REPUBLIC OF UGANDA
ENGINEERING STUDY
FOR
THE DEVELOPMENT
OF
THE KILEMBE MINE AND JINJA SMELTER

JU[illegible] 1978

JAPAN INTERNATIONAL COOPERATION AGENCY

REPUBLIC OF UGANDA

ENGINEERING STUDY

FOR

THE DEVELOPMENT OF KILEMBE MINE AND JINJA SMELTER

AUGUST 1978

JAPAN INTERNATIONAL COOPERATION AGENCY

Preface

The Government of Japan, in response to a request of the Government of the Republic of Uganda, agreed to conduct a feasibility study on the rehabilitation of the Kilembe Mine and the Jinja Smelter and entrusted the study to the Japan International Cooperation Agency (JICA).

The Agency, in consideration of the importance of the rehabilitation project of the mine and the smelter in Uganda, dispatched a survey team, headed by Mr. yoichi Hirata, to Uganda for a period of forty (40) days from January 29 to March 9, 1978.

The team, which consisted of two groups, one for the mine at Kilembe and another for the smelter at Jinja, carried out field investigation with the cooperation of the Governmental Agencies concerned of the Republic of Uganda. On returning to Japan, the team examined the findings of the survey and the data obtained in Uganda, and prepared this report.

I hope this report will contribute to the economic and social development of the Republic of Uganda as well as to the promotion of friendship between our two countries.

I wish to express my sincere thanks to the Government of Uganda and officials concerned for their cooperation extended to the survey team.

August, 1978

Shinsaku Hogen

President
Japan International Cooperation Agency.

Letter of Transmittal

Mr. Shinsaku Hogen
President
Japan International Cooperation Agency

Dear Sir:

Submitted herewith is the engineering study report for the development of the Kilembe Mine and the Jinja Smelter in the Republic of Uganda.

The survey team, organized by Japan International Cooperation Agency, consisted of 10 experts, which are eight engineers from Sumitomo Metal Mining Co. Ltd. and Furukawa Co. Ltd., a co-ordinater from Japan international Cooperation Agency and a geological engineer from the representative office of Metal Mining Agency of Japan, Nairobi, Kenya. The survey team, which was despatched to Uganda on January 29th, 1978, and stayed in the country for 40 or 24 days, examined all the facilities of the Kilembe Mine and the Jinja Smelter, and collected the information concerned.

The main subjects of the study are as follows;

(1) Examination of the current situation of the Mine and the Smelter.

(2) Examination of the approach and the schedule for resuming the presently suspended production at a normal rate.

(3) Estimation of the cost for resuming the production and operating.

(4) Economical evaluation of the Mine and the Smelter when resumed.

Upon leaving Uganda, the survey team had presented an interim report to the Minter of Power and Industry, and to the Chairman of the Uganda Development Corporation, the Government of Uganda.

This report was prepared by the members of the survey team with aids of other experts.

In this report, the subjects above mentioned are discussed on the basis of the information in regard to the ore deposits, the underground the mill, and the smelter operations, and the relevant facilities. The production rate of 50,000 tons per month seems to be appropriate for resuming the production. It is indispensable for resuming production to replace or repair a number of facilities, to improve a part of the mineral dressing process and to completely remodel the present smelting plant.

The economical aspect of the Mine and the Smelter seems to be discouraging in general. However, it may contribute to gaining foreign currency to some extent.

Togather with rehabilitating the Kilembe Mine and Jinja Smelter, it is worth-while to carry out a metallurgical research for recoverying Cobalt from pyrite concentrate, which may substantially contribute to the profitability if it is successfully achieved.

A re-exploration of the west Bukangama prospect, which is located to the vicinity of the present mining area, may be beneficial to increase the ore reserve.

I would like to express my sincer gratitude to officials of the Government of Uganda and to staffs of Kilembe Mines Ltd. for their willing and unsparing co-operation to the survey team.

I am also indebted to cordial co-operation and effective assistances extended by officials of the Embassy of Japan and other agencies of the Japanese Government in Nairobi, Kenya.

Yours very respectfully,

Yoichi Hirata
Leader
Survey Team for the Development of
The Kilembe Mine and Jinja Smelter,
The Repoblic of Uganda.

Table of Content

Part-3 Economic Evaluation

List of Drawings

List of Figures

Part-1

List of Tables

Part-3

Appendices

Appendix-1 List of Ore Reserve

Appendix-2 Present Conditions of Machines and Equipment (Kilembe Mine)

Appendix-3 Price Estimations of Machines, Equipment and their spare parts (Kilembe Mine)

Appendix-4 Basic Data for Operating Cost Estimation (Kilembe Mine)

Appendix-5 Mineral Dressing Test on Ore from Kilembe Copper Mine.

Introduction

The Kilembe Mine and Jinja Smelter, owned by Kilembe Mines Ltd. were taken over by Uganda Development Corporation, the Government of Uganda, from the former management, Falconbridge Nickel Mines Ltd., in early 1975. These Mine and Smelter have been suffering serious difficulties in their operations in recent years, due to super annuation of their facilities, lack of necessary supplies, the world-wide depression of the copper market and other reasons.

The Government of the Republic of Uganda, which had been concerned about this situation, requested the Japanese Government in late 1977 to make a study on the rehabilitation of the Kilembe Mine and the Jinja Smelter.

In response to the request, the Japanese Government decided to undertake an engineering study for the Kilembe Mine and the Jinja Smelter and entrusted its execution to Japan International Cooperation Agency, who organized a technical survey team.

The principal purposes of the study were to investigate the current problems of the operation both at the Mine and the Smleter, to find suitable approach for resuming the production in a normal rate, and to study the economical aspect of the rehabilitation.

The survey team was despatched to Uganda in Late January,1978. The members of the survey team were as follows.

Leader	Yoich Hirata	Manager, The Exploration Department. Sumitomo Metal Mining Co. Ltd.
Sub Leader	Kenzo Murao	Manager, the Overseas Department. Furukawa Co. Ltd.
	Jun Okamoto	Senior Mining Engineer Sumitomo Metal Mining Co. Ltd.
	Katsuhiro Otani	Geologist, Sumitomo Metal Mining Co. Ltd.
	Takahisa Ikunaga	Metallurgist, Sumitomo Metal Mining Co. Ltd.
	Takashi Terashima	Mechanical Engineer, Sumitomo Metal Mining Co. Ltd.
	Hiroharu Kano	Senior Mechanical Engineer Furukawa Co. Ltd.
	Yoshirisa Yamamoto	Electrical Engineer. Furukawa Co. Ltd.
	Toshio Sakasegawa	Geologist, Metal Mining Agency of Japan, Nairobi.
	Masahiro Yamamoto	Japan International Co operation Agency

The survey team was devided into two groups; the one for the mine at Kilembe, the other for the smelter at Jinja.

The former group stayed at Kilembe for 20 days between 8th and 27th of February, and examined the operation, the facilities and the equipment at the mine to collect the necessary information for the study.

The latter group stayed at Jinja for 17 days between 7th and 23rd of February, and did the same for the smelter.

Upon return of the survey team to Japan, this report was prepared after having assessed and analized all the information obtained at the mine and the smelter.

This report comprises three parts; the part-I deals with the mining-milling operation at Kilembe, and the related facilities and equipments, the Part-II with the Smelting plant and operation at Jinja, and part-III with the economic evaluation.

Acknowledgement

We are honoured to express our greatest appreciation to His Excellency, Brigadier General Dusman Sabni and His Excellency, Lieutenant colonel, William Nhahendekere, the Chairman of the Uganda Development Corporation, whose effective co-operations enabled us to accomplish our work at the Kilembe Mine and the Jinja Smelter.

We would like to acknowledge Captain S. Gala, the General Manager of Kilembe Mines Ltd., Mr.W.J. Williams, the Assitant General Manager, and the staffs of Kilembe Mines Ltd, for their co-operation to our activity at the sites.

We are also indebted to Mr.Y.Kashiwada, the Manager of the Uganda Garment Industry Ltd., whose adequate arrangements were greatly benificial to us during our stay in Uganda.

We are greatful to the officials of the Embassy of Japan, the Japan International Cooperation Agency, the Overseas Economical Cooperation Fund, and to the staff members of C Itoh and Company, in Nairobi, Kenya.,for their cordial assistances extended to us.

Finally, we would like to express our appreciation to International Cooperation Agency, Tokyo, Japan, for its effort to have organized the survey team.

Summary

Upon the request of the Government of Uganda, a Japanese survey team was sent by the Japanese Government to investigate the Kilembe Mine and the Jinja Smelter owned by Kilembe Mines Ltd., the Republic of Uganda, and to find a suitable approach for resuming the presently suspended production in a normal rate.

The Kilembe Mines Ltd. formerly managed by the Falconbridge Nickel Mines Ltd, Canada, was taken over in early 1975 by the Uganda Development Corporation, the Ministry of Power and Industry, the Government of Uganda.

The Kilembe Mine, situated in the extreme southwest of Uganda, had been operated for more than 20 years until it suspended the production in late 1977 due to lack of supplies, super-anuation of its facilities. machines and equipments, the world-wide depression of the copper market and other reasons.

The Kilembe Mine consists of eight ore deposits, namely the Northern, the Stream, the Eastern, the Buhunga, the Numhuga, and the Upper, the Middle, and the Lower Bukangama Deposits, all of which occur in specific horizons of the Kilembe Schist Series of the Toro System, Precambrian in age. These ore deposits, of the strata-bound sulphide deposit category, contain a significant amount of copper (2%±) as well as an appreciable amount of cobalt (0.15%±).

The present ore reserve of the proven and probable categories, which was estimated by Kilembe Ltd. as of Dec.31,1977 and re-estimated partly in this report for the Upper, the Middle and the Lower Bukangama Deposits, is amount to 4,120,110 tons with as average grade of 1.69% Cu, or to 2,872,500 tons with an average grade of 1.94% Cu for the cut-off grade of 0.90% Cu or 1.30% Cu respectively.

The Kilembe Schist Series, the host series for the ore deposits, has been extensively explored on the surface by the Falconbridge Nickel Mines Ltd. during the period between the late 1950S and the late 1960S. According to the exploration results, there appears to be a very little possibility to newly discover such a sizable ore deposit as the Eastern Deposit. However, it will be worth-while to explore the West Bukangama prospect more in detail, because even a medium size ore deposit may be exploitable owing to its close proximity with the present mining area. The charged potential method, a geophysical exploration technique, may be suitable for exploring those ore deposits which have no evident indications on the surface.

The underground exploration may prove some additional reserves to the above amount in the vicinity of the known ore deposits, though a little possibility is seen to find a new ore deposit since the possible ore horizons have been extensively explored by diamond drilling in the underground.

The underground operation has been ceased since late 1977, except for the maintance and the development work in parts.

The mining method, the cut-and-fill stoping, has been precticed recently in the Kilembe Mine, except for the shrinkage method which has been partly applied to the steep-dipping portions of the ore deposits.

The cut-and-fill stoping is rather expensive in comparison with other mining nethods. However, this method appears to be only applicable in the Kilembe Mine, owing to the nature of the ore deposits.

The underground structures have been maintaind in appreciable conditions and is available for resuming the production.

Many of the underground machines and equipments are in poor or inoperative conditions, and necessary to be repaired or replaced if the production be resumed at a rate of 50,000 tons a month.

Spare parts for the machines and equipments, and materials such as explosives are apparently short of supply to continue the operation.

Of all the machines and equipments, the underground pumps, which drain water from the lower levels than the 4300 level, are in critical conditions and required to be repaired or replaced as soon as possible.

The milling operation seems to have been practiced with satisfactory results until the production was suspended in Aug.1977.

Only maintainance work is presently carried on statisfactorily in general, although special cares should be still taken to prevent the machines and equipments from corrosion during the period of the suspension of the production.

Many of the machines and equipments are in poor conditions or inoperative, and required to be repaired or replaced. Spare parts for the machines and equipment, and consumables such as balls, rods and reagents, are necessary to be supplied in adequate amounts for resuming the production.

There are many unserviceable heavy equipments and motor vehicles, which cause difficulity in such an ancillary facility as the Nkombe saw mill. A sufficient number of heavy equipments and motor vehicles are necessary to be reinforced for maintaining the steady operation.

The capital expenditure is estimated at U.Sh. 53,116,000 or US.$ 6,696,000 to restore the necessary machines, equipments, their spare parts and materials for resuming the production, on the basis of the quatations obtained from Japanese suppliers and manufacturers. In addition to the above amount, a sum of U.Sh. 14,734,000 or US.$ 1,858,000 will be required for the 6 months inventry to carry on the production.

The exchange rate is herewith quoted at 7.93 Uganda Shilling or 224 Japanese Yen for one U.S.dollar, as of May 10th, 1978.

The Jinja smelter is located approximately 80 km east of Kampala, the Capital of Uganda, and is connected with the Kilembe Mine by a railway for a distance of some 500 km.

The smelting plant is, as a whole, in extremely poor conditions. Many of facilities and equipment are so obsolete that their spare parts are hardly available on the market. It is indispensable to redesign or to replace the charging device, the electrode device and the gas handling equipment. It is also recommended that the vital components of the electric furnace be completely replaced.

The operation routines, which are practiced in the Smelter, appear to be inadequate in handling ore and product. Re-training of workers will be necessary for an expedient operation.

It may be inevitable to have a sufficient stock of spare parts and other supplies for at least one year, because the landlocked nature of the country makes it difficult to obtain necessities abroad within a reasonable duration.

A sum of U.Sh. 34,341,000 or US.$ 4,330,000 will be required to remodel the smelting plant matching with the ore production rate of 50,000 tons per month at the Kilembe Mine. In addition,a working capital of U.Sh. 9,841,000 or US.$ 1,241,000 will be needed to hold one year inventry during the production.

In summary, the total capital expenditure will amount to U.Sh 112,032,000 or US.$ 14,125,000 to resume the production.

It will take about 14 months after making a go-ahead decision to prepare for resuming the production, including delivery, transportation, installation and idling times.

The total monthly operating cost for the Mine and the Smelter is estimated at U.Sh 7,502,000 or US.$ 946,000 to carry on the production at a rate of 50,000 tons per month.

A cash flow analysis of the production scheme at a rate of 50,000 ton per month was made for the case of the ore reserve of 2,872,500 tons at a grade of 1.94% Cu, by assuming the fixed copper price at UK.£ 694.5 as of May 10^{th}, 1978 at the London Market during the period of the production. The exchange rate is quoted at US.$ 1.795 for UK.£ 1, as of May 10^{th},1978.

The result indicates that the annual loss of U.Sh 13,856,000 or US.$ 1,746,000 will be accumulated to U.Sh 71,381,000 or US.$ 8,994,000 by the end of the production. As to the balance of foreign currency, annual surplus of US.$ 5,291,000 will recover the initial capital expenditure in about three years.

Provided that a copper price of UK.£ 960 be attained, the capital expenditure necessary for resuming the production will be written off at the time of exhausting the given ore reserve.

The same study was made for the case of the ore reserve of 4,120,110 tons at a grade of 1.69%. The mine life is prolong to 6.8 years from 4.8 years for the above case. However, the profitability will become worse than the above case. 16

In conclusion, the rehabilitation of the Kilembe Mine and Jinja Smelter seems to be very marginal from an economical point of view, though it may contribute to gaining a certain amount of foreign currency.

An intensive financial aid by the Government may be indipensable to keep the steady production.

It was also taken into account in this study to construct a new smelter at a site in proximity with the Kilembe Mine. However, the construction of a new smelter will cost a substantial amount of capital investment which will be hardly allowed for the scale of the Kilembe Mine.

Cobalt is one of the important elements contained in the ore of the Kilembe Mine. However, the commercial recovery of cobalt has never been realized in spite of exhausting researches done by the previous management. It is indispensable to carry out an extensive research work, including a metallurgical study and a market research for cobalt and sulferic acid.

The following are recommendations resulted from this study.

1) An immediate action should be taken for resuming the production at any rate by acquiring necessary supplies, machines and equipment, because it will become more and more difficult to maintain the underground in a working condition if the present situation be kept going. For this purpose, a financial arrangement is necessary to be made as soon as possible.

2) In this report, the full requirement for the production at a rate of 50,000 ton per month was considered. However, it is also possible to resume the production at a lower rate with a smaller amount of the capital expenditure than allocated in this report as far as the mining and milling operation concerns, although the operation would become less profitable. A list of a minimum requirement should be prepared by the staff of the Kilembe Mine.

3) A complete remodelling of the Jinja smelter is indispensable to resume the production of blister with a satisfactory efficinecy, unless otherwise sale of concentrate be taken into account.

Training of the key staffs may also be needed for effective smelting operation.

4) In progress of the operation, the productivity per person should be improved, which will gradually reduce the number of employee and the operating cost to some extent.

5) Cobalt is an important element which may contribute subtantially to the improvement of the profitability. It is worthwhile to carry out a metallurgical research for recoverying cobalt from pyrite concentrate.

In regard to recovery of cobalt, a minor modification is necessary for recovering pyrite as much as possible, 30 to 50% of which is being disposed as tailings in the present milling process.

6) The west Bukangama prospect is worthwhile to be re-explored, because it is located close to the present mining area. A charged potential method, followed by diamond drilling, may be recommended to locate blind ore bodies, if any.

PART-1 KILEMBE MINE

1-1 General Description

1-1-1 Location and Access (Fig.-1)

The Kilembe Mine is situated in the extreme southwest of Uganda, in the southeastern foot-hills of the Rwenzori Mountains, approximately 13 kilometers west northwest of the town of Kasese (Lat. 0°10' N, Long.30°00'E).

The Mombasa-Kampala Railway, extended to Kasese, provides a major transportation medium which carries the copper concentrate from the mine to the smelter at Jinja and necessary supplies to the mine. The distance between Kasese and Jinja is approximately 480 kilometers along the railway.

A major highway connects Kasese to Kampala, the Capital of Uganda, for a distance of approximately 450 kilometers. A paved road furnishes an access to the mine from Kasese.

Regular air services are also operated between Kampala and Kasese three to four times a week.

Accordingly, accesses to the mine area are reasonably well-developed.

The Rwenzori Mountains lies within the western branch of the Rift Valley System, and rises from the Rift Valley floor at an elevation of approximately 1,000 to nearly 5300meters above sea-level.

Showings of the ore deposits occur in steep hill-sides along the Nyalusegi River which joins with Namwamba river, a principal drainage in the Rwenzori Mountains. (Dwg. No.1).

The mine buildings and the townships of Kilembe are located along the narrow Valley of the Namwamba river (Dwg.4).

1-1-2 History and Past Production

The presence of copper mineralization in the Rwenzori Mountains was first reported in 1906 by Rocatti, a geologist of the expedition led by the Duke D'Abruzzi.

Tanganyka Concessions limited was granted a concession of about 640 square kilometers in the area and started extensive exploration in 1926. The company completed a substantial amount of surface work and underground development during the period between 1927 and 1937. The work indicated significant quantities of copper in the Northern, Stream and Eastern deposits.

Table 1. Annual Production: 1956-1977

Year	Ore Milled (M T)	Concentrate (M T)	Blister (M T)
1956	175,717	11,863	150
1957	434,826	33,940	7,467
1958	473,480	30,231	10,890
1959	628,051	32,293	12,125
1960	810,540	40,267	14,752
1961	827,605	38,099	13,378
1962	886,869	57,545	15,581
1963	896,578	61,358	16,221
1964	897,394	57,205	18,265
1965	933,775	56,441	17,146
1966	931,303	55,746	16,105
1967	864,629	52,657	14,430
1968	926,760	56,639	15,602
1969	979,762	60,141	16,568
1970	1,003,115	62,128	16,958
1971	947,627	56,010	15,731
1972	907,287	50,716	14,071
1973	821,153	49,839	9,643
1974	708,230	39,686	8,915
1975	479,213	29,215	8,277
1976	396,485	23,670	5,000
1977	157,022	9,361	2,272
Total	16,087,421	965,050	269,547

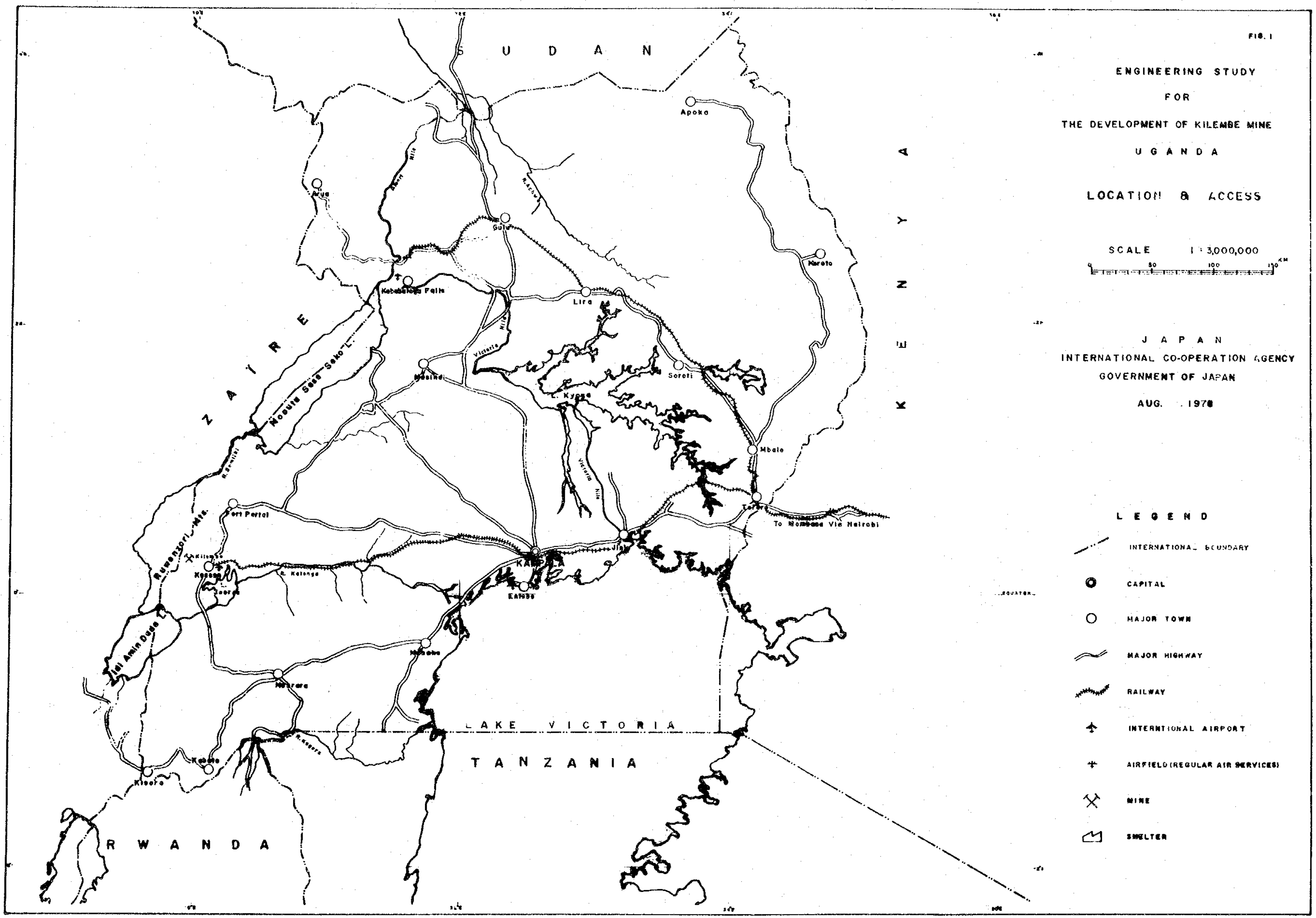


FIG. 1
ENGINEERING STUDY
FOR
THE DEVELOPMENT OF KILEMBE MINE
UGANDA
LOCATION & ACCESS
SCALE 1 : 3,000,000
0 50 100 150 KM
JAPAN
INTERNATIONAL CO-OPERATION AGENCY
GOVERNMENT OF JAPAN
AUG. 1978
LEGEND
INTERNATIONAL BOUNDARY
CAPITAL
MAJOR TOWN
MAJOR HIGHWAY
RAILWAY
INTERNTIONAL AIRPORT
AIRFIELD (REGULAR AIR SERVICES)
MINE
SMELTER
SUDAN
KENYA
ZAIRE
TANZANIA
RWANDA
LAKE VICTORIA
Apoka
Arua
Gulu
Kabalega Falls
Lira
Masindi
Soroti
L. Kyoga
Mbale
Tororo
To Mombasa Via Nairobi
Jinja
Fort Portal
Kilembe
Kasese
KAMPALA
Entebbe
Masaka
Mbarara
Kabale
Kisoro
Moroto
R. Katonga
R. Kagera
Victoria Nile
Ruwenzori Mts.
Idi Amin Dada L.
Mobutu Sese Seko L.
EQUATOR

In spite of the successful results, the concession was abandoned in 1937, probably due to the prevailing low metal price and the remoteness of the location.

In 1947, Frobisher limited was granted an exclusive prospecting licence and re-opened the T.C.L's workings. It undertook a feasibility study which included an extensive underground development, more than 30,000 meters of diamond drilling and ore treatment in pilot scale. It was decided to go ahead with exploiting the ore deposits in 1950. Kilembe Mines limited was formed and subsequently Falconbridge Nickel Mines limited took over the management of the company.

Production commenced in July,1956 at a rate of 30,000 metric tons a month on a crude ore basis, which had progressively increased to nearly 85,000 metric tons a month by 1970. The crude ore production started declining in 1971 and went down to 50,000 metric tons a month by early 1975, when the Government of Uganda (Uganda Development Corporation) took over the management. Since then, the monthly production rate fluctuated between 20,000 and 40,000 metric tons in general until the production was finally suspended in August, 1977.

In the history of the production, the blister output dropped suddenly to an extremely low level for the crude ore production in 1973 and has unrecovered except for the year 1975. This was presumably caused mainly by undesirable conditions in the smelter plant.

The annual production record is summerized in Table 1.

1-2 Outline of Geology and Ore Deposits

1-2-1 Lithology and Stratigraphy (Dwg. No.1)

The mine area is entirely underlain by metamorphic rocks of the Toro System of Pre-cambrian age, dated at about 1800 million years, although Plio-Pleistocene to recent deposits are also developed along major drainage systems and on the Rift Valley floor in the vicinity.

A series of the metamorphic rocks, in which the ore deposits occur, is termed the Kilembe schist series. The Kilembe schist series is conformably overlain by the Ruahunga Gneiss and underlain by the Katiri Gneiss. Its extention has been traced from east of the mine to the Zaire border for about 45 kilometers with variable thicknesses between 2000 and 5000 feet.

The Kilembe schist series is devided into three groups, the Upper(UK), the Middle (MK), and the Lower (LK), each of which is further subdevided into two to three formations.

Stratigraphy and lithology of the metamorphic system are briefly described in Table-2.

The LK-2 formation is excluded from the Kilembe schist series in the generalized geologic map (Dwg. No.1) to clearly indicate distribution of the host rocks for the ore deposits, because the thickness of LK-2 becomes very large and the boundary between LK-2 and the underlying Katiri Gneiss has not been indicated in any geologic maps previously prepared.

The metamorphic rocks are intruded by numerous basic dikes, which are believed to be equivalent to the Karroo dolerite of Mesozoic age.

1-2-2 Structure

a) Bedding and Schistosity

The Kilembe schist series trends generally ENE-WSW and dips 20° to 60° to SSE, although strikes and dips of bedding vary notably from place to place.

Schistosity is well-developed through the Kilembe schist series,and essentially parallel to bedding. Strikes and dips of schistosity planes were measured in the cross-cuts and drifts at the 5200 and 4900 levels in the western parts of the Bukangama Deposits, and plotted on a stereographic Schmidt net (Fig. 2).

Table-2 Stratigraphy and lithology

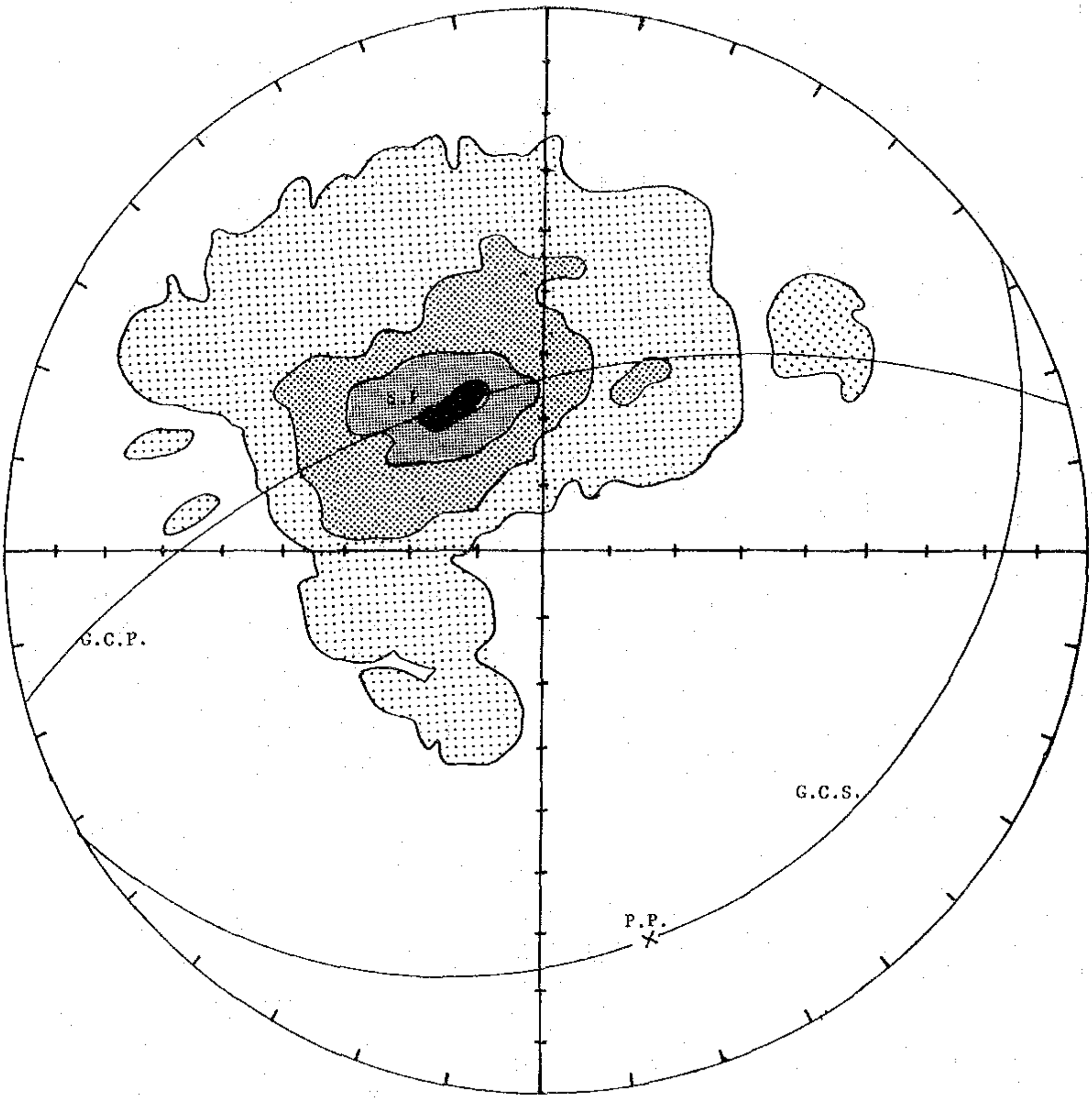
Era	Series	Group	Formation	Thickness	Columar Section	Lithology
Ceno-zoic						Glacial Drift, Lucostrine Deposits, Alluvial, Volanic Rocks, etc.
Precambrian (1800 x 10^6 yrs.)	Ruahunga Gneiss			Unknown		Potashic Gneiss; Orthoclase, Plagioclase, Microcline, Biotite Muscovite, Amphibole,
	Kilembe Schist	Upper Kilembe	UK-3	100∿400'		Highly Faliated Schist; Quartz, Sericite, Chlorite, Biotite
			UK-4	100∿1000'	UK-4 4A 4C	UK-4 Proper : Foliated Grit; Quartz, Albite, Chlorite, Biotite UK-4 A : Quartzite UK-4 C : Grit; Plagioclase, Quartz
			UK-5	20∿300'	5A 5B 5C 5D	UK-5A : Foliated Green Schist; Quartz, Albite, Chlorite UK-5B : Bonded Green Schist; Albite, Chlorite UK-5C : Porphyroblastic Green Schist; Quartz, Albite, Chlorite UK-5D : Amphibolite; Albite,Quartz,Amphibole, Biotite (Hanging Wall Ore Zone)
		Middle Kilembe	MK-1	0∿300'	MK-1	Amphibolite; Quartz,Albite, Amphibole, Biotite,(Main Ore Zone)
			MK-3		MK-3	Amphibolite; Quartz, Albite, Zoisite, Thulite, Drarite, Amphibole (Foot Wall Ore Zone)
			MK-4		MK-4	Amphibolite; Quartz, Albite, Amphibole, Biotite
		Lower Kilembe	LK-1	800∿1000'		Schist; Quartz, Albite, Garnet, Biotite
			LK-2	800∿2000'(+)		Migmatite; Plagioclase, Garnet, Sillimanite
	Katiri Gneiss			Unknown		Patashic Gneiss; Orthoclase, Plagioclase, Microcline, Biotite, Muscovite

G.C.S. : Great Circle Indicating General Strike and Dip of Schistosity; N58°E29°SE

G.C.P. : Great Circle Indicating Deviation of Poles for Schistosities

S.P. : Pole for General Strike and Dip of Schistosity

P.P. : General Plunge of Folding Axes; S16E(26)



Legend

1 ~ 5%

5 ~ 10%

10 ~ 15%

15% and above

Number of Measurments : 192

Fig.2 Contour Diagram of Schistosity (Bukangawa Area)

They appear to be widely variable in correspondence with minor foldings locally developed. However, the maximum concentration of poles of schistosity planes indicates that majority of schistosity planes strike N58°E or its proximity and dip around 20° to SE in the part of the Bukangama Deposits.

b) Folding and Lineation

Two sets of open-folding systems have been recognized in the mine area; one with axes plunging to E or ESE by 5° to 30°, and the other with axes plunging to SE or SSE by 10° to 50°.

