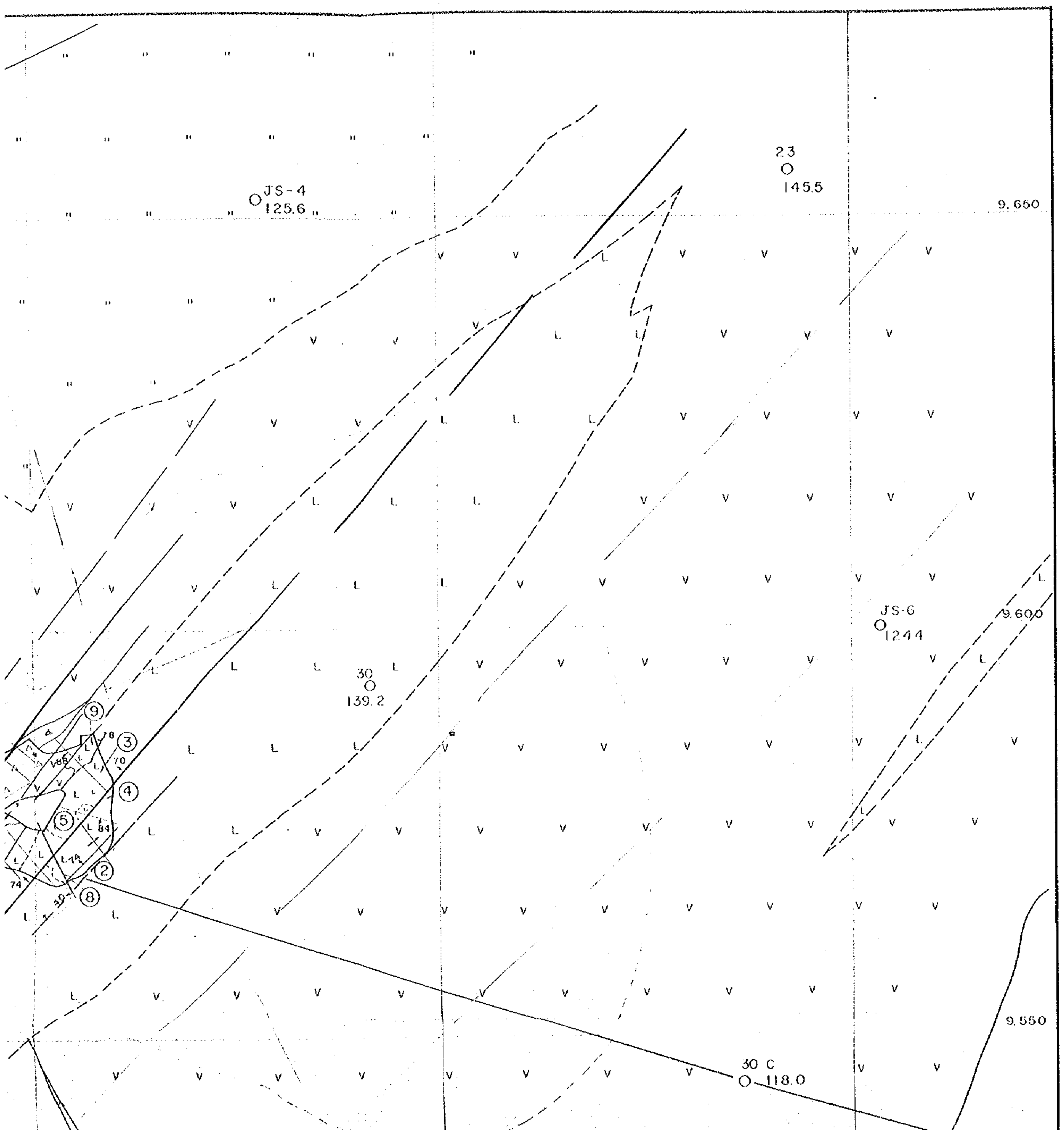
Sample locality (Wall and roof)