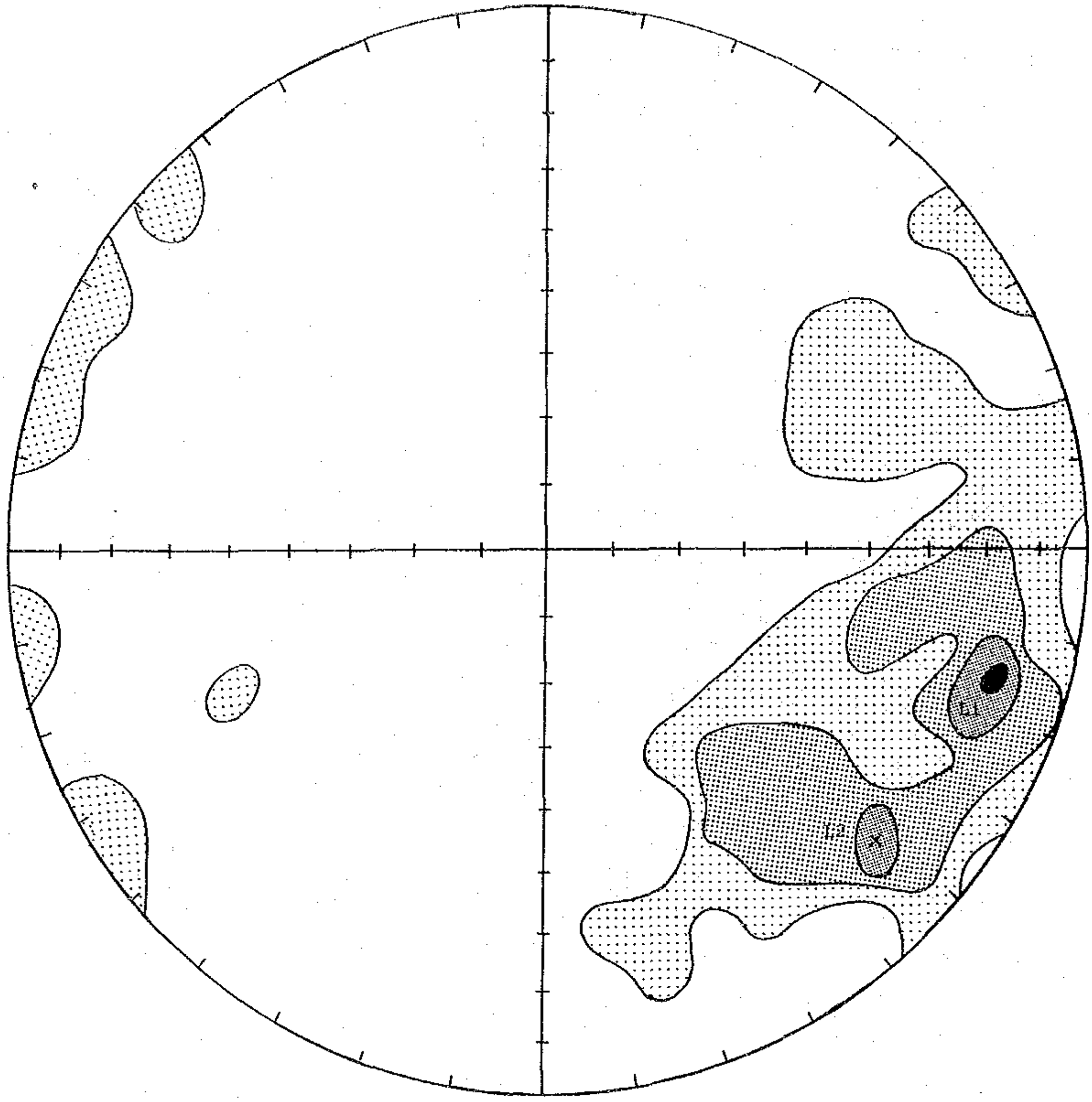
The Northern Deposit Syncline appears to be of the former type and is an exceptionally large scale folding with a wave length and an amplitude of a few thousand feet. Other minor foldings of this type are much smaller both in wave lengths and amplitudes ranging from 10 to 100 feet.

The latter type foldings are characterized by their small amplitudes in comparison with their wave length. Variation of strikes and dips of schistosity, which were measured in the Bukangama Deposits, may be resulting from foldings of this type. In Fig.2, a greater number of poles of schistosity planes appear to distribute along the great circle girdle G.C.P, the pole (P.P) of which indicates that major folding axes affecting strikes and dips of schistosity planes plunge to S16°E by 26° or to its proximity in the part of the Bukangama Deposits.

There are also observed tight isoclinal-foldings of small scales, of which amplitudes range between a few feet and a few tens feet. Their axial planes are parallel to schistosity in general, while plunges of their hinges (axes) are nearly horizontal or dip slightly to ESE.

Other than isoclinal foldings of observable scale, some geologist believe that the Toro system as a whole forms an extraordinarily large scale isoclinal folding in the area. According to them, the Katiri Gneiss is stratigraphically equivalent to the Ruahunga Gneiss and the Kilembe schist series is the core of the isoclinal folding.

Axes of microfoldings and lineations on schistosity planes were measured in cross-cuts and drifts at the 4900 and 5200 levels of the western part of the Bukangama Deposits and were plotted on a stereographic Schmidt net (Fig.3). Two centres of their concentration are observed in Fig.3. One(L-1) indicates a plunge of S74°E15°, and the other, a plunge of S49 E20°. The former seems to coincide with the plunge of some of the isoclinal folding axes afore-mentioned.



Legend

L1 : S74E(15)

L2 : S49E(20)

Number of Measurement 48

(black)	2.1 ∿ 6.3%
(dark shading)	6.3 ∿ 10.4%
(dense dots)	10.4 ∿ 14.6%
(sparse dots)	14.6% and above

Fig. 3 Contour Diagram of Lineations and Microfoldings. (Bukangama Area)

c) Fault and Dike Structures

Two major fault systems appear to be prominent in the mine area, although their strikes and dips are variable in places.

One trends NE-SW in general and dips steeply to NW or SE. The Nyamwamba and the "A" Shears are of this system. There have been reconized several faults with similar strikes and dips in the vicinity of the mine (Dwg. No.1).

The Nyamwamba Shear is one of the most prominent faults in the mine area and has dislocated the Kilembe Schist Series right-laterally by some 8000 meters.

The other fault system trends NW-SE and dips steeply to NE or SW. The Bukangama Fault and the Nyalusegi Shear are of this system. The Ruahunga shear, which trends in a similar direction, has an exceptionally gentle dip (35°-40°) to SW.

Sub-vertical faults trending nearly N-S are commonly observed in the underground but less prominent than the above two major systems.

Major basic dikes follow a direction of nearly WNW-ESE and dip subvertically to NNE or SSW, though numerous minor dikes are highly variable in their trends. Of the basic dikes, the Bukangama Dike is the most prominent one and is continuous for more than 2000 feet with an average thickness of about 200 feet.

1-2-3 Ore Deposits (Fig. 4 Dwg. No.2)

a) Occurrences

The ore deposits of the Kilembe Mine are of the strata-bound sulphide deposit category, especially of so-called the "Kieslager" or the bedded pyrite deposits type. Ore deposits of this type are well known in the metamorphic terrains of various ages in the world. Their characteristics are (1) conformability to host metamorphic rocks, (2) condiserable continuation along ore shoots in comparison with their thicknesses and strike lengths, (3) pre-domination of sulphides to gangues (mainly pyrite with a subordinate amount of pyrrhotite in general). In many cases, ore shoots are controlled by structures of early stages, and are nearly parallel to axes of isoclinal foldings or microfoldings, or to other lineations. Therefore, it would be important to observe and analize these structures for exploring ore deposits of this type.

The ore deposits occure in two different zones in the Kilembe Mine. One is the Eastern Zone originally discovered and exploited. The other is the Bukangama Zone.

The Eastern zone includes the Northern, the Stream, the Eastern, the Buhunga and the Numhuga Deposits. They appear to be a continuation of a single ore deposit which has been separated by faults and a topographic depression (the Nyalusegi River).

In the Bukangama Zone, three different units of ore deposits are recognized; namely, the Upper, the Middle, and the Lower Bukangama deposits. The Middle Bukangama Deposit is included in the Upper Bukangama Deposit in the mine. However, they are appearently independent ore shoots to each other, because there is a distinct barren zone between the two deposits.

All of the above deposit occur parallel to schistosity in stratigraphically specific horizons (UK-5D, MK-1 and MK-3) of the Kilembe Schist Series (Table 2).

Thickness of these ore deposits varies considerably from less than a inch to more than 100 feet.

The width across the ore shoots is hardly measurable in most parts of the ore deposits in the Eastern Zone, due to the complex structures. However, it is measured at approximately 3000' across the middle of the Eastern deposit, where the ore deposit is less disturbed by faults or foldings than any other parts.

The length along the ore shoots exceeds 12,000' from the west end of the stream Deposit to the Buhunga Deposit and is still continuing down-dip to east southeast.

The Bukangama ore deposits are much smaller in scale than the Stream, Eastern and Numhuga deposits combined.

The Upper Bukangama Deposit, the largest of the three, is measured at approximately 6000' in its length along the ore shoot, and ranges from 500' to 2000' in its width.

The Middle and the Lower Bukangama Deposits are similar in scale to each other. They are some 3000' long and about 300' wide in average.

Copper accumulation (thickness x Cu grade) was calculated, as a step of the ore reserve estimation, for a ore intersection of each drill hole in the Bukangama Deposits (Fig.6). The results are shown in Dwg. No.2.

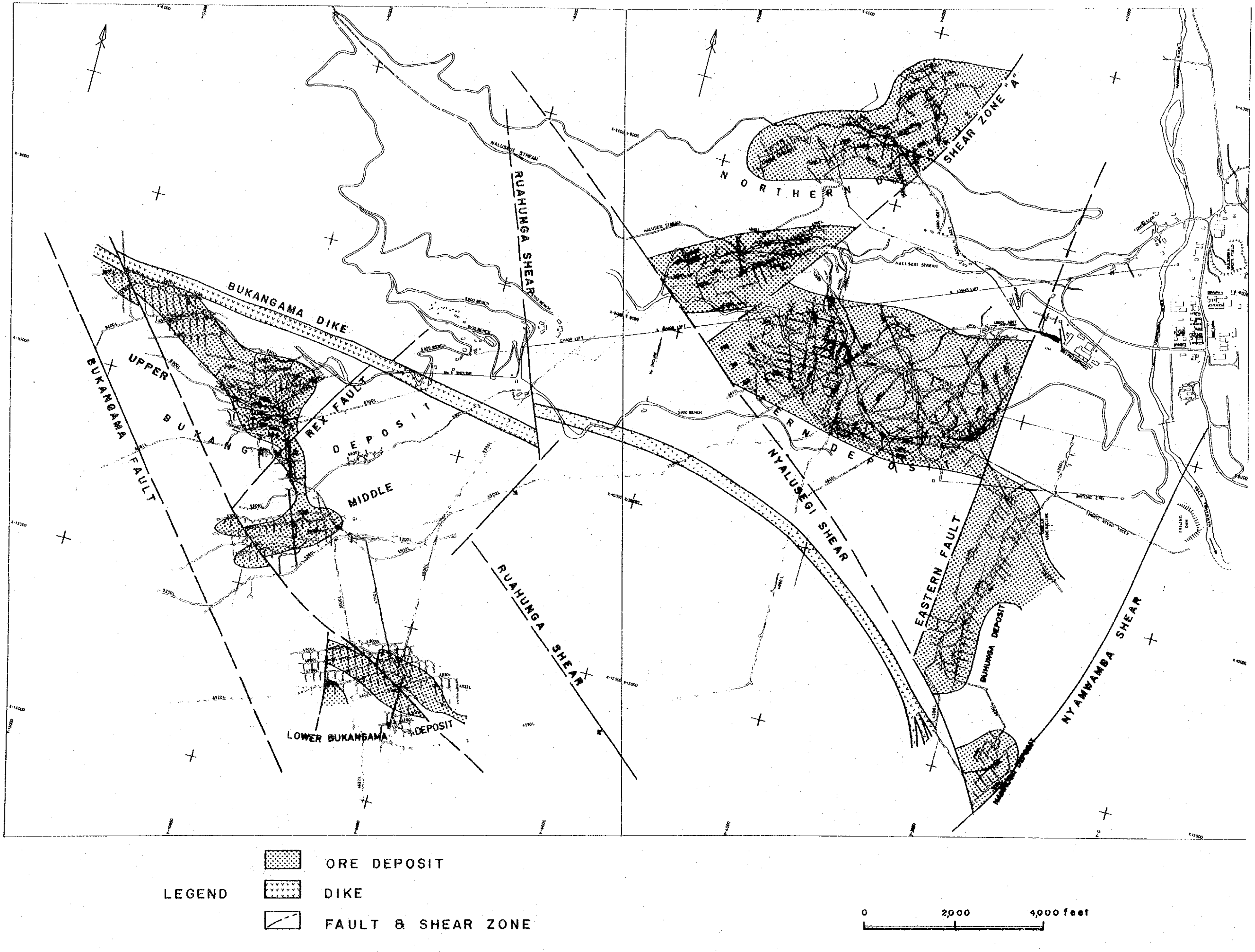


FIG. 4 UNDERGROUND PLAN & RELATIVE POSITION OF ORE DEPOSITS

The Upper and Lower Bukangama Deposits appear to plunge gently to ESE, which agrees with the plunge of the iso-clinal folding axes or that of the maximum concentration (L-1) of the microfoldings and lineations measurred in the western parts of the Bukangama Deposits (See 1-2-2 (b)). The plunge of the Middle Bukangama Deposit is, slightly different from the above, nearly horizontal in the ENE direction.

It is difficult to establish the plunge of those fault-bounded deposits in the Eastern Zone. However, it may be reasonably presumed from Fig.4. that they plunge generally to east or east southeast with very gentle dip.

b) Ore Minerals

The most prevailing sulphide mineral is pyrite and accounts for more than 50% of ore minerals. A sub-ordinate amount of chalcopyrite and pyrrhotite are visible in hand specimen.

Linaeite, pentlandite and sphalerite have been microscopically observed in very minor amounts.

Such secondary copper minerals as cuprite, chrysocolla, malachite, azurite, lubeckite, tenorite and chalcocite have been reported in the oxidized zone near surface.

Table-3 Assay Results of Ore Samples

Sample No.	Location	Cu(%)	S(%)	Co(%)	Ni(%)
US-K02201	49-08 20XC 4900L	3.37	27.10	0.70	0.060
US-K02202	4908 20XC 4900L	1.12	7.15	0.14	0.025
US-K02203	4908 20XC 4900L	5.42	21.00	0.69	0.12
US-K02204	4908 20XC 4900L	1.85	19,94	0.46	0.082
US-K02205	4908 20XC 4900L	0.94	4.87	0.10	0.024
US-K02206	24XC 5200L	0.27	3.83	0.01	0.017
A	Coarse Ore Stock Pile	9.20	25.27	0.58	-
B	Coarse Ore Stock Pile	1.92	14.15	0.35	-
C	Coarse Ore Stock Pile	1.45	5.10	0.09	-
D	Coarse Ore Stock Pile	0.36	1.22	0.02	-

Some ore samples were collected in the course of our examination, and analized for several elements. The results are tabulated in Table 3.

In general, the higher the copper content in ores, the higher the sulpher content, although there is no proportional relation between the two elements.

The ore contains a appreciable amount of cobalt. Our microscopic and EPMA observations identified siegenite instead of linaeite and pentlandite as Ni-Co minerals. Either siegenite or linaeite is too small in amount to explain cobalt content in ore. Cobalt occurs mostly in association with pyrite, without forming any specific Co-minerals. (Appendix 5)

1-3 Exploration

1-3-1 Previous Exploration (Dwg. No.1)

The entire extention of the Kilembe schist series was prospected by reconnaissance stream sediment and soil sampling in 1950S and 1960S. The follow-up exploration was subsequently carried out for four areas selected on the basis of the results of the reconnaissance prospecting.

The surface exploration was mostly completed by mid-1960S. The Table-4 is the summary of the exploration work carried out during this period.

The selected four areas were the Dubarea, the Kabili, the West Bukangama, and the Nkenda.

The Dubarea area, located in the mid-stream of the Dubarea River, was covered by a detailed soil grid for an area of 2.60 square miles. Anomalous copper values in soil samples were followed by a magnetometer survey, trenching, diamond drilling and an exploratory adit. Two samples from the trenches and a channel sample from the adit indicated copper values of 1.2%, 2.1% and 3.36% respectively.

However, judging from the structural feature of the ore deposit, it was concluded that the anomalies were associated with a small remnant of a synformal structure preserved on the hill-side.

The Kabili area, located on the south-west side-hill of the Kabili River, was investigated by a detail soil grid which defined an anomalous zone with a length of some 3000 feet. Pitting and trenching proved parts of the anomalous zone promising. A magnetometer survey indicated the presence of a strong conductor apparently associated with the geochemical anomalies.

Six diamond drill holes intersected mineralization mainly of pyrrhotite and magnetite. However, they failed to prove the presence of commercial copper values.

The west Bukangama area is located just west of the Bukangama Fault, along the up-stream of the Dungalilia River. Copper values in soil samples in the area were not exceptionally high in comparison with those in the other prospects. However, the area was considered important because of its proximity to the mine. In 1960 and 1961, pitting and trenching were carried out over the best geochemical anomalies, togather with a geo-physical survey using electromagnetic and magnetic method.

Table 4 SUMMARY OF EXPLORATION WORK

Areas Covered by Reconnaissance Stream Sediment Sampling	282.62 sq.miles
Areas Covered by Follow-Up Stream Sediment Sampling	43.22 sq.miles
Areas Covered by Ridge Soil and Drainage Stream Sediment Sampling	109.21 sq.miles
Areas Covered by Detail Soil and Geophysical Surveys	23.38 sq.miles
Total Geochemical Samples Collected	110,714 Samples
Pits and Trenches Excavated	96,448 Cu.Ft.
Diamond Drilling	7,568 Feet
Tunneling	165 Feet

The exploration was temporarily suspended after three drill holes had failed to intersect significant mineralization. In 1965, a geophysical survey was conducted by using a wide variety of instruments. In 1968, the Uganda Geological Survey undertook an I.P (Induced Polarization) survey which proved the presence of conductors. Diamond drilling was planned for the results of the geophysical survey, and a drill hole intersected a broad mineralization with a poor copper value.

The Nkenda prospect was located approximately 6 miles east of Kilembe or 5 miles north of the town of Kasese. A detailed grid soil survey, in 1963, indicated an anomalous zone some 600 feet long outlined by a 1000 ppm copper value in soil samples. Trenching and pitting revealed the concentration of mineralization associated with the UK-4/UK-5 contact. Three drill holes failed to penetrate the Middle Kilembe group (the ore horizon in the Kilembe Mine), though they intersected uneconomical mineralization at the contact between the UK-4 and the UK-5.

There was insufficient time available to assess all the tremendous piles of information in regard to the underground exploration. However, the most parts, where the ore horizon is expected, seem to have been fully explored by diamond drilling and with negative results except for the close vicinity of the known ore deposits. For examples, a number of drill holes were put along the long drifts and crosscuts at the 4500 level around the lower Bukangama Deposit. Never-the-less, none has encountered mineralization of commercial grade.

1-3-2 Future Exploration

a) Surface Exploration

Since the systematic exploration has been completed for the entire extension of the Kilembe Schist Series, there would be a very little chance to discover an extensive new ore body such as the Eastern Ore Deposit.

However, such moderate scale ore deposits as Bukangama Deposits could reveal only poor mineralization on the surface.

A geophysical method called "Charged Potential" may be suitable to search for burried ore deposits. The charged potential method, well-known for a longtime in principles, were first successfully applied for the similar ore deposits in Japan by the exploration department of a Japanese mining company.

The method is briefly described as follows and schematically illustrated in Fig.5.

One electrode is placed in electrical contact with a mineralized zone, on the surface, in the underground or in a drill hole. Another electrode is earthed infinitive distance away usually to the foot wall of the mineralized zone. Two search electrodes are required; one is fixed some distance away from a servey area not to affect a measurement at each station, and the other is moved at intervals along survey lines across the strike of the mineralized zone. A reciever measures a potential difference between the moving electrode and the fixed electrode when electric current is transmitted between the electrode in contact with the mineralized zone and the earthed electrode. Values of the potential difference are plotted on a map and an equipotential map is prepared as a result. The equipotential map indicates the location of the conductor which theoretically coincides with the maximum concentration of sulphides in the surveyed area.

The method appears to be ideally applicable to search for an ore deposit missed by two wide-apart drill holes which intersected poorly mineralized zones at the both ends of the ore deposit.

The West Bukangama prospect may be worth-while to be re-explored, because even a small scale ore deposit, if any, would be possible to be exploited without an immence investment, owing to its close proxmity with the present undeground workings.

In other prospects, exploitation of new ore deposits will require a subatantial investment, and small ore deposits may be unamenable.

b) Underground Exploration (Refer to 1-4-2)

Exploration targets in the underground seem to be limited to the possible or probable ore blocks of the known deposits and their immediate extentions. The following are major targets for the underground exploration.

1) West extention of the Lower Bukangama Deposit; west of the 22 cross-cut, above the 4900 level.

2) East extention of the Lower Bukangama Deposit; East of the 6 cross-cut, below the 4500 level.

3) The possible and the probable ore blocks below the 4050 level in the Buhunga Deposit.

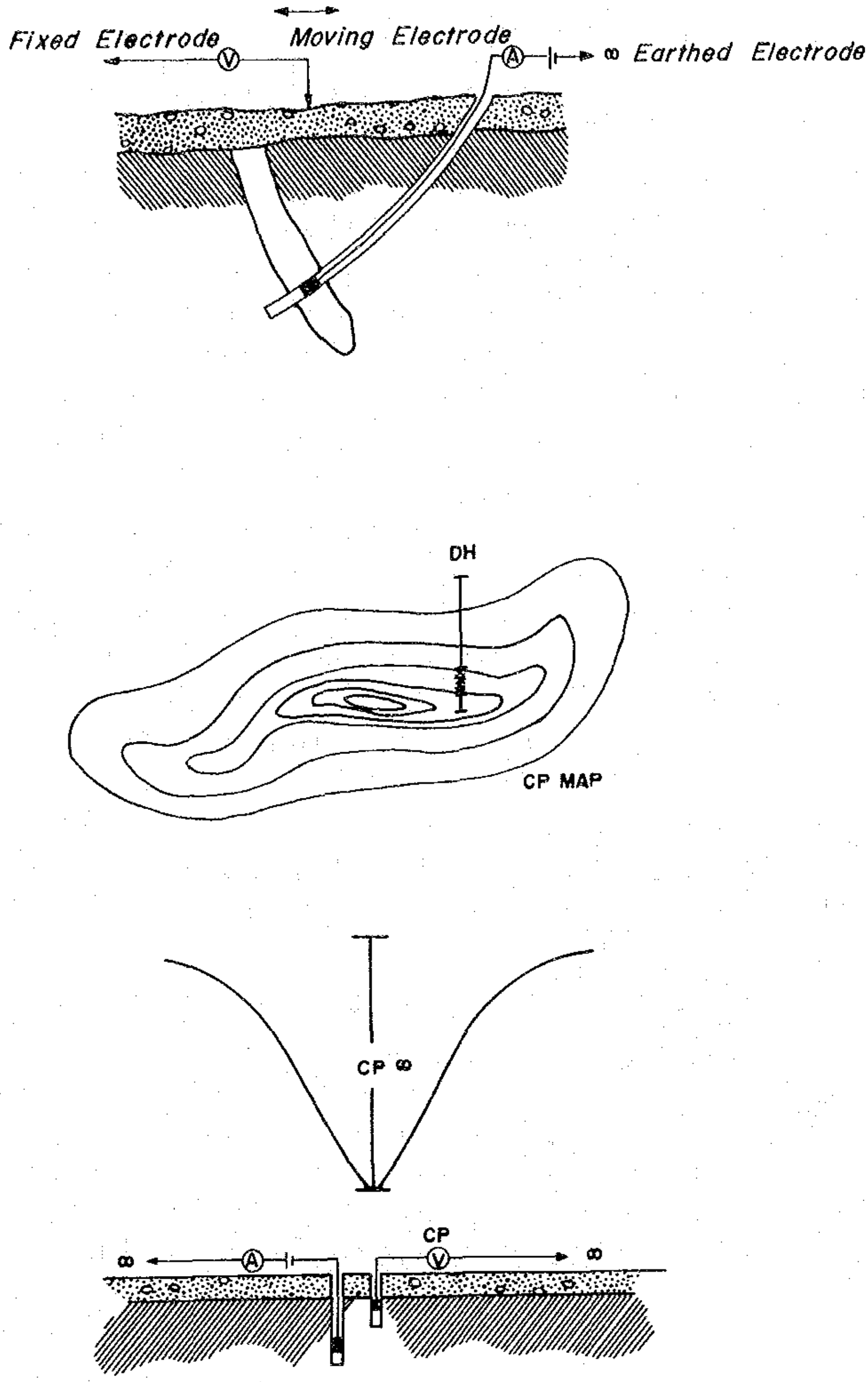


Fig. 5 Procedure of Charged Potenticel Method

1-4 Ore Reserve

1-4-1 Procedure of Ore Reserve Estimation

The ore reserve as of Dec.31st, 1977 has been estimated by the Geological Department of the Kilembe Mines Limited.

The principles, which have been applied for the ore reserve estimation by the Geological Department, seem to be fundamentally reasonable. However, the ore reserve maps have indicated no exhausted stopes and it was not always clear how to re-estimate the ore blocks partly mined, because no detail maps, which indicated the calculation procedure of their areas and volumes, were available.

The ore reserve re-estimation was made for the Upper, the Middle and the Lower Bukangama Deposits which accounted for nearly a half of the total ore reserve. The ore reserve of the possible category was excluded.

The ore reserves for the Eastern, the Stream, the Buhunga and the Numhuga Deposits were untouched, though minor arithmetic errors were corrected.

The procedure of the re-estimation is illustrated in Fig.6. and described step by step as follows.

1) An intersection of each drill hole, which indicates 0.5% Cu or above, is defined as an ore intersection.

2) An ore zone on a section is outlined by the 0.5% Cu assay boundries for the hanging and foot walls.

3) A true thickness is measured approximately perpendicular to the ore zone through the mid-point of each ore intersection.

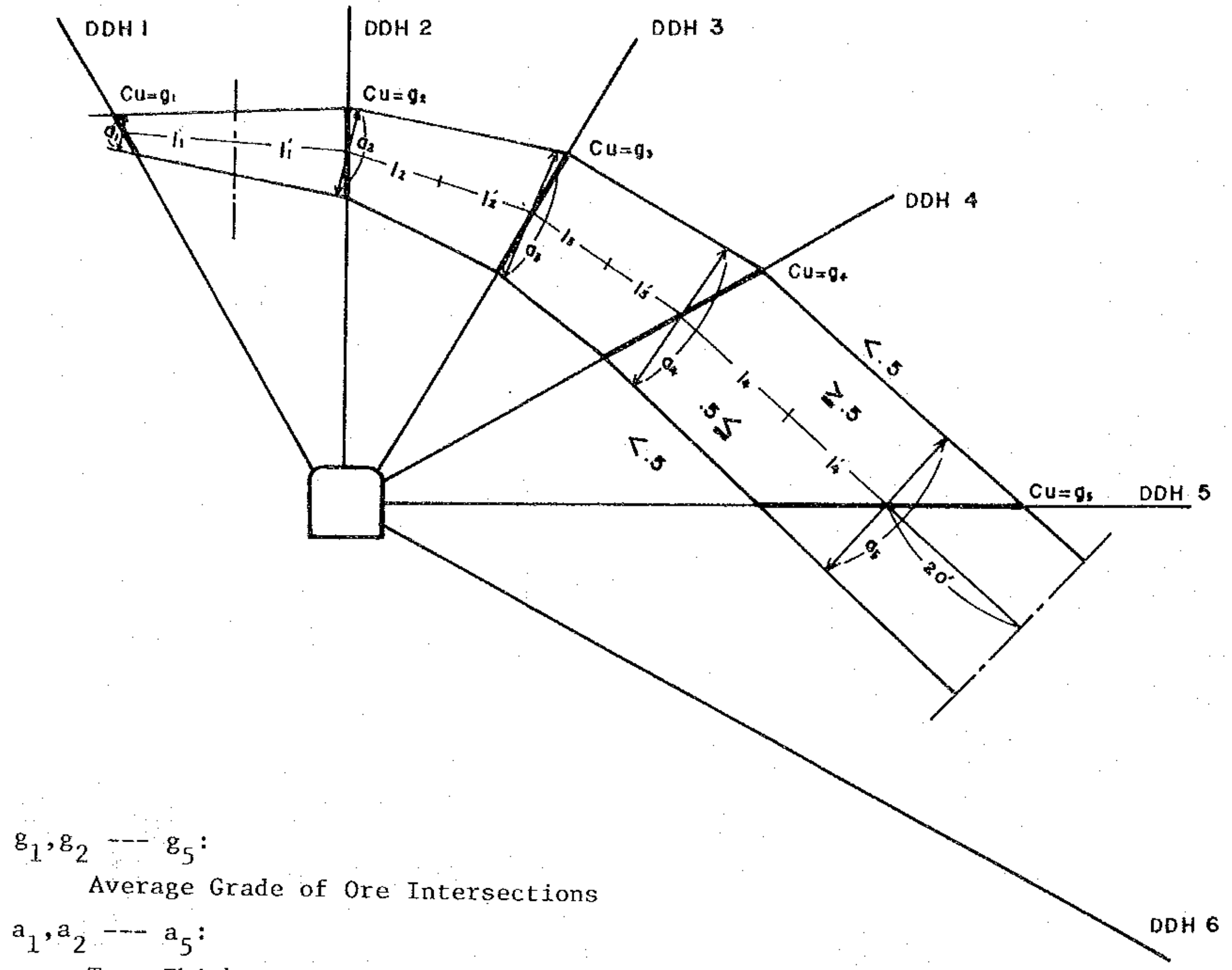
4) An average copper grade for each ore intersection is a weighted mean of copper values of cores for their assay lengths.

5) The true thickness and the average grade are applied halfway to the neighbouring ore intersections.

6) An area for each ore intersection is computed by multiplying the true thickness and the halfway distances to the neighbouring ore intersections.

7) An area for the ore zone is computed by integrating the areas for the ore intersections

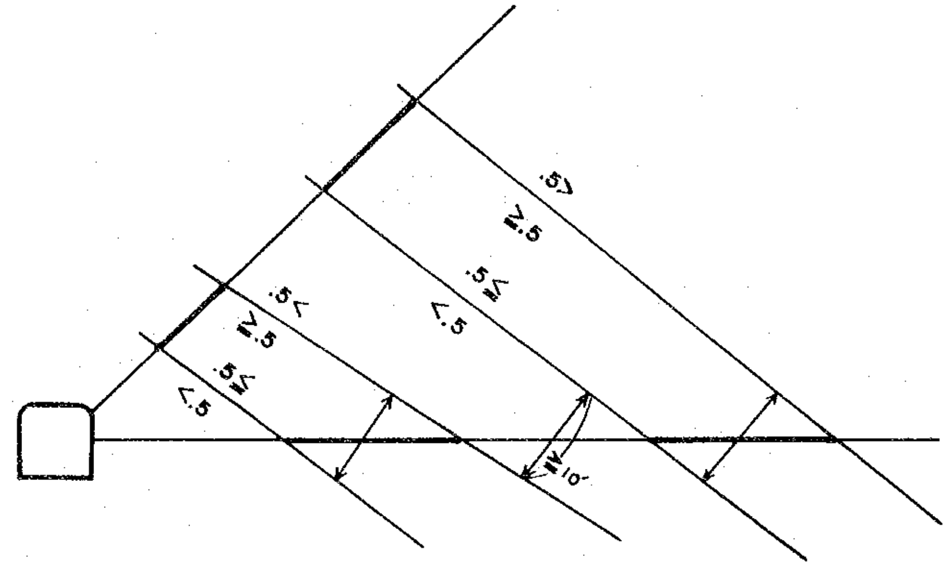
8) An average grade for the ore zone is obtained by calculating a weithted average of the average grades for the individual ore intersections in proportion to their representing areas.



g_1, g_2 --- g_5:
Average Grade of Ore Intersections

a_1, a_2 --- a_5:
True Thicknesses Through Mid-points of Ore Intersections

$l_1 = l_{1'}$, $l_2 = l_{2'}$, $l_3 = l_{3'}$, $l_4 = l_{4'}$:
Half Way Distances to Neigbouring DDHs



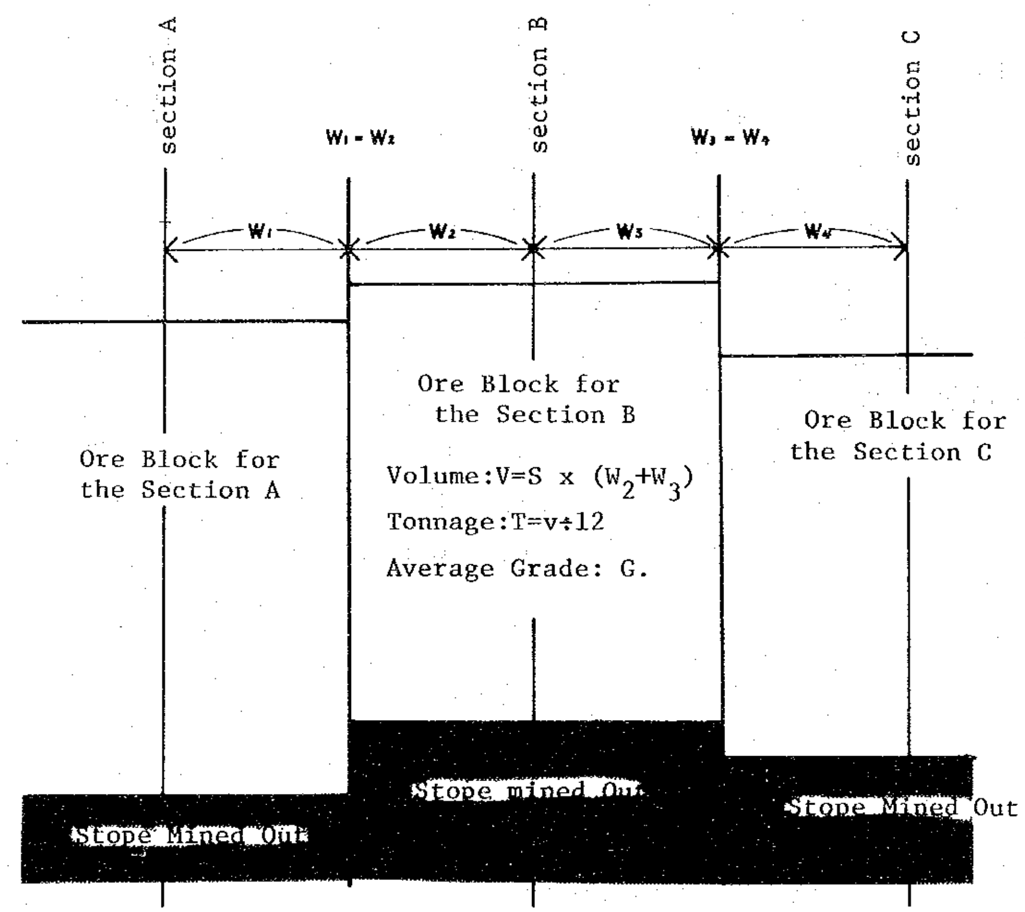
$a_1 < 10'$, $g_1 < 1.00$ or $a_1 \times t_1 < 15$

The Area for The Ore Section : S.

$$S = a_2 \times (l_1' + l_2) + a_3 \times (l_2' + l_3) + a_4 \times (l_3' + l_4) + a_5 \times (l_4' + 20)$$

The Average Grade for The Ore Section: G.

$$G = \frac{g_2 \times a_2 \times (l_1 + l_2) + g_3 \times a_3 \times (l_2' + l_3) + g_4 \times a_4 \times (l_3' + l_4) + g_5 \times a_5 \times (l_4' + 20)}{S}$$



Plan Showing Ore Block

Fig. 6 Procedure of Ore Reserve Calculation

9) The area and the average grade for the ore zone is applied halfway to the neighbouring sections

10) A volume of an ore block is computed by multiplying the area for the ore zone and the halfway distances to the neigbouring sections.

11) The average grade for the section represents the average grade for the ore block.

12) A tonnage of the ore block is computed by deviding the volume by 12 (tonnage factor), which corresponds with a specific gravity of 2.99. The tonnage is metric here-in-after unless otherwise it is commented.

13) 85% of the tonnage is assumed to be extracted by mining (extractable ore)

14) Wall rocks at a grade of 0.20% Cu. is assumed to dilute the extractable ore by 15% (dilution).

15) An ore reserve of the block is calculated by combining the extractable ore and the dilution.

16) In case that an ore zone is interrupted by mining stopes or barren zones, separated zones are individually treated by following the above procedure.

17) An ore intersection less than 10 feet in thickness, lower than 1.00% in an average grade, and/or less than 15 Cu%-Ft in a copper accumulation, is excluded from the ore zone.

18) In case that an ore zone contains a portion of lower than 0.5% Cu in its average grade with a true thickness more than 10 feet, the ore zone is differentiated into two zones, the hanging wall and foot wall ore zones. They are treated as indipendent ore zones.

19) No plan maps were available at the time of the re-estimation to figure an exact shape of a mining stope. A shape of a mining stope on a section is applied halfway to the neighbouring sections.

The result of the re-estination is tabulated in Appendix-1 and shown in Dwg. 3-A,B,C.

1-4-2 Result

a) Upper Bukangama Deposit

The result of the re-estimation indicated a substantially large ore reserve of 1,243,240 tons with an average grade of 1.40% Cu for the ore blocks with average grades of 0.90% Cu or above, in comparison with the ore reserve of 475,800 tons with an average grade of 1.58% Cu as estimated by the Geological Department of Kilembe Mine at the year end of 1977.

It is apparently one of the reasons of the difference that the Cu grade of 0.5% for outlining the ore zone in the re-estimation is much lower than the cut-off grade of 1.00% Cu applied by the Geological Department.

However, it may be also presumed for other reasons that some ore blocks would have been excluded from the table of the ore reserve due to their in accessibility, although they are not listed in the table of the irrecoverable ore reserve either, or some mining stopes would have been left undone on the geologic sections which were utilized for the re-estimation. In fact, there are a greater number of remnant ore blocks left according to the geologic sections than those listed in the ore reserve table prepared by the Geological Department.

At any event, the result of the re-estimation for the Upper Bukangama Deposit is necessary to be reviewed by the Geological Department.

A greater part of high grade ore blocks has been mined out. The ore blocks with average grade of less than 1.3% Cu account for nearly two-third of the total ore reserve (Table 5).

The ore reserves for the 1720 winz and the 1760 winz pillars are presently irrecoverable and excluded from the ore reserve as mined.

The ore reserve as mined is estimated at 1,151,010 tons with an average grade of 1.41% Cu for the ore blocks with average grades of 0.9% Cu or above, or at 521,600 tons with an average grade of 1.81% Cu for the ore blocks with average grades of 1.3% Cu or above (Table 6).

b) Middle Bukangama Deposit

The high grade ore blocks are mostly exhausted in the Middle Bukangama Deposit.

The ore reserve for the 1720 pillar is presently irrecoverable and excluded from the ore reserve as mined.

The ore reserve as mined, accordingly, is estimated at 367,740 tons with an average grade of 1.29% Cu for the cut-off grade of 0.90% Cu, or at 81,180 ton with an average grade of 1.77% Cu for the cut-off grade of 1.30%.

c) Lower Bukangama Deposit

The re-estimated ore reserve of 769,960 tons with an average grade of 1.74% Cu for the ore blocks with grades of 0.9% Cu or above is much smaller than the ore reserve of 1,238,100 tons at an average grade of 1.73% Cu as of Dec.31,1977.

The ore reserve as of Dec.31,1977 appears to be over-estimated for some ore blocks. The following blocks are major causes of the difference

1) The block 50-05 has been fully explored by a number of drill holes which intersected ore of sub-marginal grade and/or thickness. Only one hole intersected ore with a true thickness of 26 feet and an average grade of 2.26% Cu. The ore of the block may be recovered partly but for a minor portion.

2) Greater parts of the block 47-08, 46-08, and 45-08 have been proved to include only submarginal ores by a number of drill holes.

3) An ore reserve of 314,900 tons at an average grade of 1.70% Cu of the probable category is allocated in the ore reserve as of Dec.1977 for the block EX-1100 Winz Area. Along the 1100 Winz, 8 of 10 drill holes intersected sub-marginal ores. Two drill holes, located far apart from each other, indicated ore of 2.04% and 3.11% Cu for true thicknesses of 15 and 14 feet respectively. The most of the ore reserve seems to be uneconemical, although minor portions may be minable.

The ore reserves as mined is estimated at 769,960 tons with an average grade of 1.74% Cu for the blocks with grades of 0.90% Cu or above, or at 670,120 tons with an average grade of 1.84% Cu for the blocks with grades of 1.30% Cu or above.

1-4-3 Ore Reserves for the deposits in the Eastern Zone

The ore reserves for the Stream, the Eastern, the Buhunga, and the Numguha Deposits were not re-estimated.

According to the ore reserve estimation as of the year end of 1977, these deposits include ores of much higher grade than the Bukangama Deposits.

42

Table 5 Ore Reserve by Cu. Grade

Ore Deposit	Range	≥2.9		2.9> ≥2.5		2.5> ≥2.1		2.1> ≥1.7		1.7> ≥1.3		1.3> ≥.9		.9>	
		Tonnage	Cu%	Tonnage	Cu%	Tonnage	Cu%	Tonnage	Cu%	Tonnage	Cu%	Tonnage	Cu%	Tonnage	Cu%
Eastern	by Grade	105,900	3.43	35,200	2.71	289,800	2.25	284,400	1.78	236,700	1.55	159,200	1.10	0	-
	Cumulative	105,900	3.43	141,100	3.25	430,900	2.58	715,300	7.26	952,000	2.08	1,111,200	1.94	1,111,200	1.94
Stream	by Grade	16,300	3.60	23,700	2.52	11,500	2.18	91,700	1.82	28,200	1.54	0	-	0	-
	Cumulative	16,300	3.60	40,000	2.96	51,500	2.79	143,200	2.17	171,400	2.06	171,400	2.06	171,400	2.06
Buhunga Namhuga	by Grade	102,500	3.16	36,300	2.57	42,900	2.42	214,100	1.78	299,300	1.45	72,600	1.13	0	-
	Cumulative	102,500	3.16	138,800	3.01	181,700	2.87	395,800	2.29	695,100	1.92	767,700	1.85	767,700	1.85
Bukangama Upper	by Grade	45,310	3.37	9,560	2.69	40,880	2.40	119,090	1.87	333,440	1.45	694,960	1.08	349,880	.83
	Cumulative	45,310	3.37	54,870	3.25	95,750	2.89	214,840	2.32	348,280	1.79	1,243,240	1.40	1,593,120	1.27
Bukangama Middle	by Grade	0	-	14,340	2.60	0	-	20,918	1.82	45,930	1.48	337,510	1.17	47,920	.57
	Cumulative	0	-	14,340	2.60	14,340	2.60	35,250	2.13	81,180	1.77	418,690	1.29	466,610	1.21
Bukangama Lower	by Grade	0	-	72,690	2.70	98,680	2.28	158,350	1.85	340,400	1.52	99,840	1.12	36,570	.82
	Cumulative	0	-	72,690	2.70	171,370	2.46	329,720	2.17	670,120	1.84	769,960	1.74	806,530	1.70
Total	by Grade	270,010	3.33	191,790	2.65	483,760	2.28	888,550	1.81	1,283,970	1.49	1,364,110	1.11	434,370	.80
	Cumulative	270.010	3.33	461,800	3.05	945,560	2.65	1,834,110	2.25	3,118,080	1.93	4,482,190	1.68	4,916,560	1.60

Note: Including 1200, 2000 and 2300 shaft pillars of the Eastern Deposit, 1720 and 1760 Winz Pillars of the upper Bukangama Deposit, and 1720 winz pillar of the Middle Bukangama Deposit.

The Stream Deposit used to be an exceptionally high grade one and to have contributed to maintaining the mill head grade. Its reserve is presently declined to 171,400 tons at an average grade of 2.06% Cu for the cut-off grade either of 0.9% Cu or of 1.30% Cu.

The Eastern deposit is still large in its ore reserve especially in the Lower Eastern Ledgers. However, it must be noted that many of temporary pillars are included in the ore reserve as of the year end 1977. At least, the pillars for the 1200, 2000 and 2300 Shafts have to be excluded from the present ore reserve as mined, because they are irrecoverable untill the end of mining the Eastern Deposits. The reserve as mined, excluding these pillars, is estimated at 892,300 ton with an average grade of 1.96% Cu for the cut-off grade of 0.90% Cu, or at 733,100 tons with an average grade of 2.14% Cu for the cut-off grade of 1.30% Cu.

An ore reserve of 166,400 tons with an average grade of 1.60% Cu is allocated as a probable ore in the Buhunga Deposit for the blocks below the 4050 Level. The ore reserve must be confirmed by a number of additional drill holes, because the Buhunga Deposit is structurally very complex. It must be also noted that the exploitation of the ore blocks will require a substantial development work. The combined ore reserve as mined for the Buhunga and the Numhuga deposits is estimated at 767,700 tons with an average grade of 1.85% Cu for the cut-off grade of 0.90% Cu, or at 695,100 tons with an average grade of 1.92% for the cut-off grade of 1.30% Cu.

1-4-4 Summary

The ore reserve as mined is summerized in Table 6. The total ore reserve is estimated at 4,120,110 tons with an average grade of 1.69% Cu for the cut-off grade of 0.90% Cu or at 2,872,500 tons with an average grade of 1.94% Cu for the cut-off grade of 1.30% Cu.

The ore reserve re-estimation is incomprehensive and unsatisfactory in accuracy,due to lack of information available and time to be spent.

It may be recommended that a comprehensive re-evaluation of the ore reserve be undertaken by re-assessing all the past information, re-survying all the stopes and tunnels, additional drilling where necessary and so on, because small mining stopes ore scattered in various places and have been abandoned for uncertain reasons, and some ore intersections in drill holes are doubtfully located.

Table-6 Summary of Ore Reserve as mined

Ore Deposits		Cut-OFF .90% Cu			Cut-OFF 1.30% Cu		
		Tonnage(M.T.)	Cu %	Remarks	Tonnage(M.T.)	Cu %	Remarks
Eastern		892,300	1.96	1200, 2000 & 2300 shaft pillar excluded	733,100	2.14	1200, 2000 & 2300, Shaft pillars excluded
Stream		171,400	2.06		171,400	2.06	
Buhunga/Numhuga		767,700	1.85		695,100	1.92	
Bukangama	Upper	1,151,010	1.41	1720, 1760 Wins pillars excluded	521,600	1.81	1720 winz pillar excluded
	Middle	367,740	1.29	1720 winz pillar excluded	81,180	1.77	
	Lower	769,960	1.74		670,120	1.84	
Total		4,120,110	1.69		2,872,500	1.94	

1-5 Mining

1-5-1 Outline of Mining Operation

a) Underground Structure

The underground structure is schematically illustrated in Fig. 7.

The underground of the Kilembe Mine has been developed by four main haulages, at the 5200 (5220), the 4500(4522), the 4300 and the 4050 levels, and by four main shafts, namely, the 1200, the 2000, the 2300, and the 1150 shafts.

The 4500 haulage is the main adit to transport ore from each ore deposit to the mill, to drain underground water above the 4500 level, and to distribute men and materials to the mine workings.

The 4300 haulage serves mainly as a drainage adit to drain underground water below the 4500 level, though some men and materials are transported through the hanlage.

The 4050 haulage, blind to the surface, is designated mainly for the transportation of ore and waste to the 1200 shaft.

The 2000 and 2300 shafts are serving to transport ore and waste to the 4500 level and to distribute men and materials to the mine workings in the Stream, the Eastern, the Buhunga and the Numhuga Deposits.

The 5200 haulage and the 1150 shaft are located in the Bukangama Deposits to serve mainly for the transportation of men and materials

The 1720 and 1100 winzes in the Bukangama Deposits are service inclined shafts to distribute men and materials to the working places

b) Mining Method

Other than the open pit mining applied in a part of the Northern Deposit, various mining methods have been tested and applied in the history of the underground mining operation.

However, the cut-and-fill method has been finally established for the most parts of the mining stopes, though the shrinkage is limitedly applied to the portions where the dip of the ore deposits is appreciably steep.

The cut-and-fill method applied at the present time is classified into two varieties; one is termed the Longitudinal Open Cut-and-Fill Stoping with Horizontal Breast and the other, the Longitudinal Close Cut-and-Fill Stoping with Horizontal Breast.

The former method, more productive than the latter, is only applicable to the stopes with the relatively competent ground.

The latter, more prevailing in the mine recently, is much safer than the former.

The stopes are progressively filled by sand slime through bore holes drilled into the stopes. The sand slime is produced as tailings of the ore processing and transported by pipe lines to the mine workings from the mill.

The cut-and fill method is generally rather costly and unproductive than other methods such as the open-stoping and the sublevel stoping . However, the cut-and-fill method is only a suitable one for this mine owing to the gentile dip of the ore deposits in general and the brittle nature of the ground caused by a combination of cross faults and strike faults.

c) Developement and Preparation

Drifts are driven to the foot-wall of the ore deposits with cross cuts at a 80 feet interval. Diamond drill holes are drilled in a vertical fan pattern along each cross-cut to establish the grade and the shape of the ore deposits in detail.

An unit stope has a dimention of 180 to 240 feet in length along the strike and of around 40 feet in width horizonally across the strike.

Two service raises are raised into the stope from the foot-wall cross cut and subsequently connected to each other.

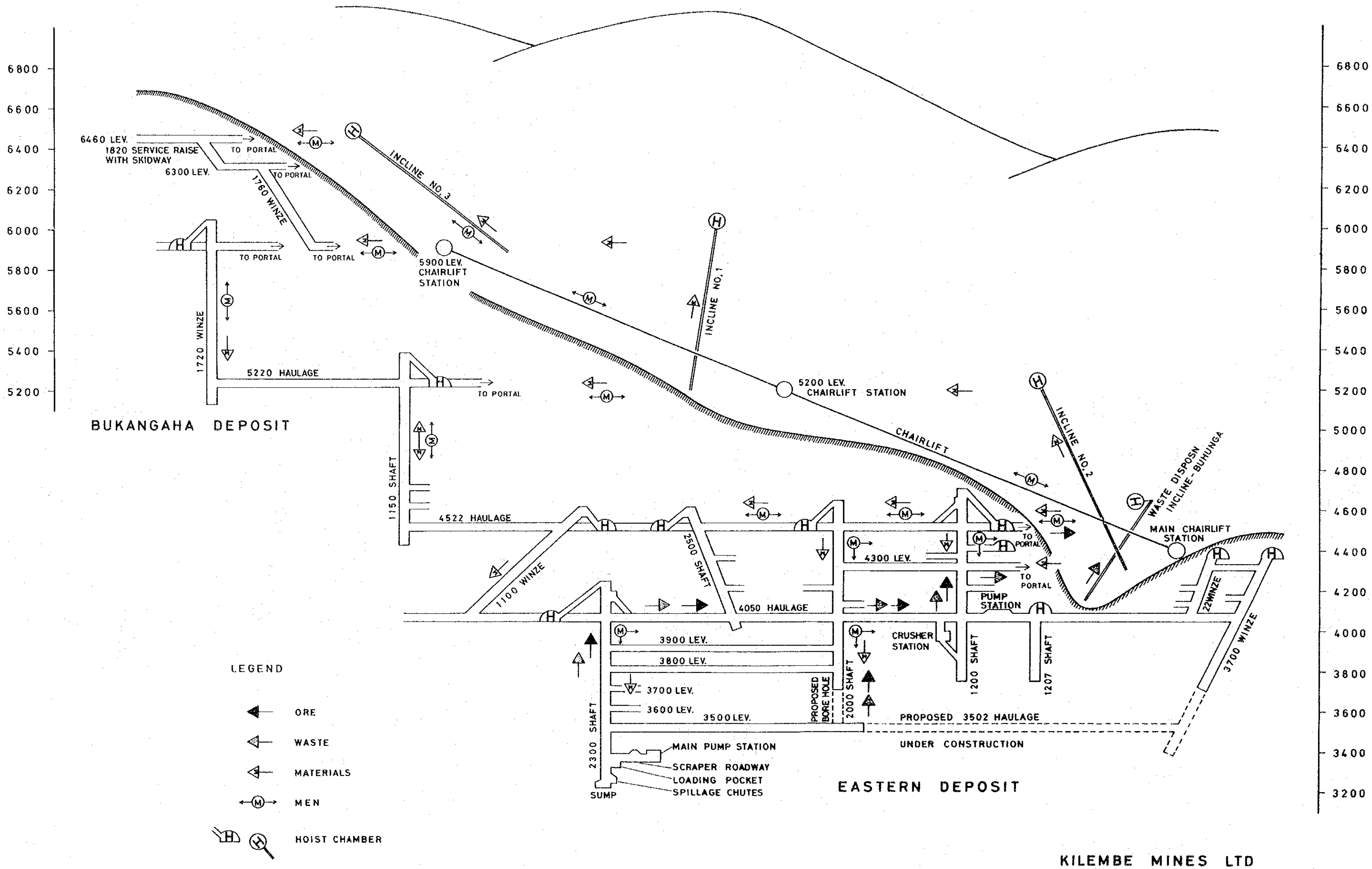
Vertical and sill (lateral) pillars are left angainst the neibouring stopes while the mining is in progress, and are recovered later.

d) Transportation System

Tram ways are installed in the main hawlages at the 5200, the 4500, the 4300 levels. Electric locomotives are serving as transportation media in these haulages.

The ore and waste from the ore deposits in the Eastern Zone are transported to the 4050 level either by the 2000 or the 2300 shaft. The ore is further transported to the crasher station located beside the 1200 shaft and hoisted to the 4500 level through the 1200 shaft after being crushed to sizes less than 270 mm in diameter. The waste is transported to the 4300 level through the 4050 haulage and the 1200 shaft, and hauled out of the portal of the 4300 haulage.

FIG. 7 LONGITUDINAL SECTION



KILEMBE MINES LTD

All the ore and the waste below the 5200 level from the Bukangama Deposits are transported through ore and waste paths to the 4500 level by gravity, and hauled to the primary crusher in the mill through the 4500 haulage.

All the main haulages and the shafts are also serving for distributing men and materials to mine workings as afore-explained.

In the Bukangama Deposits, a shaft and two winzes are, as explained before, serving for distributing men and materials to working places. Hoisting machines working for the shafts and winz above mentioned are listed in Table 7.

Other than the underground transportation system above described, a chair lift and three inclines are constructed on the surface with a purpose of transporting men and materials to the portals of adits.

For surface transportations, the chair lift, with a length of 7650 feet along its inclination, is serving between the mine office area and the 5200 or the 5900 level, the incline No.1 between the 5200 and the 5900 levels, the incline No.2 between the 4500 and the 5200 levels, and the incline No.3 between the 5900 and the 6460 levels.

The transportation system is shown in Fig 7.

e) Drainage Reticulation

The drainage reticulation is diagramatically shown in Fig 8.

The 4500 haulage drains most of underground water above the 4500 level, although some water is discharged through such adits as the 6800, the 6600, the 6460, the 6300, the 5900 and the 5200.

Two dums are placed at the 5900 and the 4500 level to store water temporarily. The 1720 winz and the 1150 shafts serve as main water passages through the Bukangama Deposits.

Underground water between the 4500 and the 4300 levels flows down to the 4300 level and is drained through the 4300 haulage.

Underground water below the 4300 level is pumped up to the 4300 level through the 2300 and the 2000 shaft.

Two main pump stations are located at the 3400 level and the 4050 level.

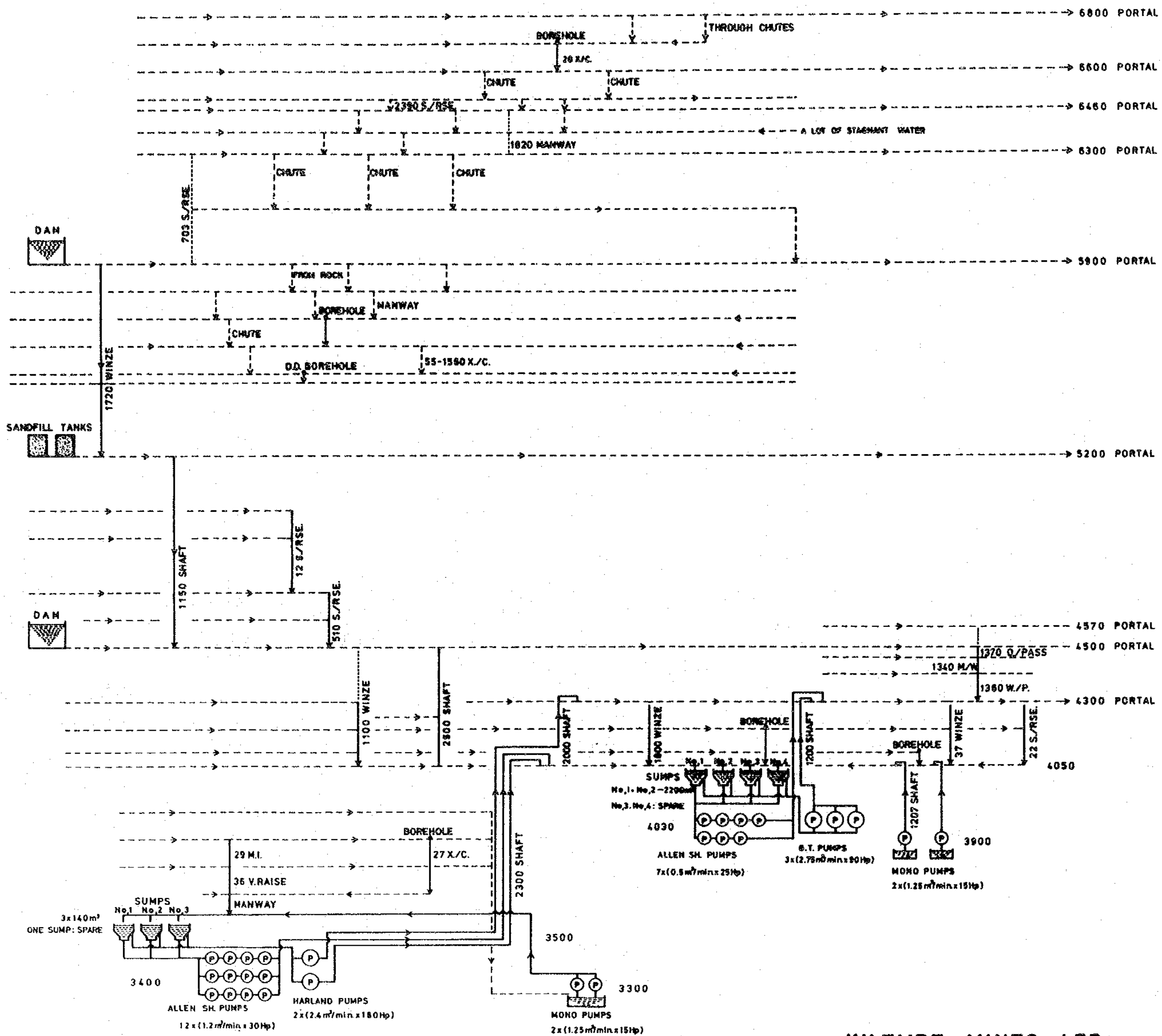
Water below the 4050 level is once collected in sumps at the 3400 level and pumped up to the 4050 level through the 2300 shaft.