(ST.110) 0.84 Sample NO. and total Cu grade %

+ $\frac{110.48}{107.50}$ Survey point (Elevation of roof/ floor)



METAL
JAPAN
GOVER
22°10
22°05
Prepared

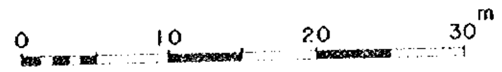


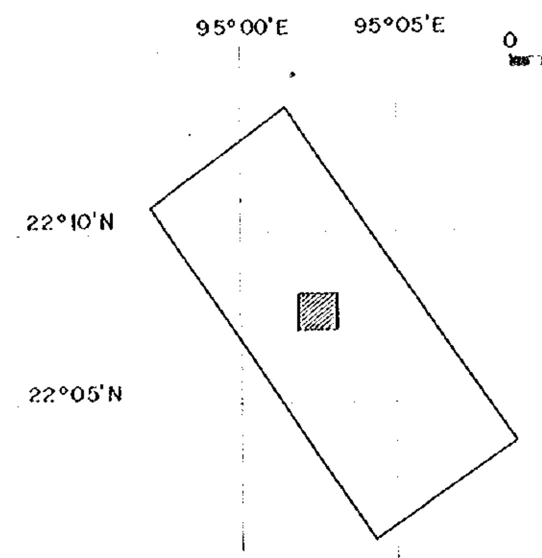
PL. I-8-1

GEOLOGICAL SURVEY OF
MONYWA AREA UNION OF BURMA
(PHASE IV)