Water above the 4050 level, combined with the water pumped up from the sumps at the 3400 level, is collected in sumps located at the 4050 level,

Table 7 Hoisting Machine

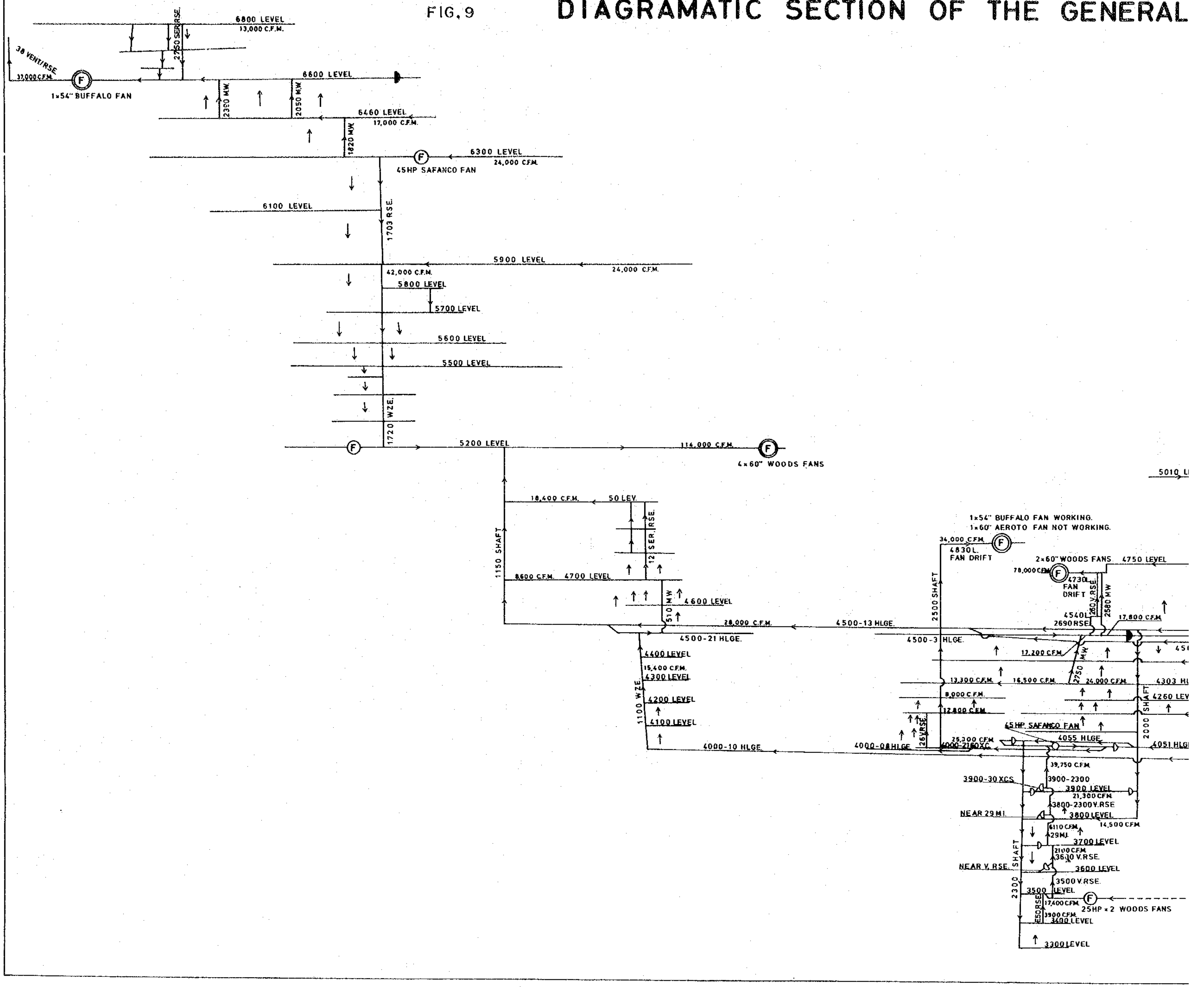
Location	Type	Capacity	Motor	Rope	Use
1200 Shaft/4500L	2 Drums, 2 Skips	3,500kg	250HP x 3300V x 585rpm	1" x 420m	Ore, Men
1200 Shaft/4400L	2 Drums, 2 Skips	2,000kg	160HP x 415V x 735rpm	7/8" x 420m	Waste
2000 Shaft/4500L	2 Drums, 1 Cage	40men	250HP x 3300V x 750rpm	1" x 420m	Men, Materials
2300Shaft/4050L	2 Drums, 1 Skip	2,000kg	250HP x 3300V x 735rpm	7/8" x 420m	Ore
1207 Winze/4050L	1 Drum, 1 Skip	750kg	100HP x 415V x 740rpm	5/8" x 150m	Materials
1100 Winze/4500L	1 Drum,		90HP x 415V x 730rpm	5/8" x 450m	Waste, Materials
1720 Winze/5900L	1 Drum, 1 Skip		75HP x 415V x 975rpm	5/8" x 450m	Material
1150 Shaft/5900 L	2 Drums, 1 Skip	15men	100HP x 415V x 740rpm	3/4" x 300m	Men, Materials
4690L Incline Shaft	1 Drum,	3300kg	160HP x 415V x 960rpm	7/8" x 900m	Waste
No.1 Incline(Surface)	2 Drums, 2 Skip	40men	250HP x 3300V x 490rpm	7/8" x 525m	Men, Materials
No.2 Incline(Surface)	2 Drums, 2 Skip	40men	125HP x 415V x 975rpm	7/8" x 750m	Men, Materials
No.3 Incline(Surface)	1 Drum, 1 Skip	15men	170HP x 415V x 585rpm	7/8" x 900m	Men, Materials

FIG. 8 **DRAINAGE RETICULATION**



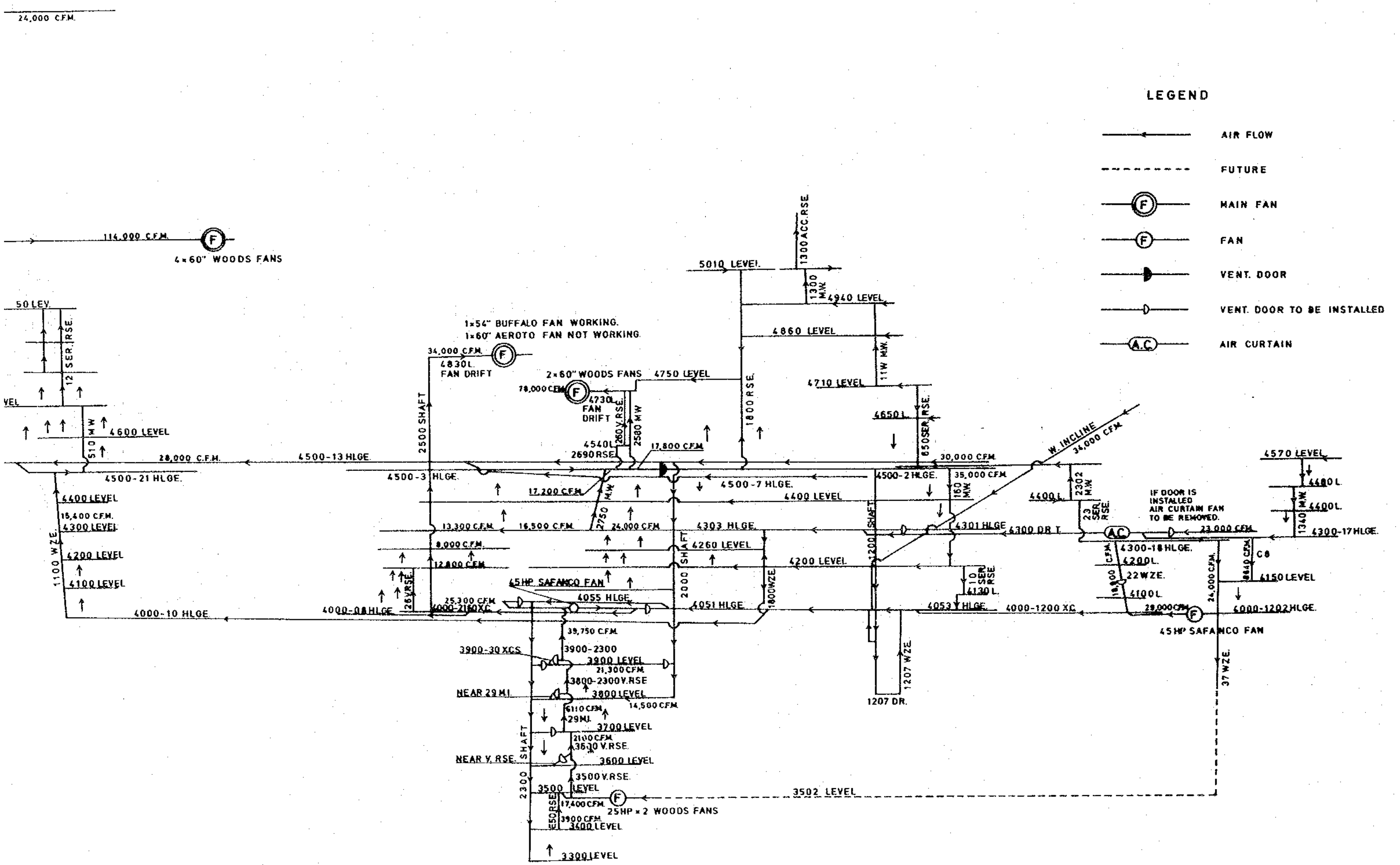
KILEMBE MINES LTD

FIG. 9 DIAGRAMATIC SECTION OF THE GENERAL



52

AGRAMATIC SECTION OF THE GENERAL VENTILATION AIR FLOW



KILEMBE MINES LTD

pumped up to the 4300 level through the 2000 shaft, and finally drained through the 4300 hanlage.

7 to 8 cubic meters per minute of water was being drained through each of the 4300 and the 4500 level at the time of our observation. However it will vary seasonally or depending on activity in the underground.

f) Ventilation

The ventilation system is diagramatically shown in Fig.9.

Inflow taken through the 6800 and the 6300 adit is circulated through working places above the 6300 level in the Upper Bukangama Deposits and is vernilated through the 38 ventilation raise forcefully by a Buffalo fan installed at the 6600 level.

Air taken through the 5900 adit, combined with a part of the inflow through the 6300 level, is circulated through working places between the 6300 and the 5200 levels in the Bukangama Deposits, and is ventilated through the 5200 haulage forcefully by four Woods fans installed at the 5200 levels.

53

Most of inflow, which is circulated through working places in the Eastern Zone and in the Lower Bukangama Deposit, is introduced through the 4500 and the 4300 haulages.

A part of the inflow, after the circulation through the working places in the Lower Bukangama Deposit, is fanned out through the 5200 level.

A greater part of the inflow is circulated through the working places in the Eastern Zone, and is ventilated by either a Buffalo fan and an Aeroto fan installed at the 4830 level or by two Woods fans installed at the 4730 level.

g) Surface Facilities

Electricity is provided from the Mubuku Power Station belonging to Kilembe Mines Ltd. and from the Owen Falls Hydro-electric Power Station of the Uganda Electric Board. A main power subtation in the mine office area, recieving the power from these power stations, distributes electricity to the underground, the mill, the mine office and the residential areas. Two transformers with capacities of 2500 KVA and 1200 KVA, and two oil circuit breakers are installed in the main power substation.

Annual supply of electricity is summerized in Table 8.

Table 8 Electricity Supply (MWH)

Power Station	1977	1976	1975	1974	1973
Power EX Mobuku	19,698	27,963	28,688	36,376	39,510
Diesel Power	143	220	115	822	478
Purchases Ex U.E.B.	3,020	5,015	7,377	6,576	7,668
Total M.W.H consumed	21,861	33,198	36,180	43,774	47,656

Two compressor stations are located in the mine office area and on the surface in the Bukangama Zone.

The compressor station in the mine office area, installed with 7 compressor with capacities ranging from 400 to 550 HP., provides compressed air to the underground, the mill, and the work shop, while the Bukangama Compressor Station, installed with 7 compressor with capacities ranging from 100 to 550 HP., serves only for the underground. The compressors are listed in Table 9.

A workshop and a garage are also located in the mine area. The workshop is serving for repairing machines and equipments of the underground and the mill. The garage provides surface vehicles and mobile heavy equipments, and repairs them. Machines and tools in the workshop are listed in table 10.

1-5-2 Current Situation

Production has been competely ceased since the fall of 1977, due to insufficient supply of materials and a number of unserviceable machines and equipment. The present mining operation is limited to the development of a few working places and the maintainance of the underground conditions, facilities, machines and equipment.

Drilling operation has been also suspended for a while due to lack of diamond crowns, although a sufficient number of drilling machines are in operative conditions.

Of all machines and equipments, the pumps which pump the underground water below the 4300 level, are most important and critical, because most parts of the Eastern, the Stream, the Buhunga and the Numhuga Deposits will be drowned under water, if these pumps stop working. They were still

Table 9 Compressor

Location	Make	Capacity(M^3/m)	Pressure(kg/cm^2)	Motor
Mine Office Area	Ingersoll Rand	85	7	500HP x 300rpm
	Elly	70	7	400HP x 386rpm
	Bellis	70	7	550HP x 250rpm
	Bellis	70	7	550HP x 250rpm
	Ingersoll Rand	70	7	450HP x 334rpm
	Bellis	70	7	550HP x 250rpm
	Bellis	70	7	550HP x 250rpm
Bukangama Area	Sullivan	14	7	100HP x 589rpm
	Ingersoll Rand	21	7	125HP x 273rpm
	Ingersoll Rand	21	7	125HP x 273rpm
	Sullivan	14	7	100HP x 589rpm
	Ingersoll Rand	70	7	500HP x 300rpm
	Bellis	85	7	550HP x 250rpm
	C.P	11	7	160HP x 1470rpm

Table 10 Michine and Tools in Workshop

Machine or Tool	No.	Description
Jameson Lathe	1	Swing:45", Length:15'6", Centre Height:26"
Denham Super Speed Lathe	1	Swing:54", Length:16', Centre Height:14-3/4"
Denham Swing Lathe	1	Swing:32", Length:8'3", Centre Height:8-1/2"
Tos Lathe	1	Swing:28", Length:7', Centre Height 9-7/8"
Wilson Lathe	1	Swing:24", Length:5' 2-1/2", Centre Height 8"
Fortuna Lathe	1	Swing:12", Length:4'2", Centre Height 5-7/8"
Turret Lathe	1	Swing:14", Length:6'11", Centre Height 7"
Shaping Machine	1	Type:6MR, Length of stroke:24"
Milling Machine	2	Type:Universal / Presatrice C.M.B.O.
Radial Drilling Machine	2	Capacity Drill : 1/6" up to 2"
Planning Machine	1	Length of Table:96", Width:36", Length of Bed: 194"
Hydraulic Press(vertical)	1	Length: 13-1/2', Capacity:1500 P.S.I.
Hydraulic Press(Horizontal)	1	Length:4', Capacity:5000 P.S.I.
Screwing Machine	2	Range of Screw Products:1/2" - 1-1/2"
Rolling Machine	1	Length: 6'3"
Cropping Machine	1	Type: 210/3
Power Saw (Small)	1	Type of Blade: 14" x 1-1/2"
Power Saw (Big)	1	Type: Tos 1965
Small Drilling Machine	1	Type: Senstive Drilling m/c, Capacity Drill 1/16" - 1/2"
Small Drilling Machine	1	Type: Handy Autmatic m/c
Shearing Machine	1	Capacity: 1/16" - 3/8" thick x 8' long, Length of Blade: top - 8'3", bottom 9'9"long
Power Hammer	1	Type: Pilkington 3
Automatic Welding Machine	1	Type:Muramatic MK. 2c
Grinding Machine	1	Type:Milford 12" Size of Stone: 12" x 1-1/2" x 1-1/4" Bore
Welding Set	8	Type:Quasi Arc Single Operator(5), Double Operators(1), D.C.Rectifier(1), Euo-mat(1).

in working condition at the time of our examination. However, most of them were in critical condition and no spare pumps were prepared.

Many of rock drills, scrapers and shovel loaders are unserviceable due to lack of their spare parts.

Hoisting machines are maintained in working condition, though some of their spare motors are unserviceable and spare ropes for some hoisting machines are required to be replaced in immediate future.

Most of electric and battery locomotives, and mine cars are inoperative condition. They have to be reinforced to resume the production.

Some of fans are super-annuated and may have to be replaced by new ones in progress of mining.

The chair lift is maintained in working condition, though such spare parts as rubbered guide rollers appear to be in short supply and its rope may be required to be replaced in immediate future.

All the compressors are maintained in excellent conditions both in the Bukangama and the Mine Office Stations.

No mechanical problems are seen in the main power substation at the time of our examination.

Machine and tools in the workshop are kept in working condition except for minor hindrances.

There are many motor velicles and heavy equipments in the garage which are in unserviceable condition. They are left to be repaired due to lack of spare parts. Some of them are completely out-of-order and have to be replaced by new ones.

Machines and equipments, which include some problems, are listed in Appendix-2.

The mining operation is under the control of the Mining Department, of which structure is summerized in Fig,10. The personel of the Mining Department is listed in Table 11.

The compressor stations, the main power substation and the operation of the chairlift and the hoisting machines of the shaft, the winzes and the surface inclines belong to the Engineering Depertment, which will be described in a later section.

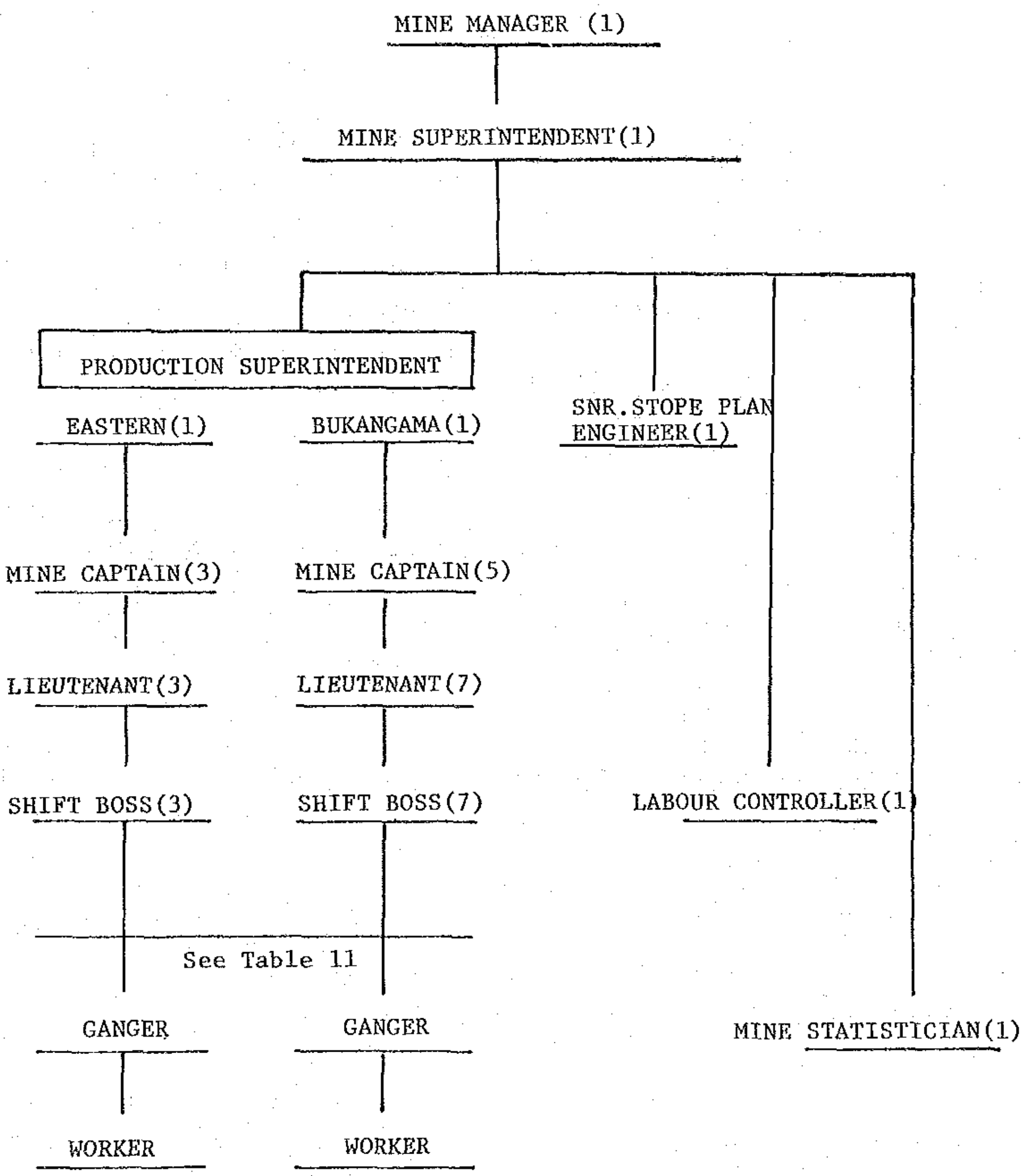
Table 11 Mining Labour Allocation

Feb.1978

	Direct Mining										Administration	Training School	Transport	Total
	Section							Shaft	Surface Engineer	Subtotal				
	1	2	3	4	5A	5B	6							
Ganger I		1								1	4			5
II			2	1		2		1		6		1	2	9
III	5	3	2	6	8	3	6	1	2	36				36
Timber Ganger										0			1	1
Timber Man	6	5	4	4	4	3	5	5		36				36
Driller I	12	9	4	10	7	10	8			60		3		63
II	11	7	3	9	7	9	8			54		6		60
Scraper	14	5	2	6	4	5	7	3		46				46
Loader	3	2	1	4		2	2	1		15		1		16
L.H.D.					3	3	1			7		1		8
Loco. I	2	4	5	9	7	8	8	2		45				45
II	4	2		1		2		2		11				11
III	6	6	5	10	7	9	8	2		53			1	54
Sand Fill					11				8	19				19
Crusher								2		2				2
Pipe & Track	4	6	1	4	2	4	4	1		26				26
Trammer		2	2		1	2	1	1		9				9
Welder	1	1		1	3		1	1	2	10			1	11
Sawyer										0			1	1
Rigger	1	1	1	1	1	1	2			8				8
SNR.B.Man					1			3		4				4
Bell man I								19		19				19
Bell man II				3	2	4		11		20			4	24
OP.Seer										0			3	3
STF.Aider			1	1		1		1		4				4
T.L.						1				1				1
Store Man	2	2	1		2	4	1			12			4	16
Clerk						1				1	2			3
Junior Clerk										0	2		1	3
Typist											1			1
Trainer												17		17
General	29	38	17	40	35	21	33	30	13	256	1	7	8	272
Total	100	94	51	110	105	95	95	86	25	761	10	36	26	833

Section 1 : Lower Eastern below 4000L
2 : Lower Bukangama
3 : Easter & Stream Above 4300L
4 : Buhunga & Numhuga
5A : Upper Bukangama
5B : Middle Bukangama
6 : Eastern 4300-4000L

Fig.10 MINING DEPARTMENT STRUCTURE



1-6 Milling

1-6-1 Outline of Milling Operation

The milling operation comprises three sections; the primary crushing, the secondary crushing and the grinding-flotation sections. The primary and secondary crushing processes are shown in Fig.11 A and B, and the grinding-flotation process in Fig.12 A and B.

Each process is briefly described below.

a) Primary Crushing

Ore from the underground is dumped into a 450-ton trench and subsequently fed on to a grizzly by an electric slusher and a Telsmith feeder. The oversize is crushed by a jaw crusher and fed to a double deck screen together with the under size. The oversize is conveyed to a coarse ore stockpile with a capacity of 10,000 tons.

The undersize is fed to an Akins classifier,the overflow of which is sent to the No.3 thickener. The classified sand is conveyed to fine ore silos.

b) Secondary Crushing

The coarse ore is drawn from the coarse ore stockpile by vibrating feeders to a STD cone crusher. The crushed ore is fed to a single deck screen. The undersize is conveyed to the fine ore silos, and the oversize is crushed by two SH cone crushers and fed to another single deck screen. The undersize of the latter screen is conveyed to the fine ore silos, and the oversize is returned to the two SH cone crushers.

The capacity of the secondary crushing section is about 180 tons per hour.

c) Primary Grinding

The fine ore, which is drawn from the five fine-ore silos (No.1-No.5) by vibrating feeders, is fed to a rod mill (No.4) after weighed by weight meters equipped to conveyor belt systems.

The discharge from the rod mill is devided into two fractions by a distributor, each of which is fed to a different pair of cyclones. The underflow from each pair of the cyclones is separately fed to two different ball mills (No.3 and No.5). The discharge from the both ball mills is returned to the distributor.

FIG. 11-A FLOWSHEET (CRUSHING)

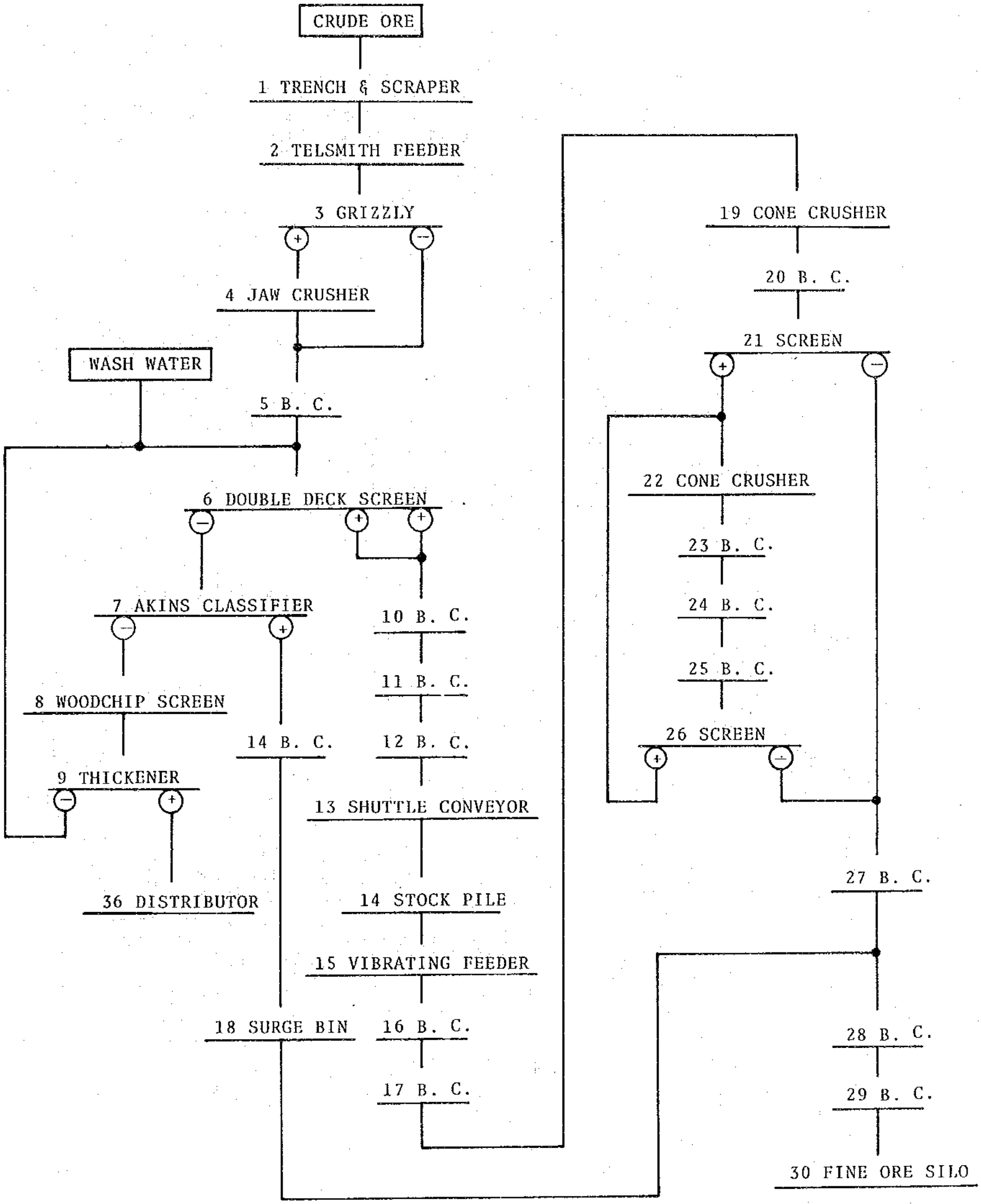


FIG. 11-B ILLUSTRATED FLOWSHEET (CRUSHING)

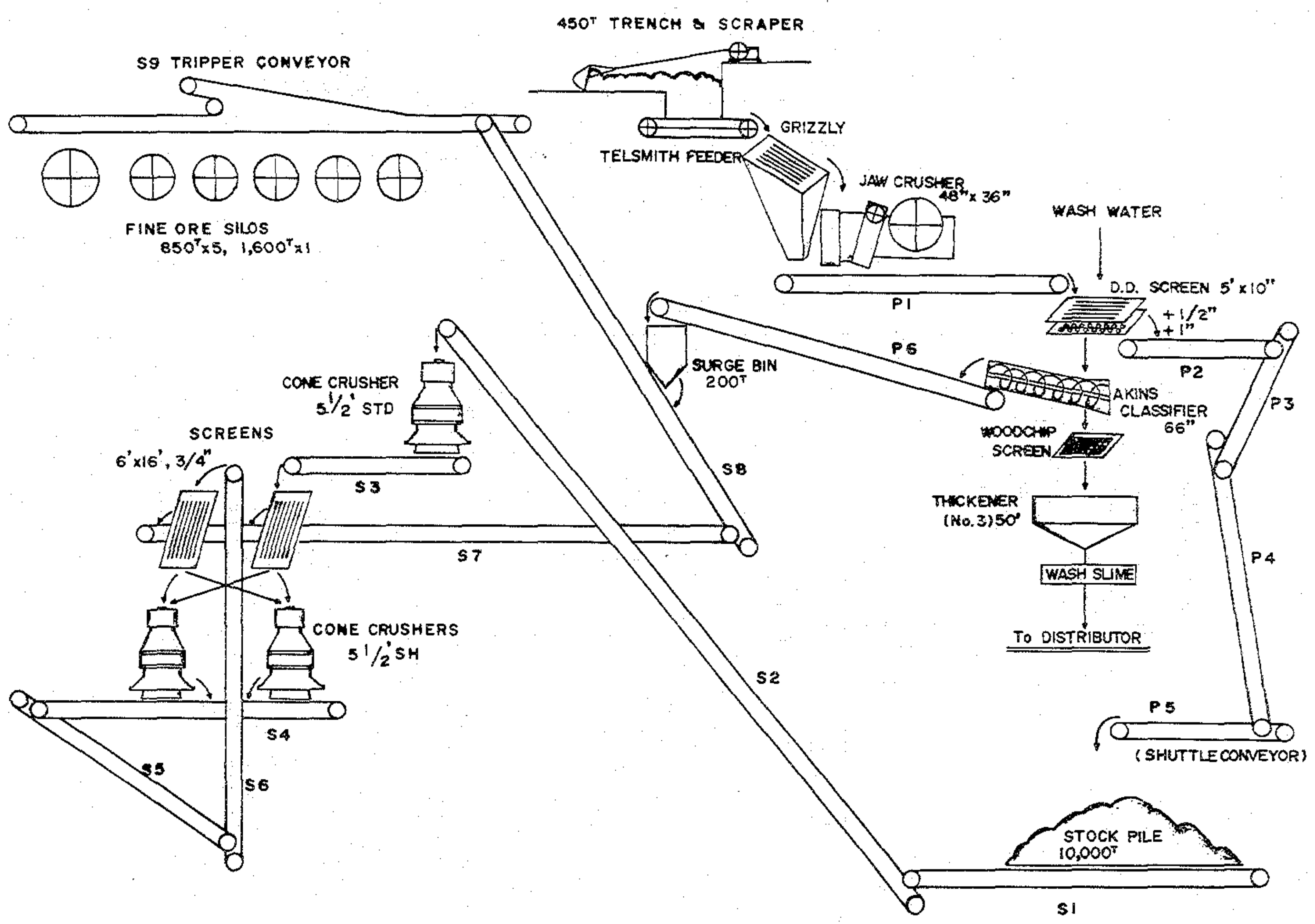


FIG.12-A　F L O W S H E E T (GRINDING & FLOTATION)

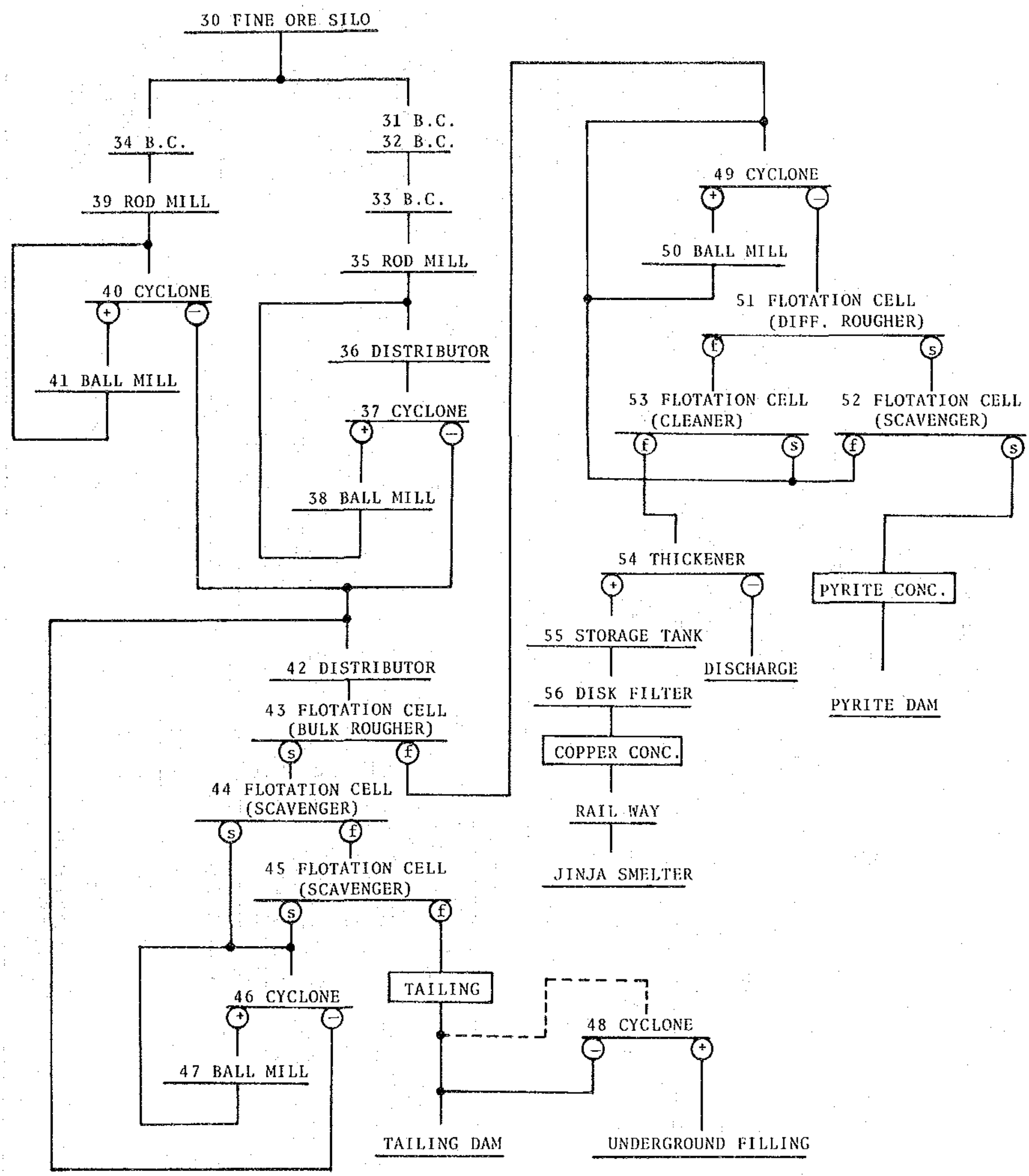
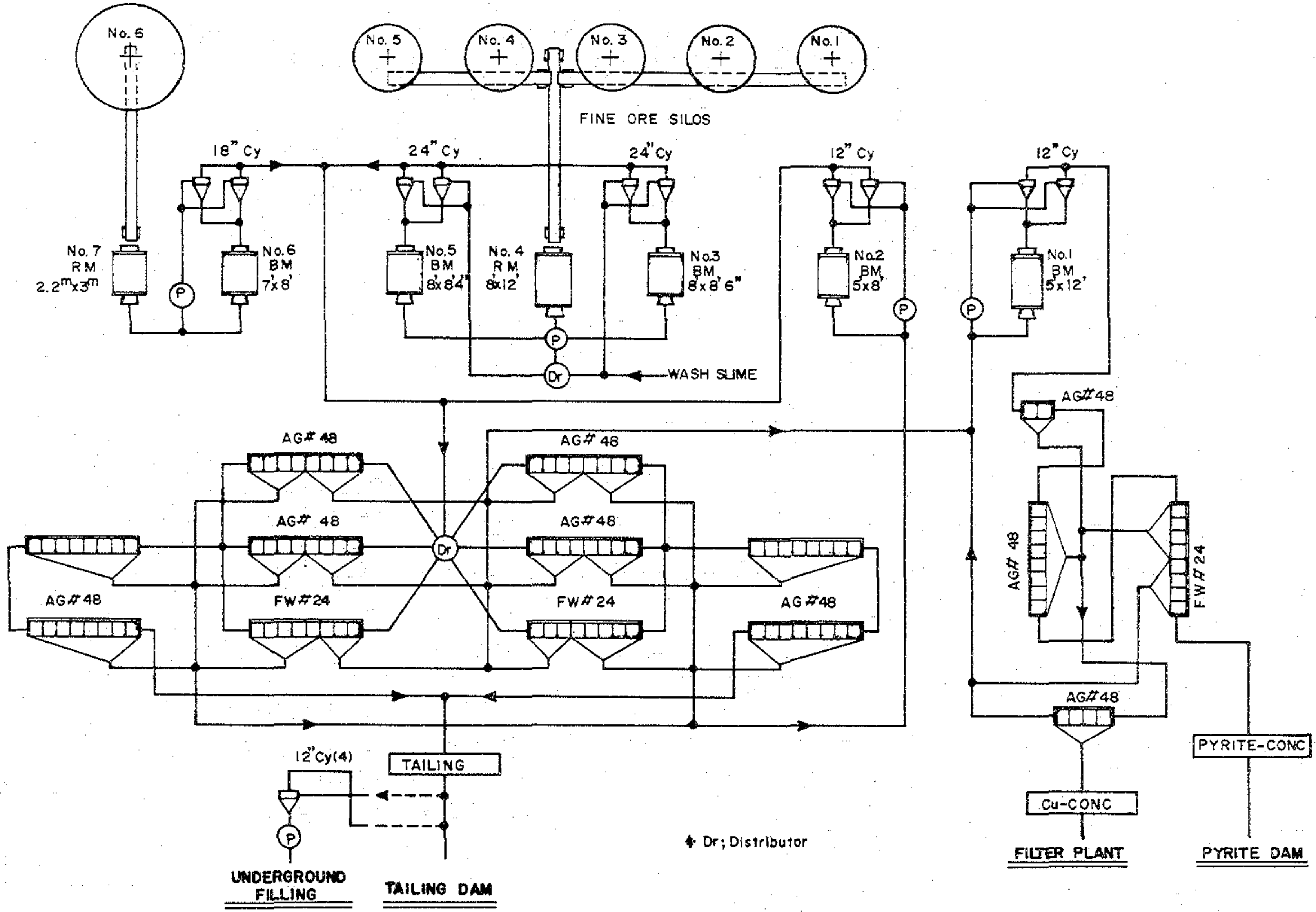


FIG. 12-B ILLUSTRATED FLOWSHEET (GRINDING & FLOTATION)



The fine ore, drawn from the No.6 fine ore silo by a vibrating feeder, is treated in the same manner as explained above in a different system.

The overflow from the cyclones in the above two systems is combined and sent to the flotation section. The fraction minus 200 mesh accounts for about 50% of the overflow.

The wash slime from the primary crushing system joins the primary grinding system at the distributor after the No.4 rod mill.

d) Bulk flotation and Middlings Regrinding

The cyclone overflow from the primary grinding circuit is devided into six fractions by a distributor and fed to 6 banks of flotators. Pulp density of the flotation feed is about 40%.

The froth of the first 3 or 4 cells on each bank forms the bulk concentrate, and is pumped to a cyclone in the secondary grinding circuit.

The rest of the froth forms the middlings concentrate, and is pumped to a cyclone in the middlings regrinding circuit.

The middlings regrinding is a closed circuit consisting of two cyclones and a ball mill. Product of the middling regrinding circuit is repeated to the distributor in the earlier section of the bulk flotation system.

Lime is added to the flotation and the mill feeds to keep pulp pH within a range between 10 and 11. Accordingly, some pyrite is depressed in the bulk flotation stage and disposed in tailing.

e) Secondary grinding and Differential Flotation

The bulk concentrate, is reground to 90% minus 200 mesh by the secondary grinding circuit consisting of two cyclones and a ball mill, and fed to the differential roughers with the adjusted pH within the range of 11 and 12 and with the pulp density of 20 to 25%.

The differential rougher froth goes to the final cleaners flotation where copper concentrate is obtained.

The differential rougher tailing forms pyrite concentrate.

A settling pond and two thickeners are installed to recover the leaked ore.

f) Treatment of Concentrates and Tailings

The copper and pyrite concentrates gravitate a distance of about 12km in two 4-inch pipes to the filter plant at Kasese.

Table-12 Operation Result (Concentrator)

Year	Product	Amount		Cu-Grade %	Cu-Distribution %
		Dry ton	Distribution%		
1968	Mill Feed	926,760	100.00	1.85	100.00
	Cu-Conc.	56,639	6.11	27.69	91.46
	Py-Conc.	70,982	7.66	0.52	2.13
	Tailing	799,139	86.23	0.14	6.41
1969	Mill Feed	979,761	100.00	1.93	100.00
	Cu-Conc.	60,141	6.14	28.86	92.02
	Py-Conc.	60,972	6.22	0.45	1.45
	Tailing	858,648	87.64	0.14	6.53
1970	Mill Feed	1,003,115	100.00	1.91	100.00
	Cu-Conc.	62,128	6.19	28.35	92.12
	Py-Conc.	68,102	6.79	0.31	1.12
	Tailing	872,885	87.02	0.15	6.76
1971	Mill Feed	947,627	100.00	1.80	100.00
	Cu-Conc.	56,010	5.91	28.38	93.09
	Py-Conc.	64,766	6.83	0.27	1.02
	Tailing	826,851	87.26	0.12	5.89
1972	Mill Feed	907,287	100.00	1.73	100.00
	Cu-Conc.	50,716	5.59	28.58	92.54
	Py-Conc.	63,138	6.96	0.26	1.06
	Tailing	793,433	87.45	0.13	6.40
1973	Mill Feed	821,153	100.00	1.90	100.00
	Cu-Conc.	49,839	6.07	29.07	93.05
	Py-Conc.	53,018	6.46	0.30	1.01
	Tailing	718,296	87.47	0.13	5.94
1974	Mill Feed	708,230	100.00	1.75	100.00
	Cu-Conc.	39,686	5.60	28.88	92.40
	Py-Conc.	45,056	6.36	0.30	1.09
	Tailing	623,488	88.04	0.13	6.51
1975	Mill Feed	479,213	100.00	1.88	100.00
	Cu-Conc.	29,215	6.10	28.56	92.78
	Py-Conc.	24,760	5.17	0.43	1.17
	Tailing	425,238	88.73	0.13	6.05
1976	Mill Feed	396,485	100.00	1.79	100.00
	Cu-Conc.	23,670	5.97	27.73	92.29
	Py-Conc.	22,429	5.66	0.35	1.10
	Tailing	350,386	88.37	0.13	6.61
1977	Mill Feed	157,022	100.00	1.76	100.00
	Cu-Conc.	9,361	5.96	27.42	92.69
	Py-Conc.	8,103	5.16	0.30	0.88
	Tailing	139,558	88.88	0.13	6.43

The copper concentrate is placed into thickeners. The overflow of the thickeners is disposed through a settling pond.

The thickened copper concentrates, once stored in two storage tanks, are filtered by two disc filters. The filtered copper concentrates(cake) are conveyed under infra-red heater panels and stockpiled in an 800-ton conical bin. The copper concentrates, containing 7.5 to 8.5 moisture, are drawn from the conical bin into 40-ton railcars via a weight meter, and shipped to the smelter at Jinja.

The pyrite concentrates, which have been produced since the begining of the Kilembe Mine, are stockpiled in an open dam and presumably exceed 1,000,000 tons in amount with grades of 0.2 to 0.4% Cu and 1.0 to 1.6% Co.

Most of tailings gravitate by a pipe to a tailing dam (the E tailing dam) located along the Nyamwamba River, about 4 kilometers down-stream from the mill plant. The E tailing dam, occupying an area of 60,000 square meters, is presently filled up to 3820' level and scheduled up to 3880' level.

A part of tailings are transported to tanks located above the 4500 portal and stored for stope filling.

Locations of the filter plant and the pyrite concentrate dam are shown in Fig.13, and that of the E tailing dam in Dwg.4.

The metallurgical results for the last 10 years are tabulated in Table 12.

1-6-2 Current Situation

The mill operation has been completely ceased since Aug. 1977, due to lack of necessary supply, and only maintainance work is undertaken at the present time.

A number of machine and equipments appear to be super-annuated, damaged or corroded.

In the crushing and grinding sections, many parts of the conveyor belt systems are damaged and necessary to be repalced or repaired. The deferential gear boxes for the No.1 and No.6 ball mills are completely damaged and in inoperative condition. Many pumps are presently working in the underground for draining underground water. The rest of the pumps in the mill plant are unserviceable due to lack of spare parts. Most of flotation cells are highly corroded. The disc filters in the filter plant are supper-annuated and necessary to be replaced.

TABLE 13 LIST OF MACHINE AND EQUIPMENT (CONCENTRATOR)

No.	NAME OF MACHINE AND EQUIPMENT	SPECIFICATION	Nos	MOTOR		REMARKS
				HP	V	
1	TRENCH & SCRAPER	450T	1			
		SCRAPER 56"	1	40	415	
2	TELSMITH FEEDER	48" x 10 1/2'	1			
3	GRIZZLY	4' x 5', 5"	1			
4	JAW CRUSHER	48" x 36"	1	150	3300	
5	BELT CONVEYOR(PI)	750 x 53.4m	1	25	415	
6	DOUBLE DECK SCREEN	5'x10', 1"Bar, 1/2"φ	1	5	"	
7	AKINS CLASSIFIER	66"	1	5	"	
8	WOODCHIP SCREEN		1	5	"	
9	THICKENER (No.3)	50' x 10'	1	2	"	
10	BELT CONVEYOR(P2)	900 x 50m	1	20	"	
11	BELT CONVEYOR(P3)	900 x 54.6m	1	20	"	
12	BELT CONVEYOR(P4)	750 x 54m	1	25	"	
13	SHUTTLE CONVEYOR(P5)	750 x 72m	1	5	"	
14	STOCK PILE	10,000T	1			
15	VIBRATING FEEDER	12' x 4'	2			
16	BELT CONVEYOR(S1)	900 x 156m	1	10	415	
17	BELT CONVEYOR(S2)	750 x 105m	1	35	"	
18	SURGE BIN	18' x 20', 200T	1			
19	CONE CRUSHER	5 1/2' STD	1	200	3300	
20	BELT CONVEYOR(S3)	750 x 45m	1	20	415	
21	SCREEN	6' x 16', 3/4" Bar	1	15	"	
22	CONE CRUSHER	5 1/2' SH	2	200	3300	
23	BELT CONVEYOR(S4)	600 x 54m	1	25	415	
24	BELT CONVEYOR(S5)	750 x 56.4m	1	20	"	
25	BELT CONVEYOR(S6)	750 x 62.7m	1	20	"	
26	SCREEN	6' x 16', 3/4" Bar	1	15	"	
27	BELT CONVEYOR(S7)	750 x 64.5m	1	20	"	
28	BELT CONVEYOR(S8)	600 x 135m	1	35	"	
29	BELT CONVEYOR(S9)	600 x 218.4m	1	35	"	
30	FINE ORE SILO	26' x 32', 850T	5			
		36' x 32', 1600T	1			
31	BELT CONVEYOR	600 x 26m	1	5	415	No.1-3 SILO
32	BELT CONVEYOR	600 x 19m	1	5	"	No.4-5 SILO
33	BELT CONVEYOR	600 x 18m	1	5	"	No.4 ROD MILL
34	BELT CONVEYOR	450 x 18m	1	5	"	No.7 ROD MILL
35	ROD MILL(No.4)	8' x 12' 18 rpm	1	350	3300	
36	DISTRIBUTOR	5'2" x 6'	1			
37	CYCLONE(No.3,5)	24", 2x2	4			
38	BALL MILL(No.3)	8' x 8'6", 21.5rpm	1	350	3300	
	(No.5)	8' x 8'4", 21.5rpm	1	350	3300	Parallel Used
39	ROD MILL(No.7)	2.2m x 3m, 20rpm	1	228	"	
40	CYCLONE(NO.6)	18"	2			
41	BALL MILL(NO.6)	7' x 8', 22rpm	1	147	3300	
42	DISTRIBUTOR					
43	FLOTATION CELL(BULK RGH.)	AG#48x4x4 Banks	16	15x9	415	One motor for 2 cells
		FW#24x3x2 Banks	6	10x2	"	
44	FLOTATION CELL(SCAV.)	AG#48x4x4 Banks	16	15x11	"	
		FW#24x5x2 Banks	10	10x2	"	
45	FLOTATION CELL(SCAV.)	AG#48x16x2 Banks	32	15x16	"	
46	CYCLONE(No.2)	12"	2			
47	BALL MILL(No.2)	5' x8', 28 rpm	1	100	3300	
48	CYCLONE	12x3x4 Sets	12			
49	CYCLONE(No.1)	12"	2			
50	BALL MILL(No.1)	5' x 12', 30 rpm	1	150	415	
51	FLOTATION CELL(DIFF. RGH.	AG#48x10	10	15x3	"	
		FW#24x4	4	10x4	"	
52	FLOTATION CELL(SCAV.)	FW#24	4	15x2	"	
53	FLOTATION CELL(CL.)	AG#48	4	15x2	"	
54	THICKENER	24'x10'	1	2	"	
		40'x10'	1	2	"	
55	STORAGE TANK	14'x14'	2	10	"	
56	DISK FILTER	6'x6	2	15	"	

The present conditions of the machines and equipments are described in Appendix-2.

The consumables such as liners, balls, rods, and reagents are apparently short of supply.

The mill plant, as a whole, appeared to be kept in reasonable condition at the time of our observation. However, it is recommended that special cares, more than ever, be taken to prevent the machines and equipment from corrosion caused by oxidation of ore residue during the suspension of the production.

The major machines and equipments of the mill plant are listed in Table 13.

The scheduled staffs and labour strength for the year 1978 are tabulated in Table 14.

Table 14 Staff and Labour Distribution in the Mill Plant

Shift	1	2	3	Total
Primary Crusher	6	5	5	16
Secondary Crusher	7	6	6	19
Grinding	7	5	5	17
Flotation	6	3	3	12
Filter Plant	5	6	4	15
Metallurgical (Sampling, Assaying & Experiment)	8	2	2	12
Mechanical (Maintenance)	42	1	1	44
Office (Administration) and Shift Supervisers	11	1	1	13
Total	92	29	27	148

Contractors: 8 for Tailing Dam, 3 for Maintainance of Cu & Py Conc. Pipe Lines, 8 for Loading of Cu conc,

A sufficient number of personnel are still kept in the mill plant to resume a normal operation.

1-7 Ancillaries

1-7-1 Mubuku Power Station

The Mubuku Power Station is located approximately 17 kilometers by air due north of the town of Kasese, along the Mubuku River. Water is once introduced to a cleaning pond from the Mubuku River and its tributaries, and gravitates by a galvanized iron flume to a charged pond located on a hill behind the Power Plant for a distance of approximately 4400 meters. From the charged pond, the water is supplied to the generators by a pipe along the slope with an average inclination of 25° for an elevation difference of 180 meters.

Two spare reservoirs are prepared for water supply during the dry seasons; one is located beside the cleaning pond and the other, the Mahoma Lake, approximately 10 kilometers further upstream along the Mubuku River.

The eight generators in the Power Station including spare generators, are maintained in excellent conditions.

The Mubuku Power Station provides 70% of necessary electricity for the Kilembe Mine. The rest of the electricity is supplied from the Owen Falls Hydro-electric Power Station, owned by the Uganda Electric Board (U.E.B.). The electricity is transported to the Nkenda Power Substation, from which transformed electricity is distributed to the Kilembe Mine, the town of Kasese and a cement factory.

The Mubuku Power Station is a branch section of the Engineering Department.

1-7-2 Hima Lime Work

A small limestone quarry is located at Hima, approximately 20 kilometers north-east of the town of Kasese along the major highway (Fig.13). The limestone mined at the quarry, is supplied to the mill plant after roasted in the kilns located at Mukokya, approximately 9 kilometers south-west of the town of Kasese.

The limestone is also shipped to the smelter at Jinja and used as flux for smelting copper concentrates.

The reserves of the limestone are estimated at 1,586,500 ton for the main limestone body and at 591,800 tons for the lower limestone body. These limestone reserves are apparently sufficient to supply both for the milling and the smelting operations.

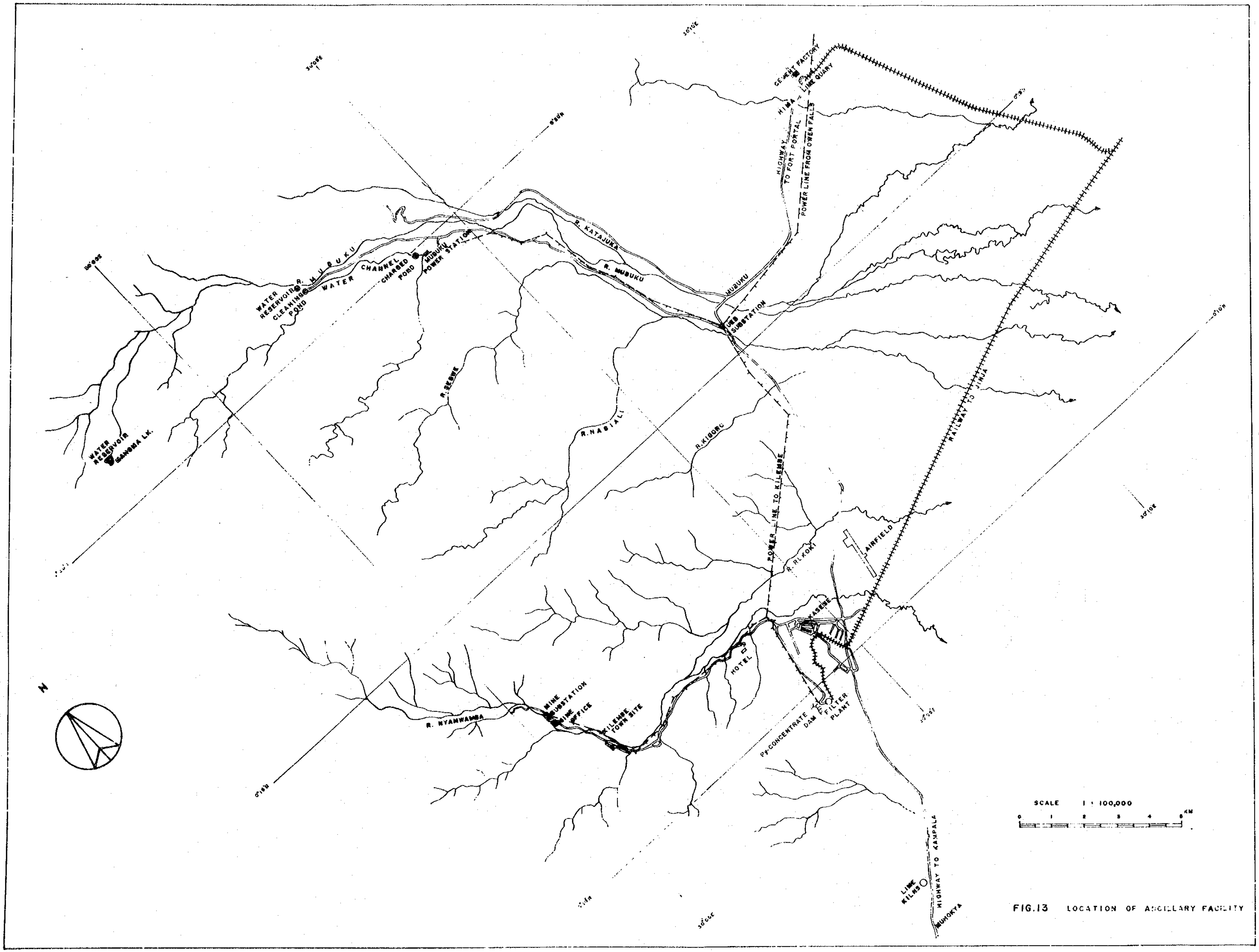
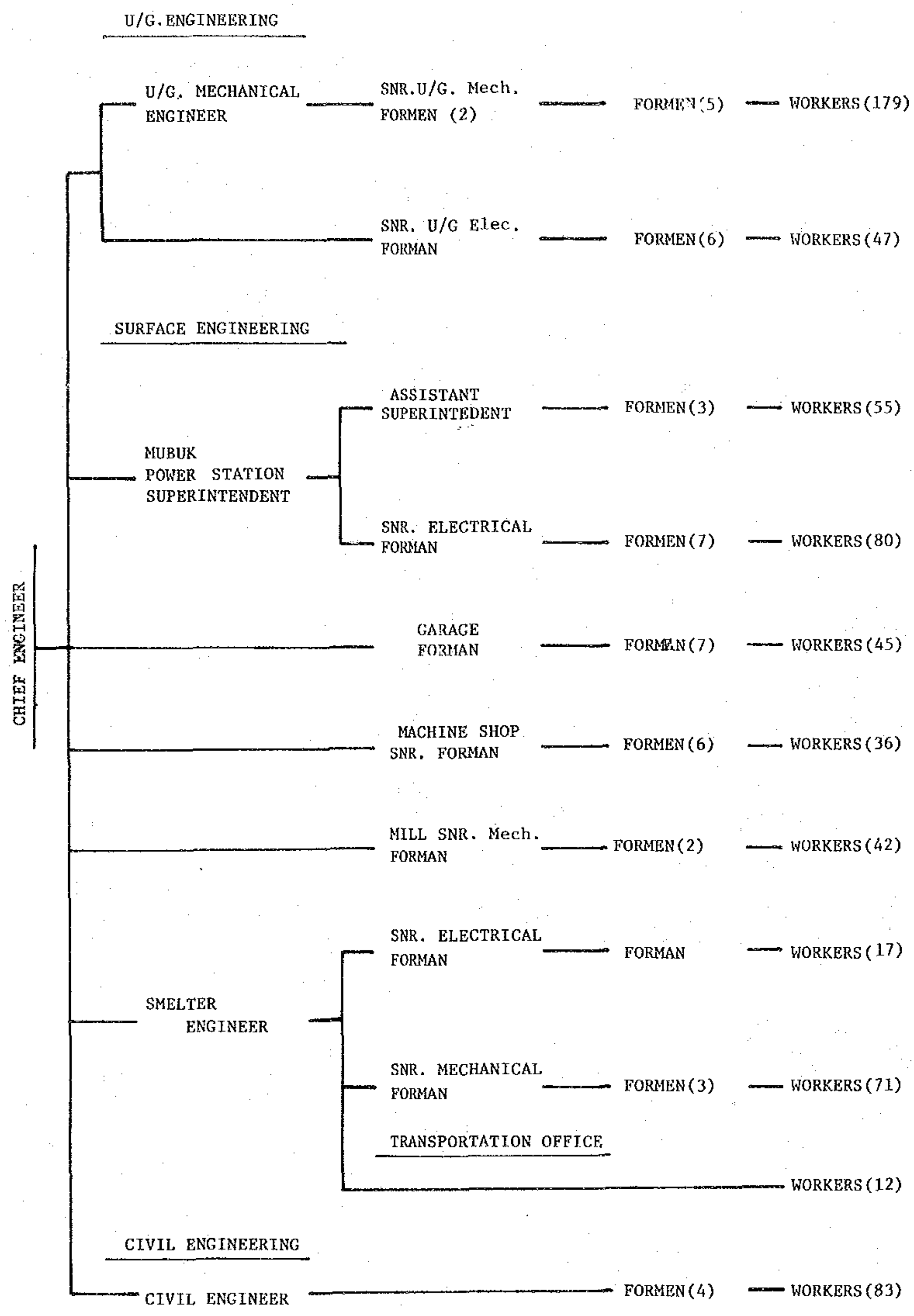


FIG.13 LOCATION OF ANCILLARY FACILITY

FIG. 14 STRUCTURE OF ENGINEERING DEPARTMENT



Total	(4)	(10)	(50)	(667)

( ) : Numbers

1-7-3 Nkombe Saw Mill

A saw mill is located at Nkombe, in the Kalinzu Forest, abont 100 kilometer south of the town of Kasese (Fig.1).

Kilembe Mines Ltd. is leasing an area of approximately 17 square miles at this location, which reserves a sufficient amount of timber.

The saw mill supplies timbers to the Kilembe Mine for underground and other uses.

A D-4 and two D-7 bulldozers, two lorries, and three power saws are equipped for the saw mill operation. The bulldozers and the lorries are in very poor conditions at the present time.

1-8 Administrative Departments and the Overall Oeganization

The Engineering Department is responsible for the mechanical and electrical engineering of the underground, the mill plant, the smelter at Jinja and other surface facilities, including the civil engineering and the operation of the Mubuku Power Plant.

The organization and the labour strength of the Department is shown in Fig.14.

The Technical Department is devided into the mining engineering section and the geological section.

The mining engineering section is responsible for the planning of mining operation, the production control, the surveying in the underground and the mine statistics.

The geological section undertakes the surface and underground exprolation, and the ore reserve estimation.

The Safety and Security Department takes care after the safety of the workers in the underground, the mill plant and the surface areas, and the security guard for all the properties belonging to the Kilembe Mine.

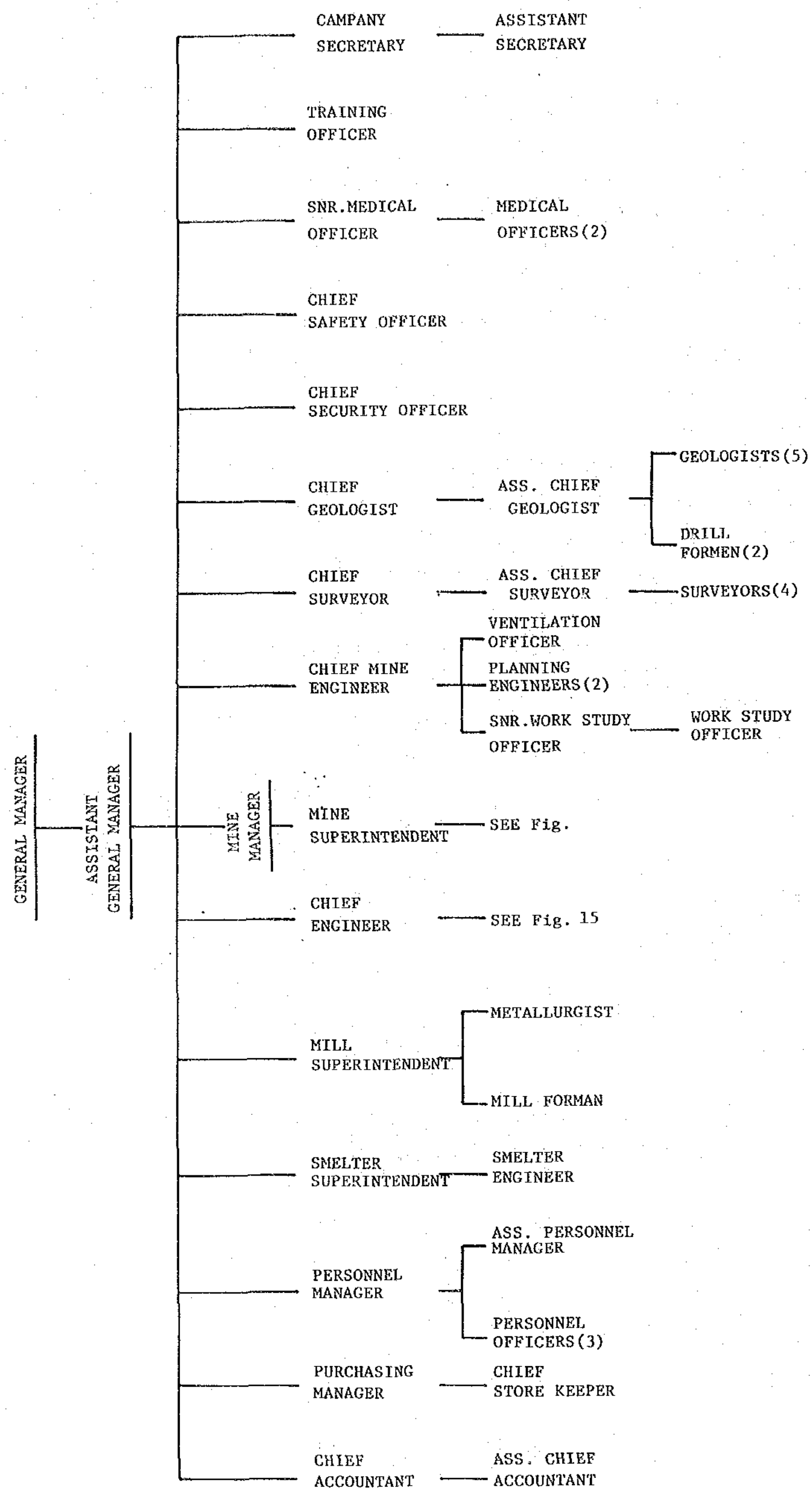
The training system for safety is well established. The workers appear to be highly conscientious for safety. The number of accidents have been kept in satisfactory low level for the last three years. However, caplamps and safety gears seem to be insufficient for the number of the workers at the present time

Other than above described, there are the personnel, the purchasing the medical, and the accounting departments.