GEOLOGICAL MAP OF SABEDAUNG TUNNEL

Scale 1:500





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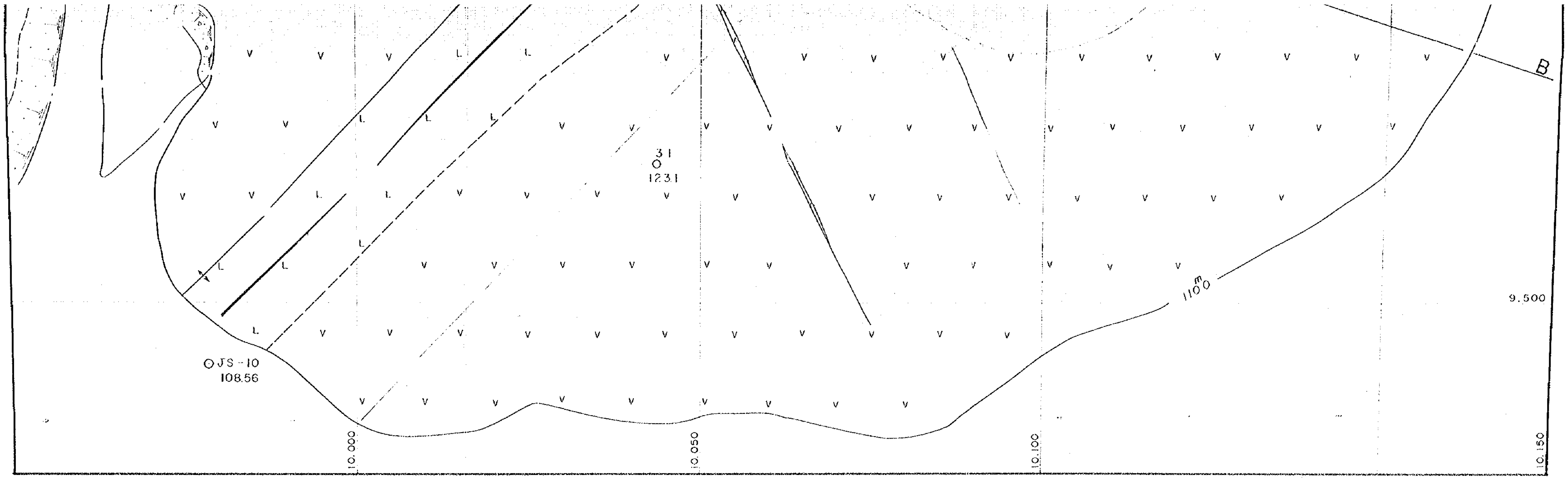
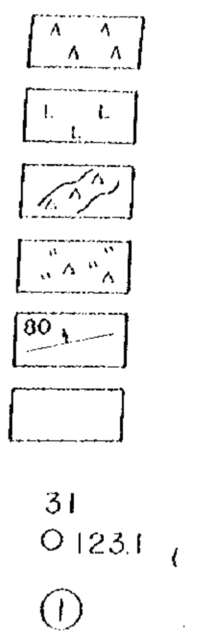
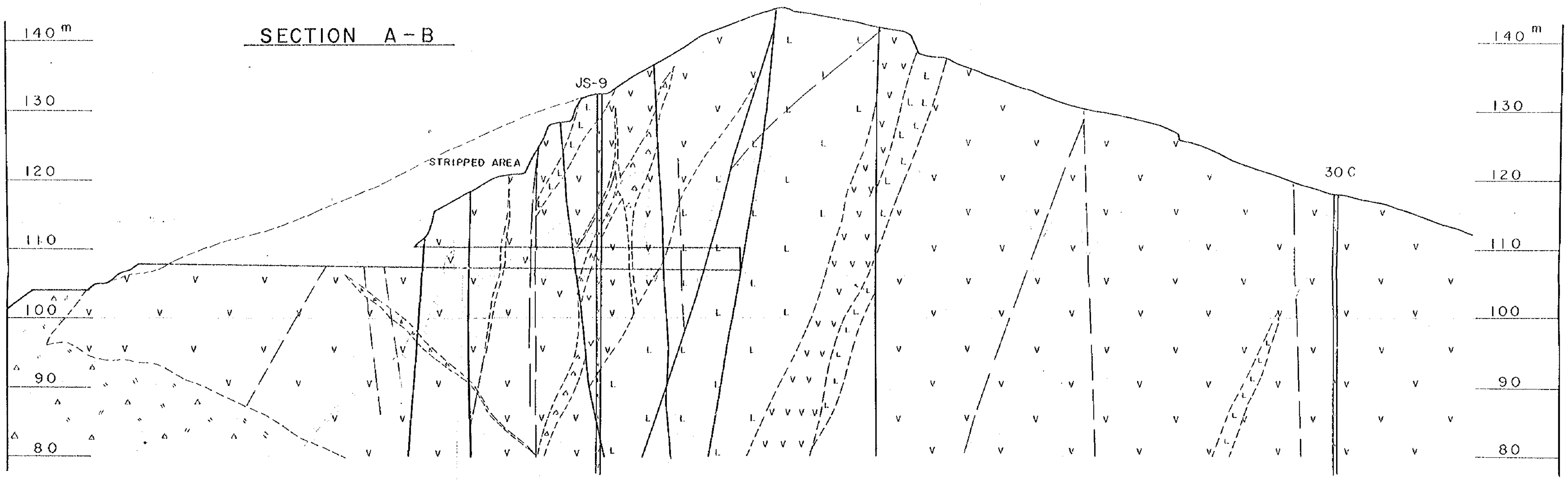