These departments are controlling the general activity in the mine and the residential area, including public relations.

The overall organization is summerized in Fig.15. The staff and the labour as of Jan.1978 is tabulated in Table 15.

FIG.15 MANAGEMENT STRUCTURE (KILEMBE MINES LTD.)



( ) Numbers.

Table 15 SUMMARY OF STAFF AND LABOUR EMPLOYED

Year	1978						1977					1976					1975					1974				
Group	ABC	D	E	F	CAS	Total	ABC	D	E	F	Total	ABC	D	E	F	Total	ABC	D	E	F	Total	ABC	D	E	F	Total
Expatriates	8	-	-	-	-	8	8	-	-	-	8	8	-	-	-	8	28	-	-	-	28	54	-	-	-	54
Residents of E.A.Africa of Poreign Origin	5	2	-	-	-	7	5	2	-	-	7	6	2	-	-	8	8	3	-	-	11	9	2	-	-	11
Indigenous of East Africa	154	278	751	1520	171	2874	158	280	761	1570	2769	163	283	843	1624	2914	118	280	927	1782	3107	95	296	930	1813	3134
TOTAL	167	280	751	1520	171	2889	171	282	761	1570	2784	177	285	843	1624	2930	154	283	927	1782	3146	158	298	930	1813	3787
Mining	26	37	245	672	12	992	27	37	252	685	999	27	37	315	869	1248	28	45	329	999	1401	43	55	366	1081	1545
Concentrator	8	16	22	78	-	124	8	16	23	79	126	9	21	20	78	128	10	18	26	86	140	8	18	24	86	136
Technical	24	25	38	44	8	139	24	25	38	45	132	30	24	40	54	148	29	18	48	51	146	18	33	58	63	172
Mechnical & Construction	14	38	58	88	-	198	14	40	61	89	204	14	41	67	72	194	11	43	83	67	204	8	38	91	81	218
Underground Subsidiary	9	19	69	97	-	194	11	17	70	100	198	11	17	73	110	211	9	16	80	97	202	10	13	81	82	186
Eleotrical	16	29	88	90	12	235	18	31	86	91	226	18	41	62	76	197	13	42	70	81	206	16	36	72	68	192
Porsonnel	7	15	15	19	13	69	7	15	14	18	54	8	17	15	18	58	7	17	14	14	52	5	19	21	16	61
Servioes (Schools,Medical, Sinitation)	22	26	37	21	2	108	21	27	37	21	106	22	14	42	19	97	15	17	38	21	91	18	14	39	19	90
Training (Surface)	2	4	21	1	-	28	2	4	21	1	23	4	2	47	1	54	5	2	63	3	73	6	3	23	30	62
Administration/Mine Stores	16	28	14	27	-	85	16	28	15	26	85	14	31	14	32	91	14	24	24	32	94	9	24	8	6	47
Safety & Security	4	9	26	90	7	136	4	9	26	92	131	3	7	35	88	133	2	7	20	93	122	2	6	18	63	89
Smelter	16	28	104	148	38	334	16	27	104	156	303	15	27	110	178	330	10	29	126	213	378	10	26	113	179	328
Self-Supporting and Subsidised	1	4	1	16	-	22	1	4	1	16	22	2	6	3	29	40	1	5	7	25	38	2	6	5	18	31
Hiima Lime Works	-	1	5	35	17	58	-	1	5	38	44	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Nkombe Saw Mills	2	1	8	94	62	167	2	1	8	115	126	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

1-9 Requirements For Resuming Production

It is assumed that the production be resumed at a rate of 50,000 tons per month on a crude ore basis. The price estimations for the required machines equipments and other materials are quoted from Japanese manufacturers or suppliers on the Japanese yen basis. The Japanese yen values are converted to the Uganda Shillings and the U.S. Dollars by using the exchange rates of 224 Japanese yens and 7.93 Uganda Shillings for one U.S. Dollar, as of May 10, 1978.

1-9-1 Capital Requirements

For the mining operation, major items which are required to be reinforced, are locomotives, mine cars, dump loaders, scrapers, slashers, rock drills and pumps. Others are spare motors, parts and consumables necessary for repairing machines and equipments. The locomotives, mine cars and their spare parts account for more than a half of the total capital expenditure required for the mining operation.

For the milling operation, it is necessary to renew a number of pumps in the various sections of the mill plant,and two disc filters in the filter plant. A great length of conveyor belt is required for replacement and spare. A large amount of consumables, such as balls, rods, liners and are reagents provisioned to start the production.

In the Engineering Department, bulldozers and service trucks are necessary to be replaced by new ones. Some passenger cars and pick-up trucks are also required to be reinforced.

The capital requirements for the mining and the milling operations are summerized in Table 16 and tabulated in detail in Appendix 3.

The contingincy is assumed at 7% of the total expenditure for the capital requirements.

It is assumed that the present staffs and workers are capable to install new machines and equipments. Therefore, no cost is allocated for the installation and the expertirates concerned.

TABLE 16 SUMMARY OF CAPITAL REQUIREMENTS

SECTION	MAJOR ITEMS	AMOUNT		
		YEN (1000)	U.SH. (1000)	US.$ (1000)
HOIST & CHAIR LIFT	ROPES, SPARE PARTS	10,350	365	46
ROCK DRILL	ROCK DRILLS, AIR LEGS	54,960	1,946	245
U/G. CRUSHER	LINERS, BEARINGS	10,500	372	47
SCRAPER	SLASHERS, BUCKETS	54,210	1,920	242
DUMP LAORDER	LAORDERS, SPARE TIRES	88,020	3,117	393
MINE CAR	MINE CARS, WHEELS, BEARINGS	270,530	9,580	1,208
LOCOMOTIVE	LOCOMOTIVES, BUTTERIES CHARGERS	252,660	8,947	1,128
PUMP	TURBINE & SLURRY PUMPS, SPARE MOTORS	67,320	2,384	301
FAN	FANS, CASINGS, BEARINGS	22,100	783	99
COMPRESSOR	VALVES, AFTERCOOLERS, V-BELTS	9,500	336	42
MISCELLANEOUS	WELDING MACHINES MINE SAFETY LAMP	23,700	841	106
SUBTOTAL		863,850	30,591	3,857
PRIMARY CRUSHING	PUMPS, CONVEYOR BELTS, LINERS	18,060	640	80
SECONDARY CRUSHING	CONVEYOR BELTS, LINERS, BEARINGS	31,100	1,101	139
GRINDING	DEF. GEAR BOX, PUMPS LINERS, BALLS, RODS	39,900	1,412	179
FLOTATION	IMPELLERS, STABILIZERS, LINERS DEF. GEAR BOX, PUMPS	45,870	1,624	204
LIME GRINDING	PUMPS, LINERS	1,300	47	5
FILTERING	DISC FILTERS, PUMPS, RAKES	55,230	1,955	247
SUBTOTAL		191,460	6,779	854
GARAGE	BULLDOZERS, LORRIES, PICK-UP TRUCKS, PASSENGER CARS	143,230	5,071	638
WORKSHOP	HORIZONTAL BORING MACHINE, MILLING MACHINE, LATHE	72,100	2,553	321
SUBTOTAL		215,330	7,624	959
TOTAL		1,270,640	44,994	5,670
CONTINGENCY	(7%)	88,860	3,206	300
GRAND TOTAL		1,359,500	48,200	6,070

The ocean freight estimate is quoted from a Japanese ocean freight company as follows.

Port of Departure : Yokohama

Port of Arrival : Mombasa

a) Package Weight : Less than 4 tons

   Base Rate US.$ 112.85

   Banker Surcharge 14.8%

   Currency Surcharge 31.5%

   Packing 5%

b) Package Weight : 4 tons and above; Heavy lift charge is added as shown in Table 17.

Table 17 Hearvy Lift Charge

Package Weight(Tons)	Heavy Lift Charge (US.$/ton)
4 - 5	18.00
5 - 6	27.00
6 - 8	36.00
8 - 10	45.00
10 - 12	49.45
12 - 15	54.00
15 - 18	58.45
18 - 21	63.05
21 - 25	67.50
25 - 30	71.95
30 - 35	76.45
35 over	76.45

A total weight of machines, equipments and other materials is estimated at 895.3 tons, of which 577.3 tons consists of those to be assembled into packages less than 4 tons in weight.

The ocean freight for the packages less than 4 tons in weight is calculated as follows.

577.3 x 112.85 x 1.05 x (1.148+0.315) = US.$ 100,078

Table 18 List of Heavy Lift Packages

Items	Weight Ton	No.s	Freight/Package	Total Freight
Chair Lift Rope	14.0	1	3,588	3,588
Dump Loader	7.0	4	1,601	6,404
Gramby Car	6.0	10	1,372	13,720
Locomotive	8.0	3	1,940	5,820
Locomotive	4.0	2	804	1,608
Locomotive	8.0	4	1,940	7,760
Locomotive	6.0	6	1,372	8,232
Disk Filter	5.0	2	1,074	2,148
Truck Crane	4.0	2	1,074	2,148
Truch Crane	4.0	1	804	804
Crane Car	20.0	1	5,404	5,404
Boring Machine	5.0	1	1,074	1,074
Turning Lathe	5.0	1	1,074	1,074
				49,784

The heavy machines and equipments, each package of which weighs at 4 tons or more, are listed in Table 18 and their ocean freights are estimated item by item.

The total ocean freight is estimated at us.$ 149,862 between Yokohama and Mombasa.

An ocean freight insurance estimate is also quoted from a Japanese insurance company. The ocean freight insurance is estimated at US.$ 28,727 for all the cargos afore-mentioned between Yokohama and Mombasa.

A inland freight rate of U.Sh.415.5 or US.$ 52.38 per ton is provided by the Purchasing Department of Kilembe Mines Ltd. on the basis of the recent actual data. The total inland freight is estimated at US.$46,898.

A sum of US.$42,513 is assumed for other expenses in regard to the transportation, such as the inland freight insurance, handling charge and so on.

The over all transportation cost is summerized in Table 19.

Table 19 Summary of Freight Cost.

	US.$
Ocean Freight	149,862
Insurance For Ocean Freight	28,727
Inland Freight(Mombasa-Kasese)	46,898
Others	42,513
Total	268,000

Two to seven months must be allowed for delivery in addition to the duration of the ocean and inland transportation.

A sum of US.$ 360,000 is allocated for the general expense, which will cover service charges for the installation of heavy machines and equipments and other expenses in regard to purchasing.

The summary of the capital expenditure in the Kilembe Mine is shown in Table 20.

Table 20 Summary of Capital Expenditure (Kilembe Mine)

	1000yen	1000U.Sh.	1000US.$
Mining	863,850	30,591	3,857
Milling	191,460	6,779	854
Engineering	215,330	7,624	959
Sub total	1,270,640	44,994	5,670
Contingency	88,860	3,206	398
General Expense	80,500	2,850	360
Freight	60,000	2,125	268
Total	1,500,000	53,116	6,696

1-9-2 Operating Requirements

The present mining method seems to be only a suitable one in this mine as afore-mentioned (1-5-2). However, productivity of each stope appears to be too low to keep an effective operation. It is recommended that the productivity per stope be maintained at a rate of 1000 tons a month or more.

The ore-grade control is another important factor to perform a profitable operation. It is required for the supervisors to watch the change of ore grades in stopes at times, not only relying on dilling results, and to give workers adequate instructions for minimizing dilution.

For the milling operetion, the last 10 years' record(Table 12) indicates satisfactory results in mill recovery and concentrate grade. The expected mill recovery and concentrate grade, when the production is resumed, are estimated on the basis of the last 10 years record as shown in Table 21.

The monthly operating cost is estimated on the basis of the actual material consumption for the last few years with some modifications, and is tabulated in detail in Apendix 4.

The staffs and labours as of Jan,1978. (Table 15) is assumed to be kept for the operation, and is summerized in Table 22. It could be possible to decrease the number of the staffs and labours in performing the production of 50,000 tons per month. At the present time, however, no information in regard to productivity is available to estimate the optimum number of the staffs and labours.

Table 21 Estimated Metallurgical Result(monthly)

Mill Head Grade	Product	Tonnage (wet ton)	Moisture (%)	Distribution dry ton	Distribution %	Cu-Grade (%)	Cu-Distribution (%)
1.94	Crude Ore			50,000	100.00	1.94	100.00
	Copper Conc.	3,451	8.5	3,158	6.32	28.50	92.80
	Pyrite Conc.			3,347	6.69	0.35	1.21
	Tailing			43,495	86.99	0.134	5.99
1.80	Crude Ore			50,000	100.00	1.80	100.00
	Copper Conc.	3,192	8.5	2,921	5.84	28.50	92.50
	Pyrite Conc.			3,096	6.19	0.35	1.20
	Tailing			43,983	87.97	0.130	6.30
1.69	Crude Ore			50,000	100.00	1.69	100.00
	Copper Conc.	2,988	8.5	2,734	5.47	28.50	92.20
	Pyrite Conc.			2,898	5.80	0.35	1.20
	Tailing			44,368	88.73	0.126	6.60

TABLE 22 SUMMARY OF STAFF AND LABOUR STRENGTH

GROUP		ABC	D	E.F	CASUAL	TOTAL
MINING	DEVELOPMENT			176		176
	DRILLING		2	22		24
	STOPING			612	42	654
	HOIST & TRAMMING			400		400
	ADMINISTRATION	26	37	18		81
CONCENTRATOR		8	16	100		124
ENGINEERING	MECHANICAL & CONSTRUCTION	14	38	117		169
	UNDERGROUND SUBSIDIARY	9	19	113		141
	ELECTRICAL	16	29	110		155
TECHNICAL SERVICE		30	36	129	15	210
ADMINISTRATION		16	28	21		65
PERSONNELS		7	15	24	13	59
SERVICES		22	26	30	2	80
SELF SUPPORTING & SUBSIDISED		1	4	10		15
HIMA LIME WORKS			1	30	17	48
NKOMBE SAW MILL		2	1	92	59	154
TOTAL		151	252	2,004	148	2,555

Table 23 MONTHLY OPERATING COST(KILEMBE MINE)

		WAGE		*SUPPLY & OTHERS				TOTAL	
				DOMESTIC		IMPORT			
		U.Sh. (1000)	US.$	U.Sh. (1000)	US.$	U.Sh (1000)	US.$	U.Sh (1000)	US.$
MINING	PROSPECTING					33.3	4,203	33.3	4,203
	DEVELOPEMENT	159.4	20,096	39.2	4,947	174.2	21,964	372.8	47,007
	DRILLING	23.3	2,940			45.6	5,749	68.9	8,689
	STOPING	592.4	74,705	111.7	14,090	488.5	61,601	1,192.6	150,396
	HOISTING & TRAMMING	362.2	45,676	25.3	3,185	48.9	6,167	436.4	55,028
	MINE ADMINISTRATION	207.8	26,198	47.4	5,981	16.9	2,133	272.1	34,312
	TECHNICAL SERVICES	345.9	43,613	11.5	1,452	31.0	3,908	388.4	48,973
ENGINEERING		695.8	87,748			492.9	62,153	1,188.7	149,901
ANCILLARIES		177.1	22,327	72.6	9,155	55.2	6,957	304.9	38,439
MILLING		150.2	18,936	25.0	3,157	542.8	68,452	718.0	90,545
ADMINISTRATION		422.7	53,308	133.8	16,878	215.0	27,113	771.5	97,299
SUBTOTAL		3,136.8	395,547	466.5	58,845	2,144.3	270,400	5,747.6	724,792
CONC. TRANSORTATION (KASESE-JINJA)				307.9	38,830			307.9	38,830
BANK INTEREST (8% ANN.)				49.3	6,219			49.3	6,219
PREVISION FOR APPROPRIATION				95.3	12,017			95.3	12,017
TOTAL		3,136.8	395,547	919.0	115,911	2,144.3	270,400	6,200.1	781,858

* DOMESTIC: Including Electricity Transportation Cost, Legal Fees etc.
Imported: Material Supply Only From Foreign Countries

Their salaries and wages are estimated on the actual basis during the period between Jan. and Aug. 1977. An average working-day is assumed to be 21 days a month or 252 days a year, after excluding 9 national holidays.

The monthly operating cost is summerized in Table 23.

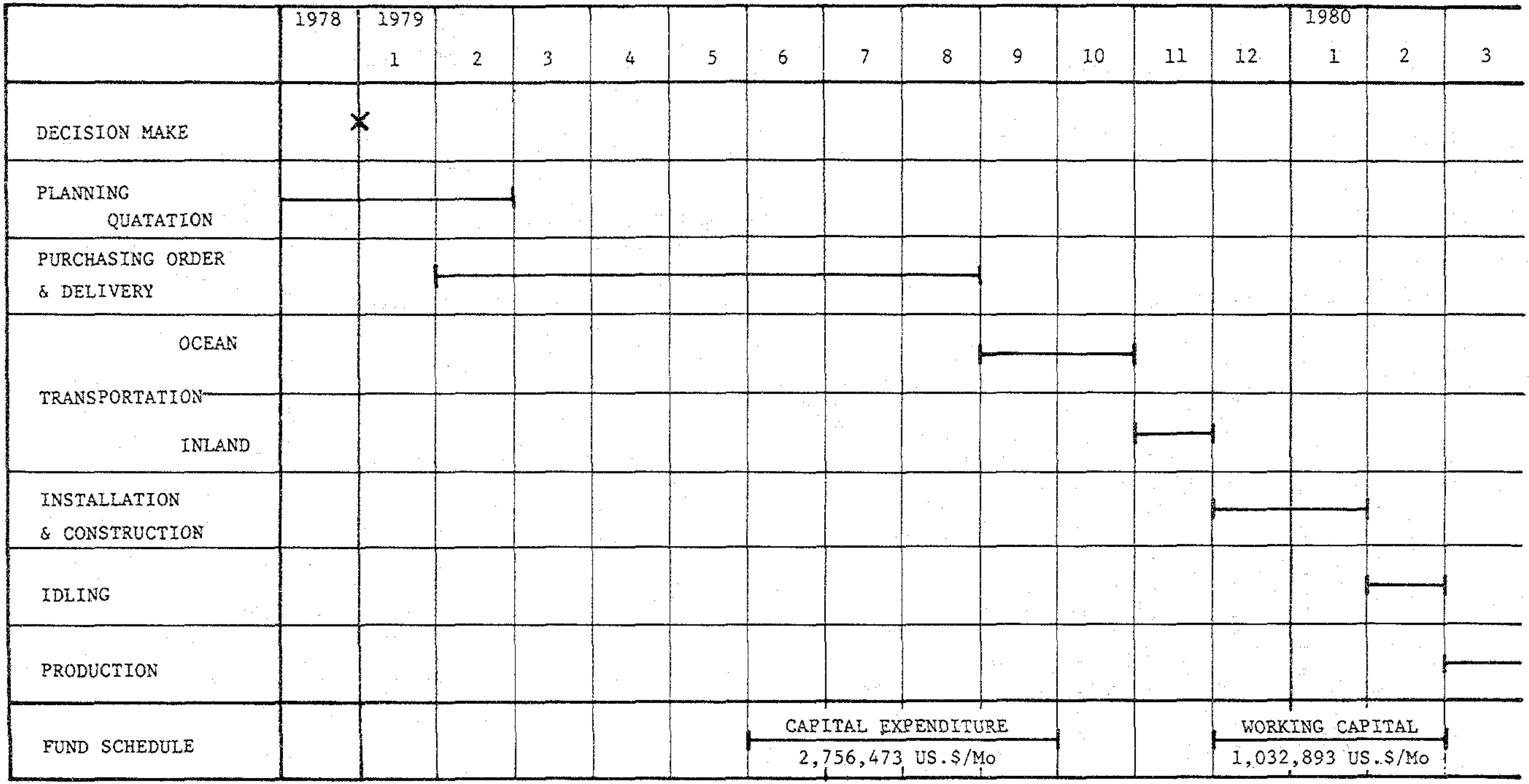
1-9-3 Schedule for Resuming Production

The schedule to start the production is shown in Fig.16.

It is assumed that the decision to go ahead be made on Jan.1st,1979.

A total of 14 months will be required before starting the production, including the idling period.

Fig.16 PREPRODUCTION SCHEDULE



PART-2 JINJA SMELTER

2-1 General

As the Republic of Uganda is landlocked, supply of materials from overseas countries is difficult and costly. Overseas materials must first be shipped to Mombasa, Kenya, from where they are transported by what was once the East African Railway. Urgently needed materials must be freighted directly by air.

At present, presumably on account of the limited funds, minor replacement parts such as electrode paste and electrode steel plates, are imported by air, aboard Uganda Airlines places, in small lots.

To secure smooth and continuous operation, a proper stock of spare parts and supplies would seem to be indispensable, but actually, stocks have depleted almost completely, including such standard items as bolts,nuts,and washers.

In addition, the obsoleteness of all the equipment is aggravating the situation, as replacement parts for outmoded units are mostly not available anymore on the market, and must be specially ordered at extra cost and time.

Under these adverse circumstances making a full restoration of the plant under study will be extremely difficult. The following two basic requirements must first of all satisfied before any feasible measures can be contemplaced:

A. Investment of funds sufficient to procure spare parts and supplies for at least one year.

B. Replacement of the existing outmoded equipment and parts with products commonly available on the market. The very fact that at present all stocks have been consumed presents a particularly good opportunity to do so.

With regard to the operation routine of the workers, there are many drastic improvements required.

For example, expensive ore is being handled very carelessly. Proper operation standards must be worked out in such operations as ore transportation, furnace charging, electrode manipulation, tapping, converter operation, and copper casting, and the workers must be trained patiently. Only after such training has been successfully given, can daily operation schedules may be formulated and the smelting speed, tapping interval, converter blowing time, and other operating parameters determined, according to these schedules.

If these measures are not carried out, improvement in copper yield ratio, and reduction of copper loss due to lost ore and copper content inclusion in slag cannot be expected, and stable production and quality will be impossible. At present, semi-finished product about 25,000 tons is piled on the premises.

The greatest factor contributing to this unproduct accumulation of semi -finished products is the un-controlled random operation performed from day to day.

With respect to production equipment, among the greatest are to be found in the smelting equipment, the control area of the present plant.

Of the electric furnace equipment the smelter proper, the charging device, the electrode device, and gas handling equipment must first of all be either redesigned or replaced.

The present equipment is prone to explosions inside the furnace, short-circuiting incidents in various parts of the electrode circuits, water leaks from the holders, deterioration of the working environment, and production drop.

Also the furnace body with the brick exposed unprotected by outer steel shells, without any cooling provision, cannot withstand further use without causing a fatal eruption of molten contents.

For these reasons, it is recommended that this opportunity for the total recommended of the vital component in the plant, the electric furnace, be taken.

Although the copper mine in Kilembe is said to be capable of yielding ore at a rate of 50,000 m.tons per month (Cu%=1.75%) for at least 7 years to come, operation has been suspended for several months in both the mining and milling operations due to the some shortage of funds as the smelting plant.

Under these circumstances, it is not only meaningless but also impossible to continue the smelting operation alone, and the suspension of the entire smelting plant operation until full restoration of the mine operation, the full repair and replacement of the plant facilities in the meantime, and the cashing of the stocked ore and semi-finished products are strongly recommended.

Fortunately, in the present plant, the electric furnace smelting system, the simplest and certainly the most appropriate method for Uganda, has been adopted, so that the primary targets of improvement of copper yield rate and achievement of smooth and stable operation can be attained quite simply, if improvement of the material blending equipment, re-building of the electric furnace (or its replacement), and re-education of the engineers are successfully performed.

At present, the following major materials are improted:

(1) Electrode paste: yearly consumption: 200 m tons, imported from Norway
(2) Steel materials: imported from England and Germany
(3) Conveyor belts: imported from England
(4) Bricks: imported from England
(5) Pneumatic hammers: (for tapping the converter and for crushing semi-products): made in England and the U.S.
(6) Cement: imported from England
(7) Motors: imported from England
(8) Main cranes: made in England
(9) Blowers and compressors: made in England
(10) Diesel locomotives: made in England
(11) Pumpa and crushers: made in England
(12) Wheel loaders: made in the U.S. and England
(13) Machine tools: made in England
(14) Automobiles and trucks: made in Japan

Flux, clay, timber, oxygen and some other materials are procured domestically.

They are consumed at the following rates:

Lime stone: 150 m tons/month

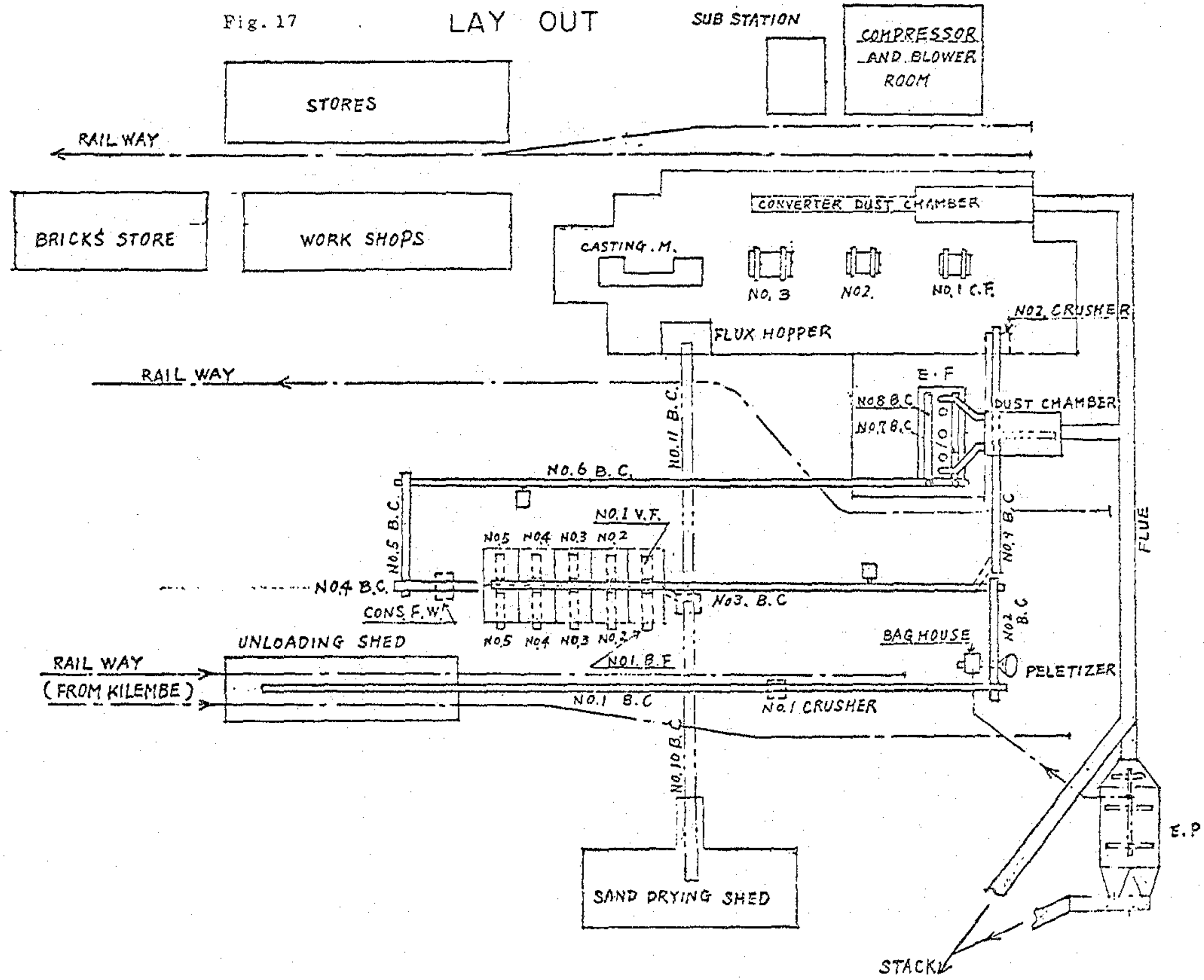
Silica sand: 600 m tons/month

Clay: 30 m tons/month

Timber: 150 m^3/month

Oxygen: 500 Nm^3/month.

Fig. 17 LAY OUT



2-2 Plant Survey Results

2-2-1, Acceptance of ore, flux and other materials

The quantity and transportation routes of these materials produced domestically or from overseas sources are given separately. Although no major defect was noticed in the ore transportation system up to the unloading and storage, spilling of ore from the conveyor or other facilities was rather excessive.

(1) In unloading ore from railway wagons, in addition to simple improvements in the facilities, the attitude of careful operation eliminating the possibility of ore spilling, and the custom of picking up all the spilled ores and returning them to the conveyor, after the completion of an unloading process, must be fully inculcated in the workers.

(2) The belt conveyor speed seemed to be a little too high. Especially for the conveyors installed at large angles, the violent vertical vibration of the belt resulting form the high speed caused the ore to fall off.

(3) Care most also be taken to eliminate ore spill caused by the side-shifting of the belt, and at the transfer points of the conveyors. Although at present, limestone is delivered on railway wagons and trucks in large lumps, and crushed manually, it must be crushed by proper crushing machines to grain sizes less than 30 mm when the proper preparation method is introduced in the future. The proposed crushing system is shown in Fig. 18.

2-2-2 Blending Preparation

In any smelting system, the preparation of ore, flux and other materials is the most important factor for improving the smelting efficiency and the copper yield ratio.

At present, (1) limestone is loaded manually on the conveyor belt with shovels, (2) no analysis is made of the CaO and SiO_2 contents in the charge and in the electric furnace slag, and (3) no weighing instruments are used for the materials.

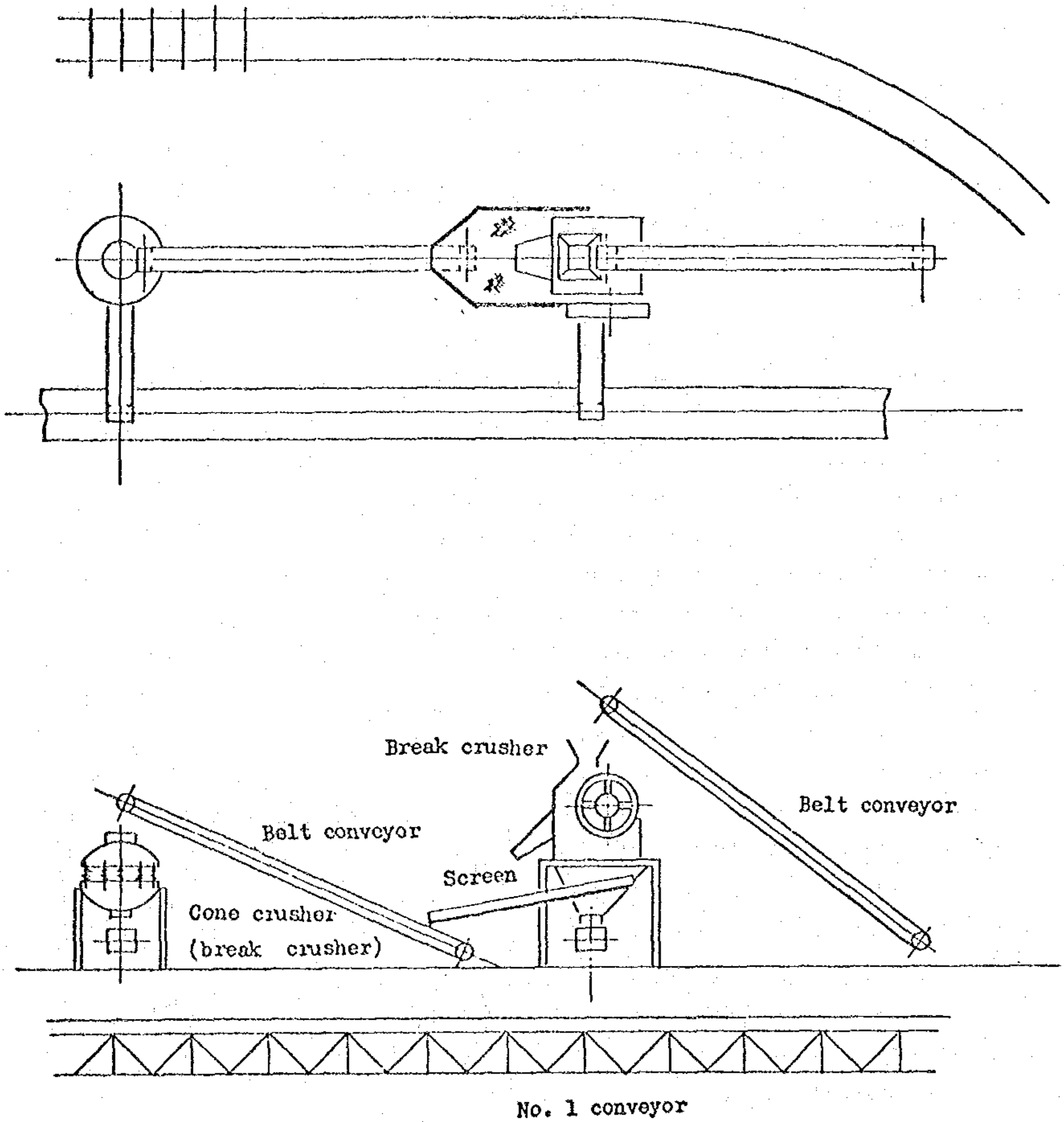


Fig. 18

Proposed Crushing System

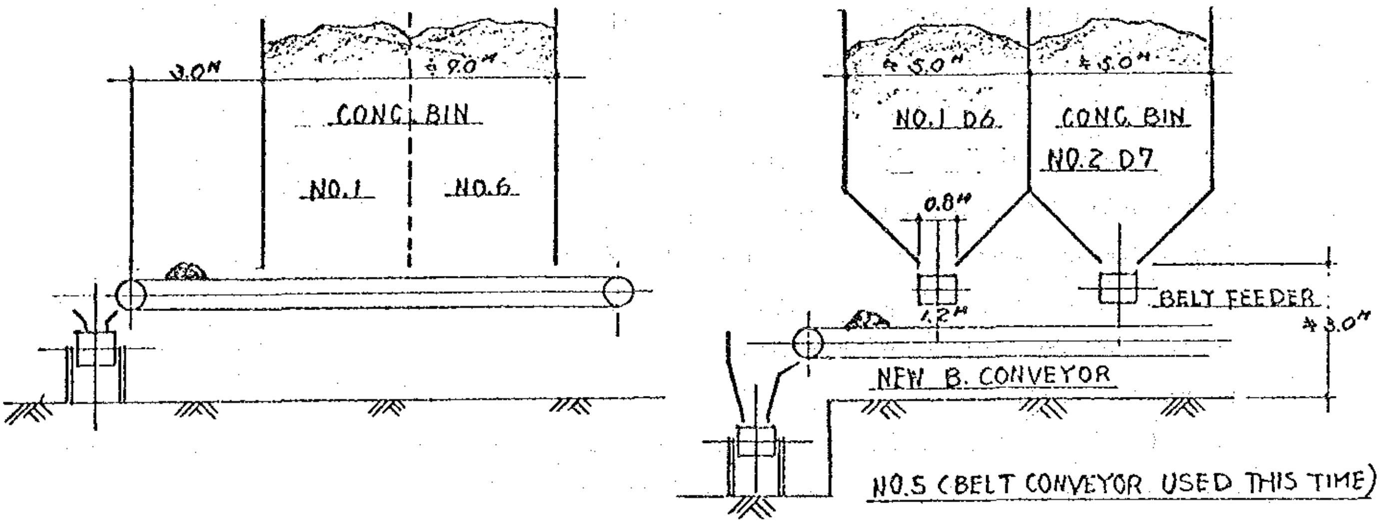
However, in the event that the mostly problematic electric furnace will be improved and its operation become stables, needless to say, the present material preparation facility must be improved. The main problems requiring consideration in this connection are as follows;

(1) The hourly ore delivery rate from the storage bin must be maintained at a constant level. The present ore storage bin is structurally extremely unsuitable for continuous discharge. The width of the ore discharge opening (30-35 cm at present) must be widened, and to eliminate ore hanging, the inclined portion (45° at present) must be made steeper. The belt feeder installed underneath the bin must be enlarged, and its speed must be reduced to below 4 m/min.

(2) The ratio of limestone to ore must be maintained at a definite level, and to achieve this, the charge facility, including the crushing facility, must be improved. Limestone must be crushed to 30 mm or smaller pieces, and a limestone bin provided with a constant discharge rate delivery device must be employed.

In any case, the ore and limestone must be mixed at a precise ratio by some means such as a feeder speed control or gate opening control (Fig.19). The revert matte normally contains white matte and crude copper pieces which are detrimental to crushers. Either the white matte and crude copper pieces should be manually separated before sending the revert slag to the crusher or else it should be crushed by means of dump weights or concrete breakers. However, even with these means, the revert slag matte cannot be crushed into small pieces, and, therefore, it cannot be mixed with ore and limestone. A separate charge opening must be provided in the furnace for accepting the revert matte at regular intervals in a predetermined quantity (Fig.20).

1ST IDEA (FOR DISCHARGE OF CONCENTRATE)



2ND IDEA FOR DISCHARGE OF CONCENTRATE

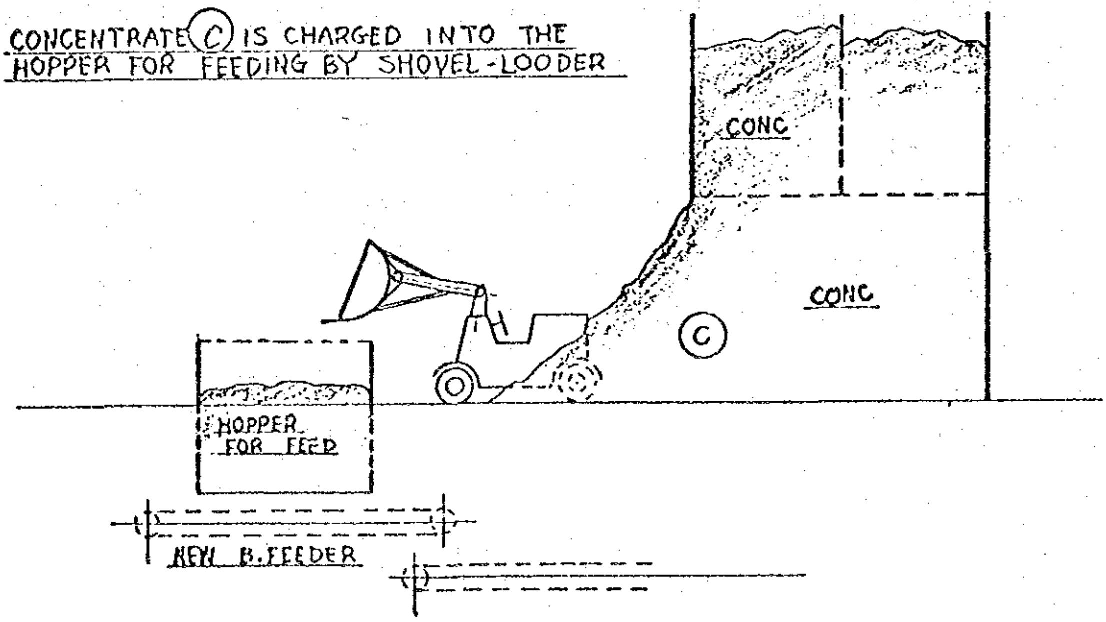


Fig. 19 SHOULD BE REVISED A LITTLE

Discharge of concentrate

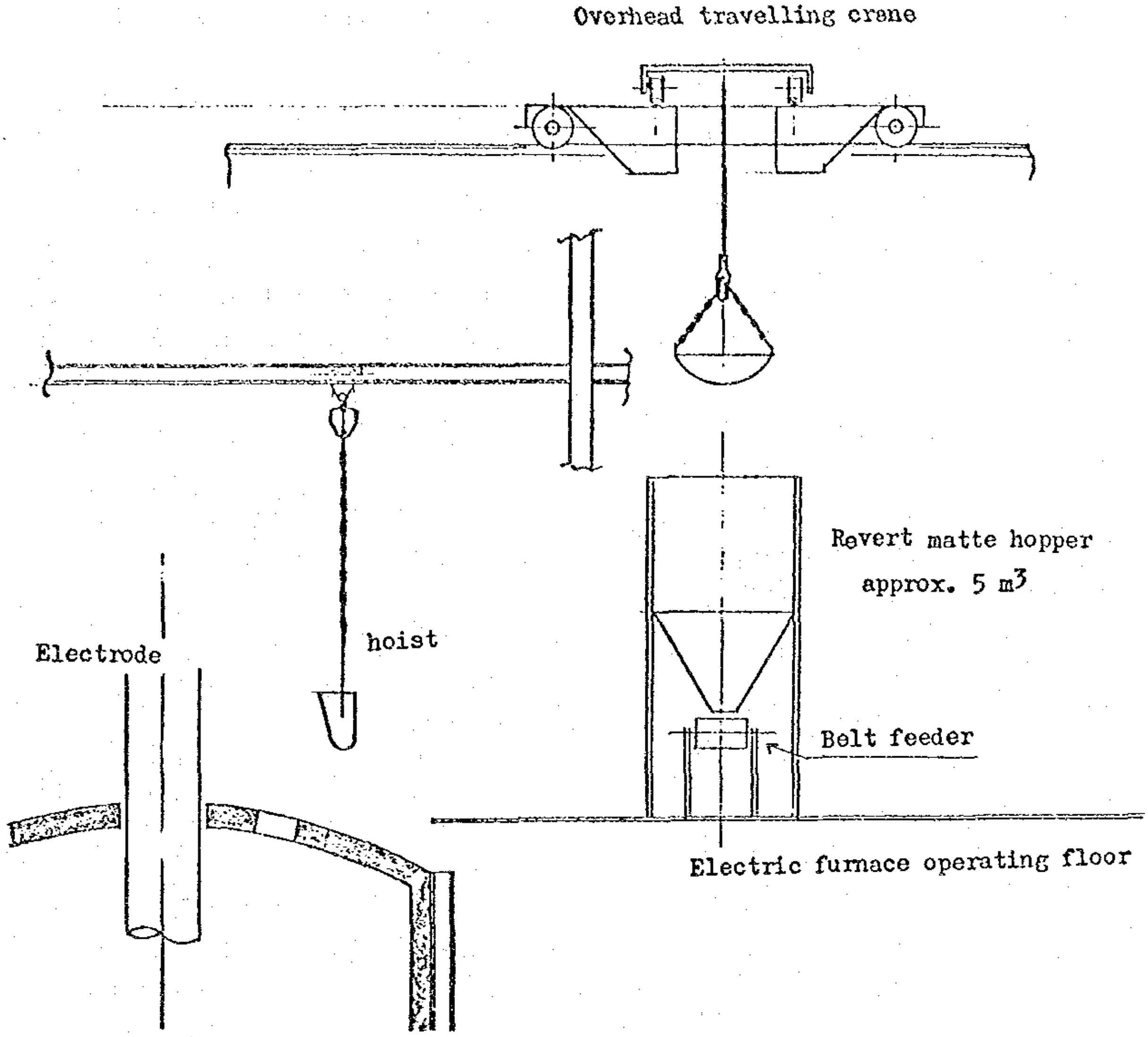

Overhead travelling crane
Revert matte hopper
approx. 5 m3
hoist
Electrode
Belt feeder
Electric furnace operating floor

Fig. 20

A revert slag hopper with a capacity of $5m^3$, approx. 10 tons, should be installed on the electric furnace operating floor, on the crane side, to be filled with 10 tons of matte with the crane. The revert matte is to be metered by the belt feeder installed underneath the hopper, and then transported sideways by a hoisted transport hopper or a wheelbarrow into the furnace.

2-2-3 Electric Furnace Charging Equipment

The electric furnace must ideally be charged with a constant amount of materials commensurlate with the power input. If much material is charged too quickly, the furnace gas and furnace body is cooled suddenly which causes difficulty in tapping, or the level of the molten content moves too fast when the matte is tapped or converter slag is charged, and the non-molten material in the furnace may be violently disturbed to create conditions condusive to explosion.

However, the existing charging equipment as shown in Fig.21 is not suitable for proper regulation of the charge rate in relationship to the melting speed for the following reasons:

(1) The hopper above the furnace is not capable of smoothly measuring and conveying the material due to its structural defective construction.

(2) The charge opening is too near to the furnace wall.

(3) The shuttle conveyor is not utilized to regulate the charge amount.

An example improvement proposal is shown in Fig. 22. The mixed material is first stored in the two bins installed over the electric furnace. Underneath each bin, a belt feeder and a chain conveyor are installed, the former for metering the material by means of the load height and the feed speed, and the latter for continuously charging the furnace with metered material through the respective charge openings. The charge dampers are opened and closed under the control of a timer.

2-2-4 Electrode Equipment

The existing Wisdom system electrode shown in Fig.23, operating at a phase-to-phase voltage of 180-200V is prone to water leaks from the holders and insulation failures leading to a great deal of sparkings, and overall operation instability. The major defects are as follows;

Fig. 21

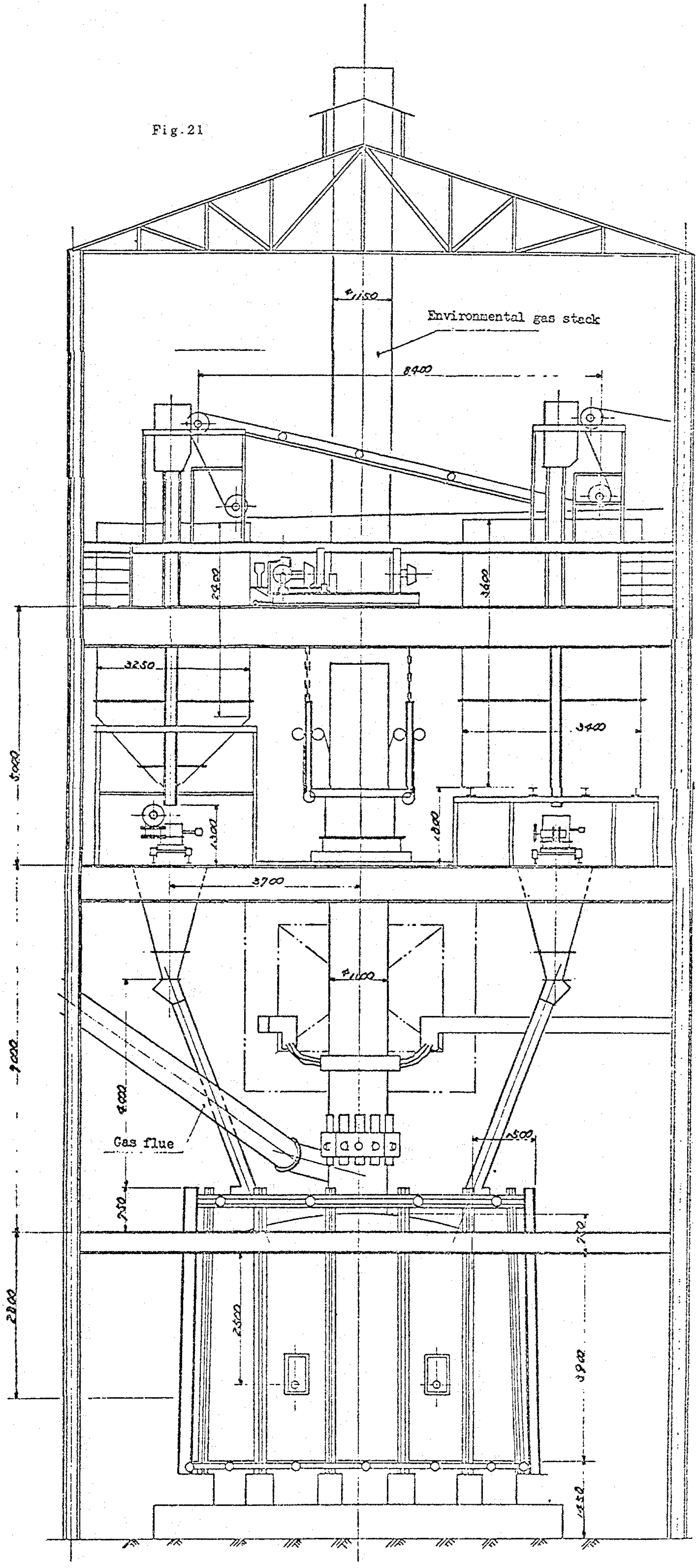


Fig. 21
Environmental gas stack
φ1150
8400
2400
3600
3250
3400
5000
1300
1800
3700
φ1000
9000
4000
Gas flue
500
750
750
2800
2500
3900
1250

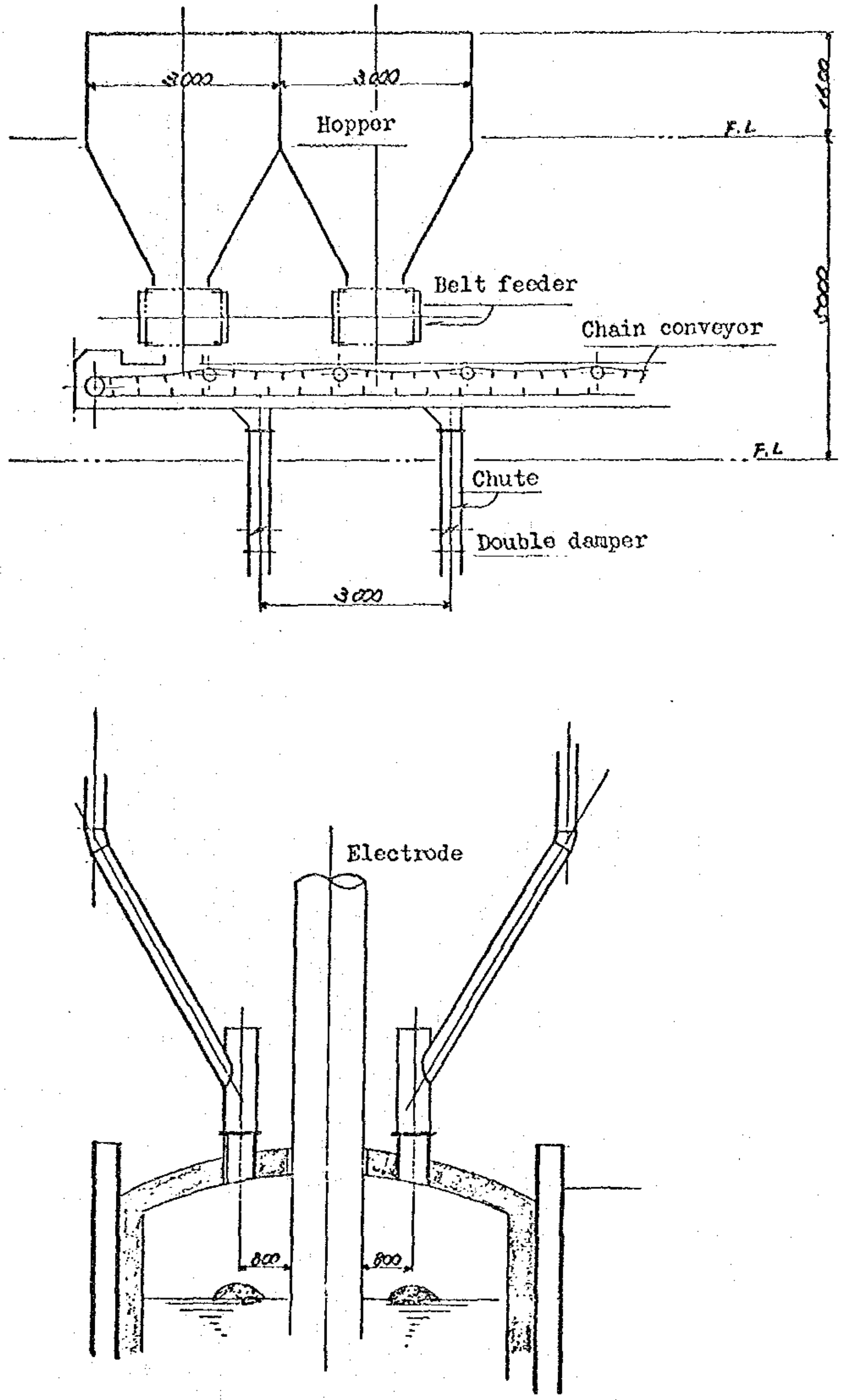

3000
3000
Hopper
F.L
1600
Belt feeder
Chain conveyor
15000
F.L
Chute
Double damper
3000
Electrode
800
800

Fig. 22

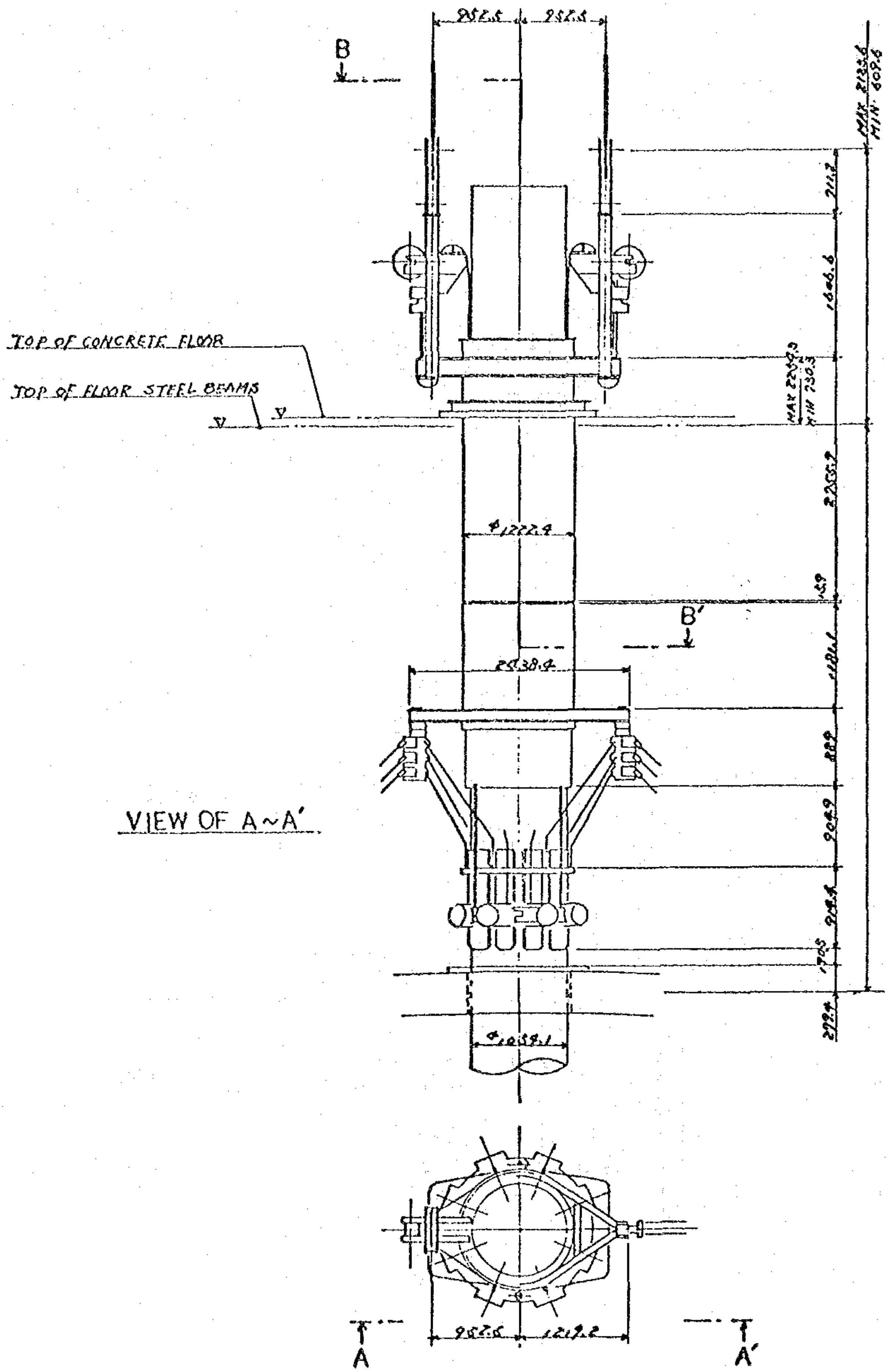


Fig. 23 SECTION OF B~B'

(1) The holders are always damaged through friction with the cases when the electrodes are slipped down.

(2) The 8-piece design of the holders is prone to uneven tightening, and the electrodes are usually noticeably bent due to uneven tightening.

(3) In addition to the above causes, also the steel plate electrode suspension strips make for imperfect contact between the holders and the cases.

The these reasons, the complicated Wisdom system electrode should be abandoned and replaced by a more stable and reliable electrode system and slipping system, and if possible, water cooling should be discontinued.

2-2-5 Slag and Matte Tapping

The present practice is to carry all slag in ladles and transport these ladles by diesel locomotives to the slag yard. However, due to frequent trouble with the locomotives and the unavailability of ladles, the furnace operation is greatly restricted. Whether the slag water-granulation system or the open slag pouring under water spray with subsequent truck loading by means of wheel loaders or crawler shovels is considered advisable for stable production and future increase of production from a conprehensive viewpoint.

2-2-6 Revert Slag Treatment

At present, 25,000m tons of high-copper-content revert slag seem to have been stocked, requiring effective measures for proper utilization, to be effected with at great effort. The method of treatment is as described earlier in this report.

In parallel with the revert slag treatment measures, also measures to reduce the amount of accumulated product is important. Because the copper content of matte is very low, the amount of revert may be reduced without much difficulty. The possible reasons for the increasing amount of revert product are as follows;

(1) The furnace is often shut down.

(2) The matte tapping speed is too low.

(3) Matte and converter slag are held too long in ladles.

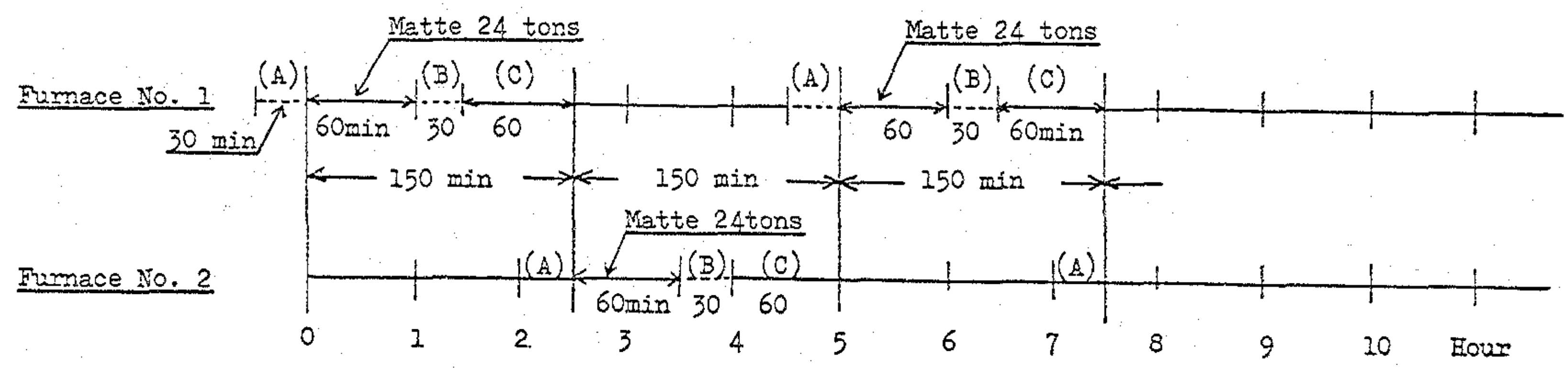
(4) Crude copper is cast at too low speeds.

The equipment must be re-built to eliminate these defects, and a sound operation standard must be formulated and rigorously inforced thorough training of the personnel.

Examples of operation schedule actually employed in Japan are shown below.

Fig. 24

(Example 1) Copper content of matte : 45 %

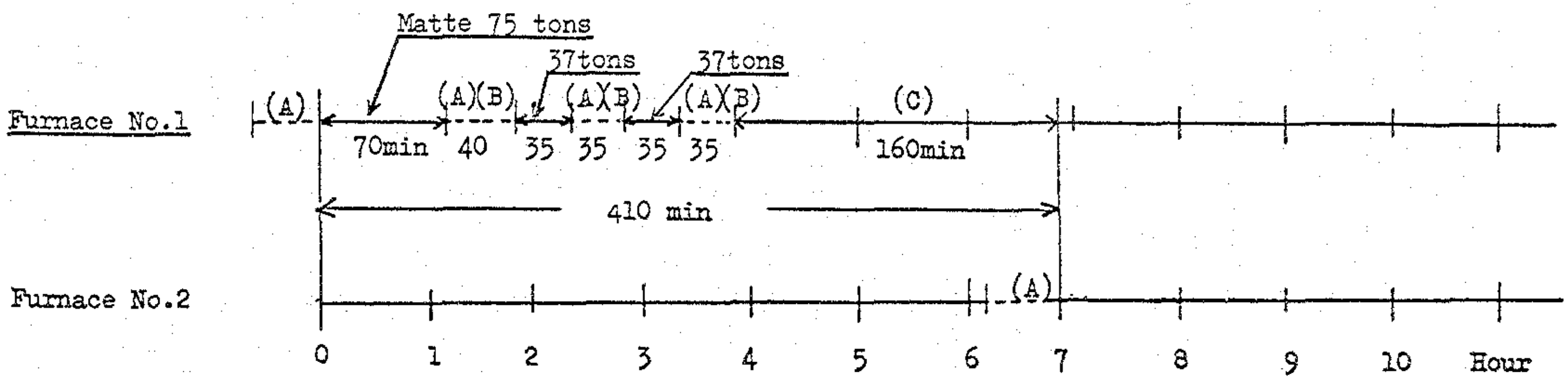


A: Matte tapping

B: C.F. slag flowing-out

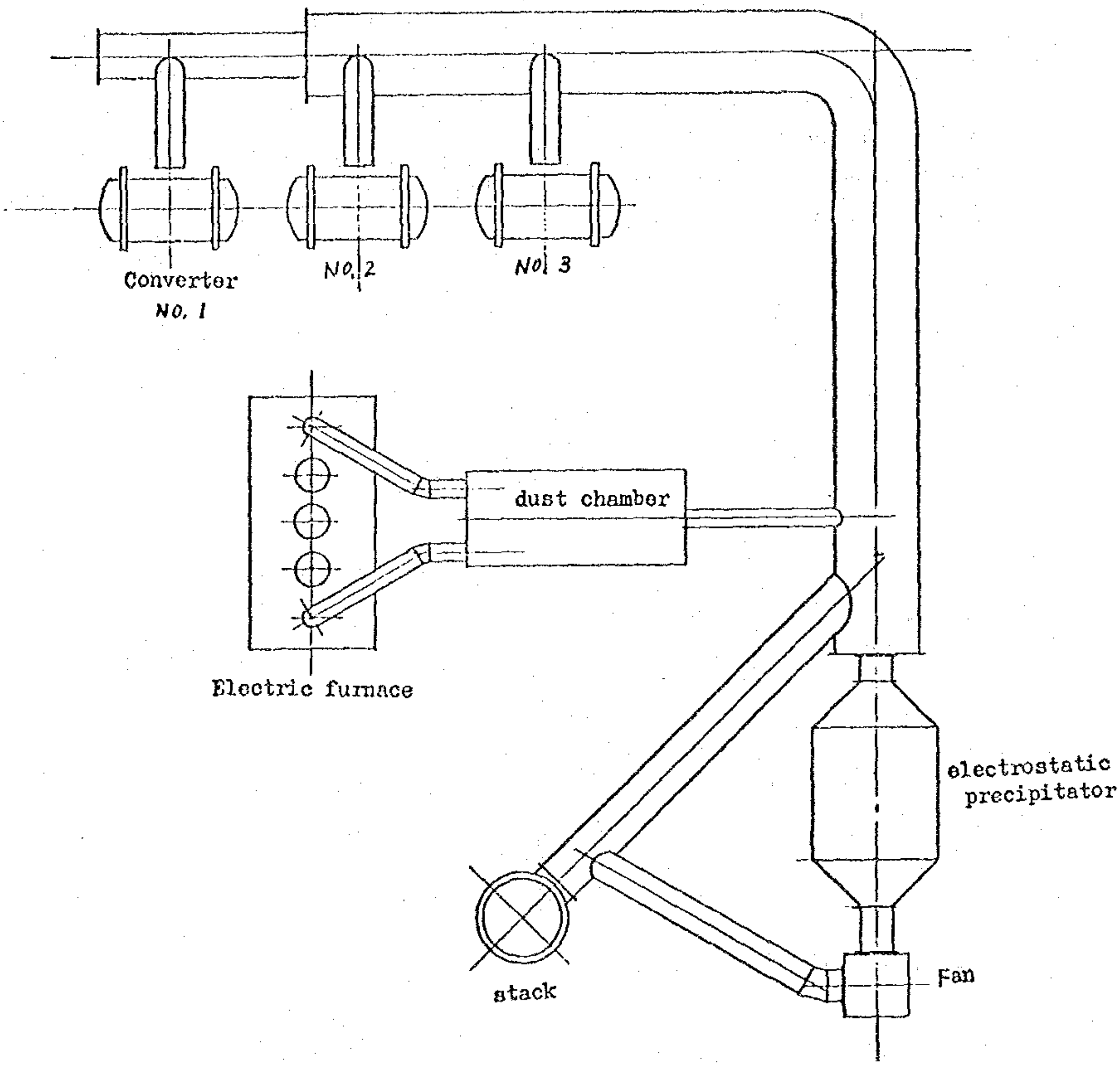
C: Copper blowing

(Example 2) Copper content of matte : 52 %



2-2-7 Electric Furnace Smoke Discharge Equipment

With the present, design, smoke is conducted through two 700 mm dia. flues from the furnace cover into the dust chamber, thence to the brick-lined main flue, and, conbined with conveter discharge gas, and discharged, via an electrostatic precipitator and a blower, into the atmosphere through the main stack. However, at present, both the electrostatic precipitator and the blower are out of service, and all smoke is discharged through the bypass flue by natural convection.