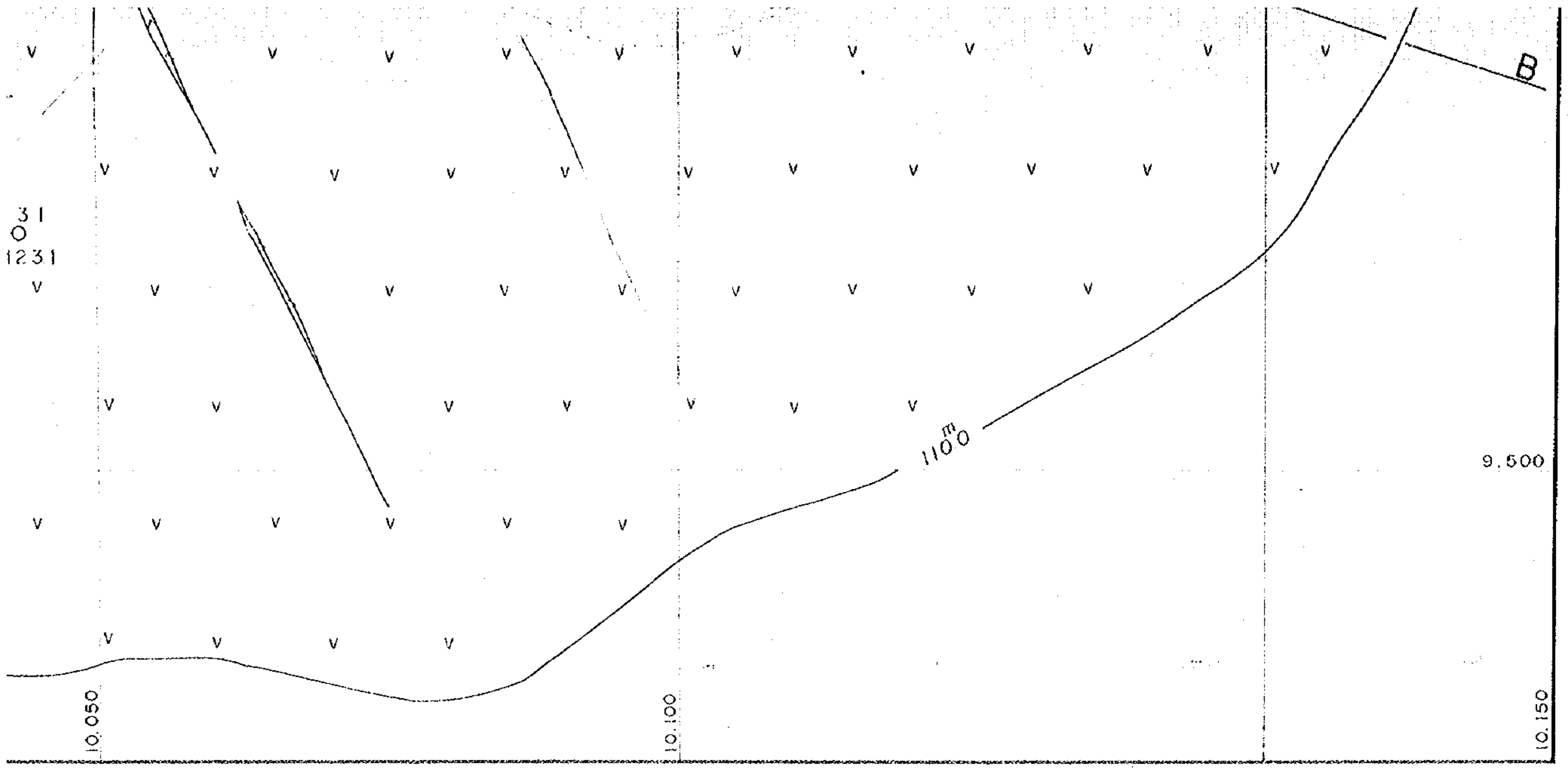
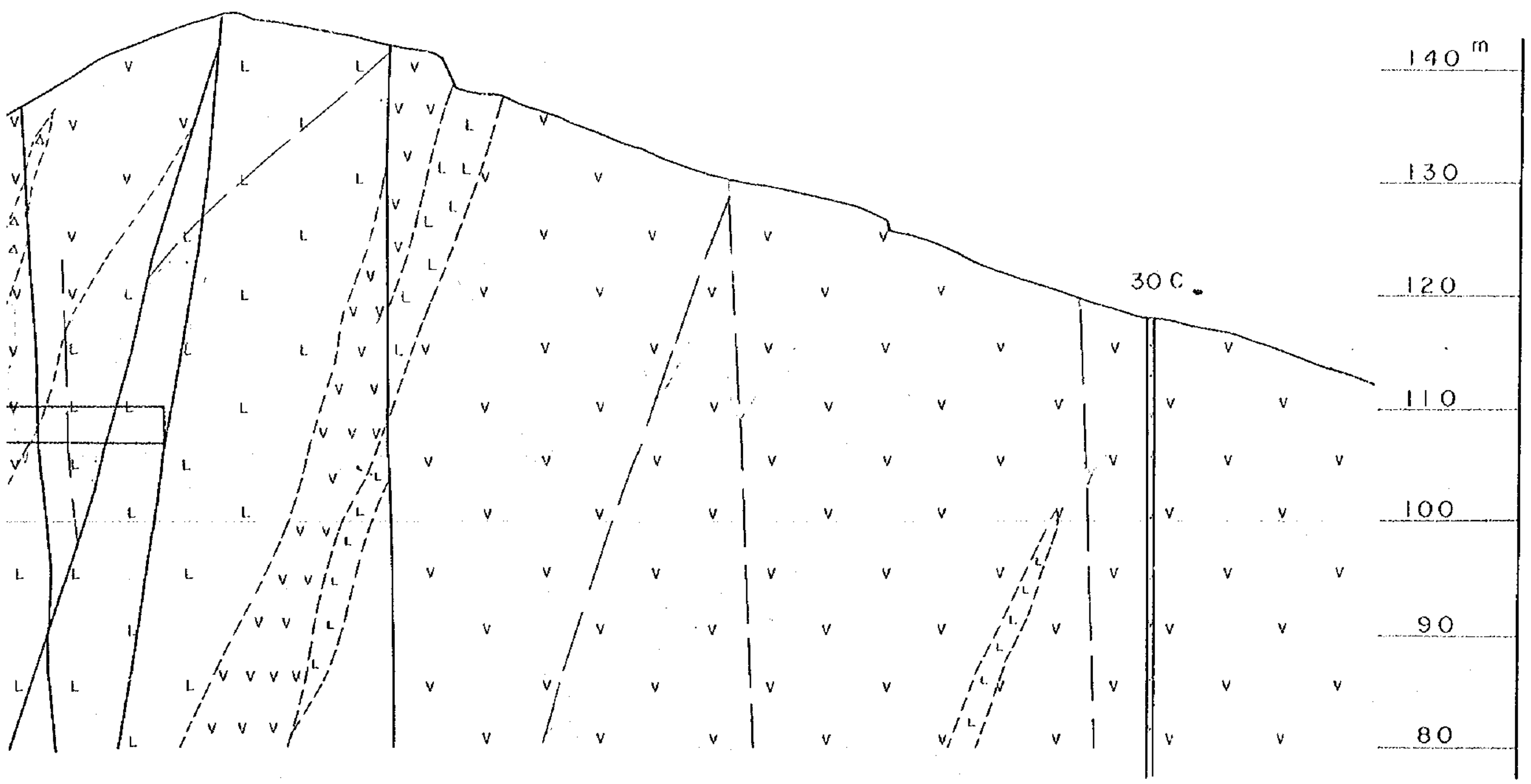
LEGEND


B
31
123.1
JS-10
108.56
1100m
9.500
10.000
10.050
10.100
10.150


80
31
O 123.1
(1)


SECTION A-B
140 m
130
120
110
100
90
80
JS-9
STRIPPED AREA
30 C





LEGEND

	Biotite porphyry
	Altered rhyolite
	Brecciation
	Tuff
80	Fissure & Joint
	Ore body
31 123.1	Drilling hole (D.D.H NO. & Collar elevation)
①	Sample NO.