Converter
NO. 1
NO. 2
NO 3
dust chamber
Electric furnace
electrostatic
precipitator
stack
Fan

Fig. 25

However, because of the improper operation of the converter gas flue dampers, an extremely large amount of converter gas is discharged, and hardly any electric furnace exhaust gas is discharged. Most electric furnace discharge gas is discharged through the openings around the electrodes the view openings at the furnace cover into the furnace building, filling it with smoke and dust. Operation and control inside the furnace building during furnace operation is practically impossible at the present time. To achieve the following important objectives, the smoke discharge equipment must urgently be improved:

(1) Improvement of working environment

(2) Establishment of operation control system

(3) Maintenance of plant facilities

(4) Safety of personnel

The problems of SO_2 and dust contained in the exhaust gas will be left for future study.

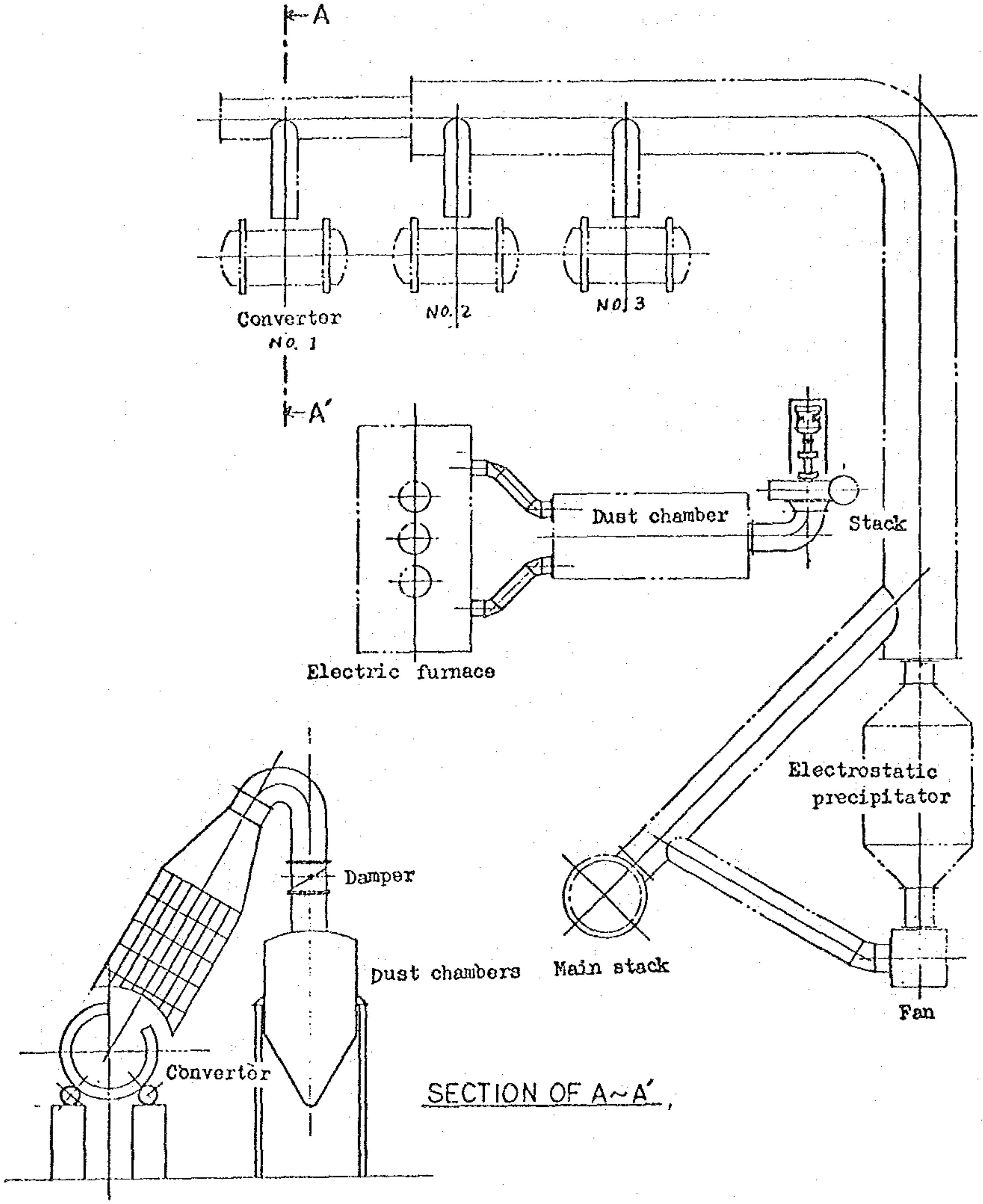

A
Convertor
NO. 1
NO. 2
NO. 3
A'
Dust chamber
Stack
Electric furnace
Electrostatic
precipitator
Damper
Dust chambers
Main stack
Fan
Convertor
SECTION OF A~A'

Fig. 26

Recommendations for exhaust gas disposal syatem

1. Electric furnace gas

The present up-take gas flue system should be replaced by a side take system, and a draft control fan should be installed at the discharge part of the dust chamber to discharge gas directly into the atmosphere. Care must be taken to maintain the discharge gas temperature below 300°C.

2. Converter exhaust gas

All the dampers at the converter outlets, i.e., dust chamber inlets, must be completely repaired.

3. The electrostatic precipitator and the discharge fan will not be taken into consideration this time. They will be considered when operation has been stabilized.

4. Main flue, et.

The existing flue, and the dust chambers for the electric furnace and the converters are lined with bricks, and are usable without any improvement. The dust discharge device and the manholes must be improved.

2-2-8 Converter Equipment

As the converters were out of operation during the survey period, their operation conditions were not observed. However, the fact that the turboblower suction side damper, disigned to operated during the converter shut-down time, was found to be frequently mis-operating due to the extremely comlicated construction of the pressure switch employed, this constituted a major cause of tuyere blockage trouble. The existing pressure switches must be replaced by accurate and simple pressure switches. Although the turbo-blower bearings are automatically cooled by oil, no measuring instrument for measuring the temperature of oil cooling water, its flow rate, and the temperature and flow rate of the oil itself is provided, and the bearing temperature continually rises above the tripping level. As the blowers are the most important units in the converter process, they must be provided with a complete set of control instruments. In addition, as previously mentioned, all sorts of spare parts including tuyere parts are running short and operation is severely restricted. As bricks are especially hard to procure, at present, the operation schedule is being adjusted to reduce converter temperature to prolong the service life of the brick linings, at the expense of production efficiency and at the sacrifice of limiting reduction of revert.

2-2-9 Copper casting equipment

Although there is no special problem, the copper casting time should be shortened as much as possible to reduce the production of revert.

As 10 - 13m tons of crude copper is produced in one blowing operation, the weight of a blister cake (presently 150 Kg) must be increased to 1.0 - 2.0 tons, and the molten copper should preferably be pound directly from the ladles. In this way, the trouble of copper casting machine will be disposed of, and the copper casting time can be greatly reduced, and the production of revert will be reduced. The existing copper casting machine is shown in Fig.28.

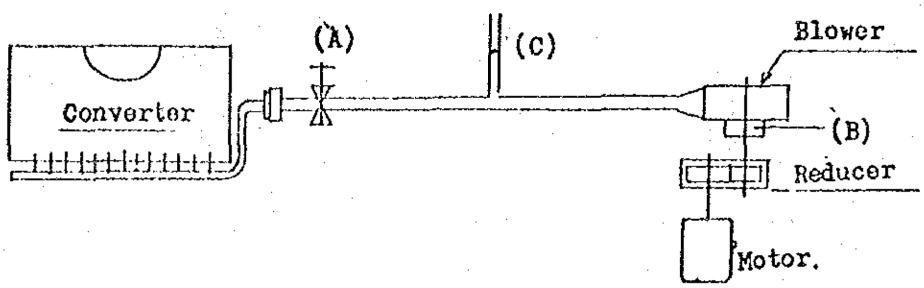


Fig. 27

To start operation

(1). (C) is closed and (A) is opened (manual operation)

(2). (B) is opened (automatically)

(3). converter starts to operate (air blow starts)

(note) As the tuyere resistance increases, blow air flow rate decreases.

To stop operation

1. converter operation is stopped
2. (A) is closed and (C) is opened (manual operation)
3. (B) is closed (automatically)

(note) (C) is a surge prevention damper.

Although (B) is supposed to operate automatically controlled by a pressure switch, its design is very obsolete and operation is unreliable, and often misoperation causes tuyere blockage trouble.

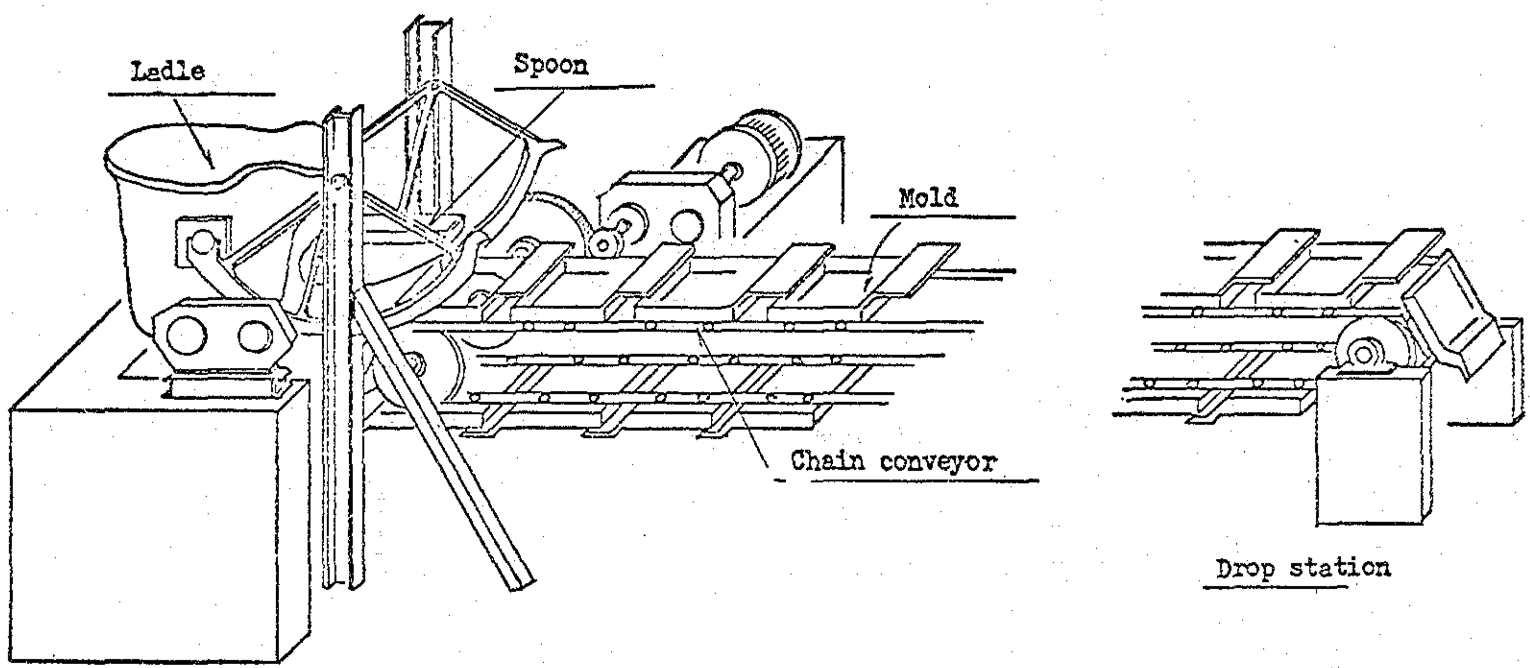


Fig. 28

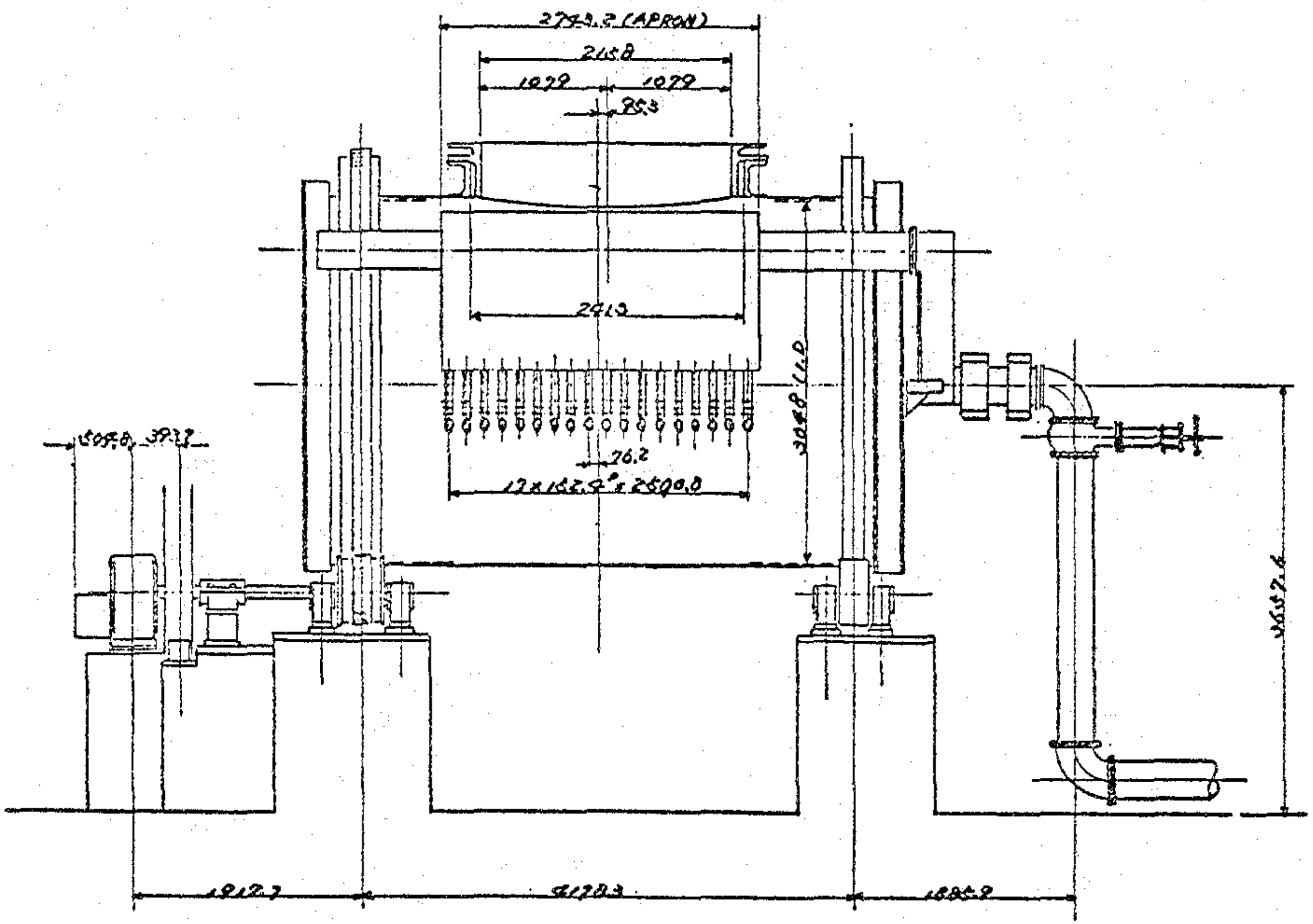


Fig. 29

2-3 Specifications of Main Facilities

2-3-1 Ore Accepting Facility

(a) Conveyor

The copper concentrate coming from the mill plant is manually shoveled on belt conveyor No.1, and via belt conveyors No.2 and No.3, it is charged into the ore storage bin. Revert matte is conveyed on No.9 and No.3 belt conveyors and charged into the storage bin, and flux is stored in the storage bin after passing over the same route as the copper concentrate. Sand is fed into the special bin installed on the side of the ore storage bin by belt conveyor No.10. All the conveyors operate at approx. 60m/min, and therefore, the conveying rate of all the conveyors is approx. 150 tons/hr.

No.1 belt conveyor 500 mm (w) x 114.5m (l) x 21m(h) 5.5 kw

No.2 belt conveyor 450 mm (w) x 14.4m (l)

No.3 belt conveyor 500 mm (w) x 77.0 m (l) x 14.5m(h)(11.0kw / 1.5kw (tripper)

No.9 belt conveyor 500 mm (w) x 37.0 m (l) x 3.5 m(h) 5.5 kw

No.10 belt conveyor 500mm (w) x 53.0 m (l) (out of operation)

(b) Crusher Facility

The two existing crushers, one for flux and the other for revert matte, are both out of service at present.

Flux crusher

Model:	Break crusher
Inlet size:	600 mm (l) x 230 mm (w)
Electric motor:	Removed
Installation location	Above No.1 belt conveyor

Revert conveyor

Model:	Break crusher
Inlet size:	600 mm (l) x 230 mm (w)
Electric motor:	Removed
Installation location:	Inside the crane yard, 5m away from electric furnace side wall
Attached facilities:	Revent receive hopper, 20 ton capacity

(c) Ore Storage Bin

For storing copper concentrate, flux, and revert, a total of 10 concrete bins are integrally installed.

The top of these bins is covered by a steel roof structure, and underneath them, five funnel shaped hoppers, with tow vertical and two inclined walls, are installed.

Outside dimensions: Approx. 10m (w) x 11m (1) x 26 m (1) (10 bins integrally built)

Capacity : 240 tons/unit x 10 units 2400 tons (120m^3)

Attached facilities:A steel plate sand storage bin with a capacity of 10m^3 is installed next to the ore storage bins.

2-3-2 Ore Charging Facility

(a) Ore Takeout Facility

Facility for taking out the contents of the ore storage bins and the sand storage bin.

Copper concentrate feeder
  Type and size: Belt feeder: 450 mm (w) x 6.0 m (1)
  No. of units: 5 (No.1 -- No.5 belt feeders)
  Electric motor: 2.7 kw x 5 sets

Recert and flux feeder
  Type and size; Vibrating feeder, 500 mm (w) x 1.0 m (1)
  No. of units: 5 (No.1 -- No.5 vibrating feeders)
  Electric motor: 0.75 kw

Sand feeder
  Type and size: vibrating feeder, 400 mm (w) x 4.5 m (1)
  No. of units: 1
  Electric motor: 0.75kw
  (Note) At present, out of service

(b) Transport Conveyor

Facility for transporting the copper concentrate and flux taken out of the storage bins into the bins above the electric furnace, consisting of three belt conveyors, No.4-No.6. A metering unit is incorporated in the No.4 conveyor.

No.4 belt conveyor: 450mm(w)x39.0m(1), 2.2kw

No.5 belt conveyor: 450mm(w)x14.0m(1)x5.0m(h) 5.5kw

No.6 belt conveyor: 500mm(w)x84.0m(l)x17.0m(h), 15.0kw
1.5kw(shuttle)

Attached facilities: Metering unit(out of service because of failure)

There are two bins above the electric furnace, one on each side, each provided with a charging conveyor installed underneath. These conveyors are designed to travel to charge ore alternately the 6 charge chute(3 by 2 rows).

Bin above electric furnace

Capacity: 20 m^3 xl unit, 30 m^3x 1 unit

Charging conveyor:

Type and size: Travelling belt conveyor, 450mm(w)x 120m(l)

Nos. of unit: 2(No.7 and No.8 belt conveyors)

Electric Motors: 2.2kw, 2 sets

2-3-3 Electric Furnace

The electric furnace is nearly rectangular(6,550mm(w)x 16,000mm(l) x 4,000mm(h)), with arched top, and in-line electrodes. The electrodes are supported by WISDOM BRAKE system holders, and are made of 3.2mm steel plates, 1,800mm long, and 1,050mm in dia. Current is supplied through the 8-piece holder which is water-cooled.

There are 6 charge openings in the furnace, symmetrically disposed on the two sides of the three electrodes, approx. 1,100mm from the outside walls of the furnace.

The exhaust gas is discharged through two 700mm dia. flues provided in the furnace ceiling, via a dust chamber and, main flue, and the main stack into atomosphere. There is installed additionally, a ventilation fan for discharging into the atomosphere the furnace gas which has leaked through electrode openjngs, but this is presently out of service.

Furnace body: 6,550mm(w) x 16,000mm(l) x 4,000mm(h)
arched top, kined with bricks

Electrodes: 1,050mm dia., made of 3.2mm steel plate
(with 220mm wide 12 ribs)

Electrode support system: WISDOM BRAKE system

Electrode interval: 3,000mm(surface to surface)

Water supply pump:

Nos. of unit and kw; 2 sets, 15 kw

Specification; 450 lit./min., 15 m head

Return pump:

Nos. of units and kw; 1 set, 10 kw

Specifications: 150 lit./min., 15 m head

Charging chute:

Shape and size; Rectangular shape, 350x300x4,000mm, 70inclination

Nos. of unit; 6

Gas flue;

Type; Uptake type, natural draft

Size; 700 mm dia., made of steel plate

Dust chamber:

Type and size; Rectangular, 6600mm(w)x12,000mm(l)x4,000mm(h) (straight portion)

Environment fun:

Type; Propeller fun

Nos. of units; 2

Electric motor; 5.5 kw

Electric lifting winch:

Type; Gear reduced, thruster braked

Nos. of units; 3

Electric motor; 7.5 kw

Transformer:

Capacity; 5,500 kwA

Primary voltage: 33,000 v

Secondary voltage; 120-220 v

Nos. of taps; 21

2-3-4 Converter

Three existing converters are all in a line facing the matte tapping hole of the electric furnace, and to supply them with the matte discharged from the electric furnace, double lipped ladles are used, and the crane is provided with two auxiliary hoists. To tilt the converter during a power failure or other emergency occuring during a blowing operation, a 75KVA diesel generater is installed to supply power to the converter tilting motor. The exhaust gas of the converters is drawn out through the hood, and introduced into the dust chamber through its ceiling, and discharged into the atomosphere via the main flue and the main stack by natural draft.

The exiting two turbo blowers are connected to the gate valves located near the converters. The respective discharge pipes are provided with 250mm dia. vent pipes which are provided with manually controlled valves. The air blow into the converters is also started and stopped by means of manually controlled valves.

Converter:

Size and nos. of units; 3,000mm dia.x3,900mm, 2 sets
3,000mm dia.x4,500mm, 1 set

Tuyere dia. and nos.; 62.5mm dia.x17 and 18

Electric motors; 20 kw, 965rpm, 3 sets

Ladle:

Type; Cast and steel ladle

Capacity; $2.55m^3$

Hood(extensively damaged)

Type; Natural cooling type

Material; SS(JIS)

Dust chamber:

Type; Arched ceiling, rectangular, brick lined, with steel hopper, externaly heat-insulated

Size; 4,000mm(w)x20,000mm(l)x4,000mm(h) (straight portion)

Turbo blower:

Type; Unidentified

Specifications; Pressure:15 psi, Air flow rate: 9,600 cf/min.
(apprx. $270mm^3$/min.)

Electric motor; Approx. 700 kw

Emergency power supply equipment:

Type; Diesel generator (415V)

Nos. of units; 2 sets (of which, 1 is not in operating condition)

2-3-5 Casting machine

The copper smelted in the converters is cast into molds by means of this machine and the crude copper is taken out after cooling. The machine consists of two rows of link chains, a series of molds mounted between the two rows of chains,and a pair of sprockets for driving the chains at a slow speed. Molten metal is poured into the molds in the pouring position located near the driven sprocket, and the filled molds are cooled by sprayed water while they move towerds the driving sprocket, where the molds are removed from the chains.

Type: Double chain drive type, continuous casting machine

External size: 1.2m(w)x12.2m(l) (sprocket distances)

Chain speed; 10.4 m/min.

Nos. of molds; 50(weight of one crude copper piece = 150 kg)

Drivingdevice: Motor 11 kw, 720 rpm
Reducer 1/58.2
Gear ratio 18/86

Ladle($90ft^3$) tilting device: Motor 15 kw, 720 rpm
Reducer 1/118
Gear ratio 15/68

Attachment device: Spoon tilting device(pneumatically operated)
(Note) Not in operating condition
Cooling device(water spray)

2-3-6 Smoke Discharge Facility

Both the exhaust gas of the electric furnace and that of the converters are led through the electrostatic precipitator for dust removal, and then discharged into the atmosphere by means of a discharge blower. Both the electric furnace and converters are provided with dust chambers connected to their outlets.

Dust chamber at elecric furnace outlets

Outside size; 6m(w)x8m(h)x13m(l)

Attachment; Screw conveyor , 300mm dia.x 18m(h)

(Note) This screw conveyor is for sending the dust collected in the chamber to the dust storage bin, but is presently not in operating condition.

Dust chamber at the converter outlet

Outside size; 5 m(w)x 7m(h) x 45 m(l)

Attachment; Dust discharge hoppers at 6 positions

Electrostatic precipitator

Already out of service for some time, and specifications such as dust collecting efficiency, flow rate, gas conditions, and electric facilities are not known. The initial design was for the dust collected in this precipitator to be pressure-fed to a pelletizer and eventual stored in the storage bin.

Outside size; 10.2 m(w)x 19.0 m(h)x 8 m(l)

Induction blower

Excessive damage through corrosion, etc. has made the blower completely useless, as is the case with the electrostatic precipitator. Performance data are unknown. With the precipitator and the blower out of operation, all the exhaust gas is discharged through natural induction into the atmosphere via the bypass flue.

Type; Plate fun

Blade size; 1.8 m dia.x 2 m(w)

2-3-7 Crane

There are two cranes of the same specifications installed near the roof for charging the converters with the molten material taken out of the electric furnace, and for pouring the molten metal taken out of the converters into the molds in the molding machine.

Type: Overhead travelling crane

Main hoist: Hoisting weight; 25 tons

Hoisting speed; 10 m/min.

Electric motor; 22 kw x 2sets

Auxiliary hoist: Hoisting weight; 10 tons hoist x 2 sets

Electric motor; 11 kw x 2 sets

Travelling mortor: 3.7 kw

Crane span: 15 m

2-3-8 Air Compressor

For removing the slag attached to the converter mouth, for tilting the molding machine spoon, and for other applications in the plant, an air compressor is installed.

Pressure: 5.0-7.0 kg/cm^2

Air flow rate: 16 $m^3/min.$

Electric motor: 140 kw

(Note) The electric motor is a second-hand motor and its exact output rating is unknown.

2-4 Material Balance

2-4-1 Composition of Copper Concentrate

(1) Composition of copper concentrates

Cu: 28%

Fe: 30%

S : 32%

SiO_2: 6%

(2) Component(estimated)

$CuFeS_2$: 80.5 %

FeS_2: 7.5 %

FeO: 2.6 %

Silicates: 6.0 %

Others: 3.4 %

Total; 100 %

2-4-2 Charge and Production Per Day

(1) Solid charge 143.3 m tons/day

Copper concentrates;	120.0 m tons/day	
Limestone:	5.3 m tons/day	
Revert matte:	18.0 m tons/day	
(120 m tons/day x 15 %)	Cu: 35 %	
	S : 6 %	
	Fe: 31 %	
	SiO_2; 17 %	
	Others:11 %	

(2) Production/day

Matte	102.5 M.T/D
2 $Cu.Fe.S_2$:	Cu_2S + 2FeS + S
97T :	42T 46T 9T
FeS_2 :	FeS + S (Desulfurization rate 30%)
9T	66T 2.4T
Revert matte	Cu_2S
	7.9T
Total	Cu_2S 49.9T + FeS 52.6T(+S 11.4T↑
Cu	39.9T (38.9%)
Fe	33.5T(32.4%)
S	29.1T(28.4%)

Slag/day 28.2T/D

	SiO_2	FeO	CaO	Others
Copper Concentrate	7.2T	3.1T		4.1T
Limestone			2.8T	0.3T
Revert matte	3.1T	7.2T		0.4T
Total	10.3T	10.3T	2.8T	4.8T

Gas (214,950/24 hrs)		8,960 Nm^3/h
SO_2	7,980 NM^3/D	214,950 NM^3/D
CO_2	1,124 NM^3/D	
N_2+O_2	205,846 NM^3/D	

(3) Molten charge

Converter slag

FeO	33.5T x 72/56 ∓ 43.1/D	
Flux(SiO_2 X)		0.92X/D
Others		0.08X/D

If SiO_2 28.0 % in slag, ... $\frac{0.92X}{43.1+X} = 0.28$

If tons of white matte is mixed with Cu 5.0% in converter slag(Cu_2S6.3%)

..... $\frac{0.8Y}{43.1+19+Y} = 0.05$

Therefore, 66.2T/D

Fe	33.5T	50.6%
O	9.6T	14.5%
SiO_2	17.5T	26.4%
Cu	3.3T	5.0%
S	0.8T	1.2%
Others	1.5T	2.3%
Total		100.0%

(4) Slag from electric furnace

	Slag from electric furnace	Converter slag	Total
Fe	8.0T	33.5T	41.5T(46.3%)
O_2	2.3T	9.6T	12.4T(13.2%)
SiO_2	10.3T,	17.5T	28.7T(30.7%)
CaO	2.8T	0	2.8T(3.0%)
Others	4.8T	1.5T	6.4T(6.8%)
Total	28.2T	62.2T	90.4T(100.0%)

(Note) The electric furnace slag is noticably high in Fe content and low in SiO_2. This may be beneficial to the life of the furnace lining, but certainly not conducive to the reduction of slag copper loss. The copper content in slag seems to be 1.0% on average, with 0.9% min.

To improve the copper yield ratio, a new leak-free electric furnace must replace the existing one, and the Fe/SiO_2 ratio in the slag must be reduced to below 1.2.

In this case, the silica must be charged in the electric furnace in the following amount.

$$\frac{41.5T}{28.7+0.92X} = 1.15$$

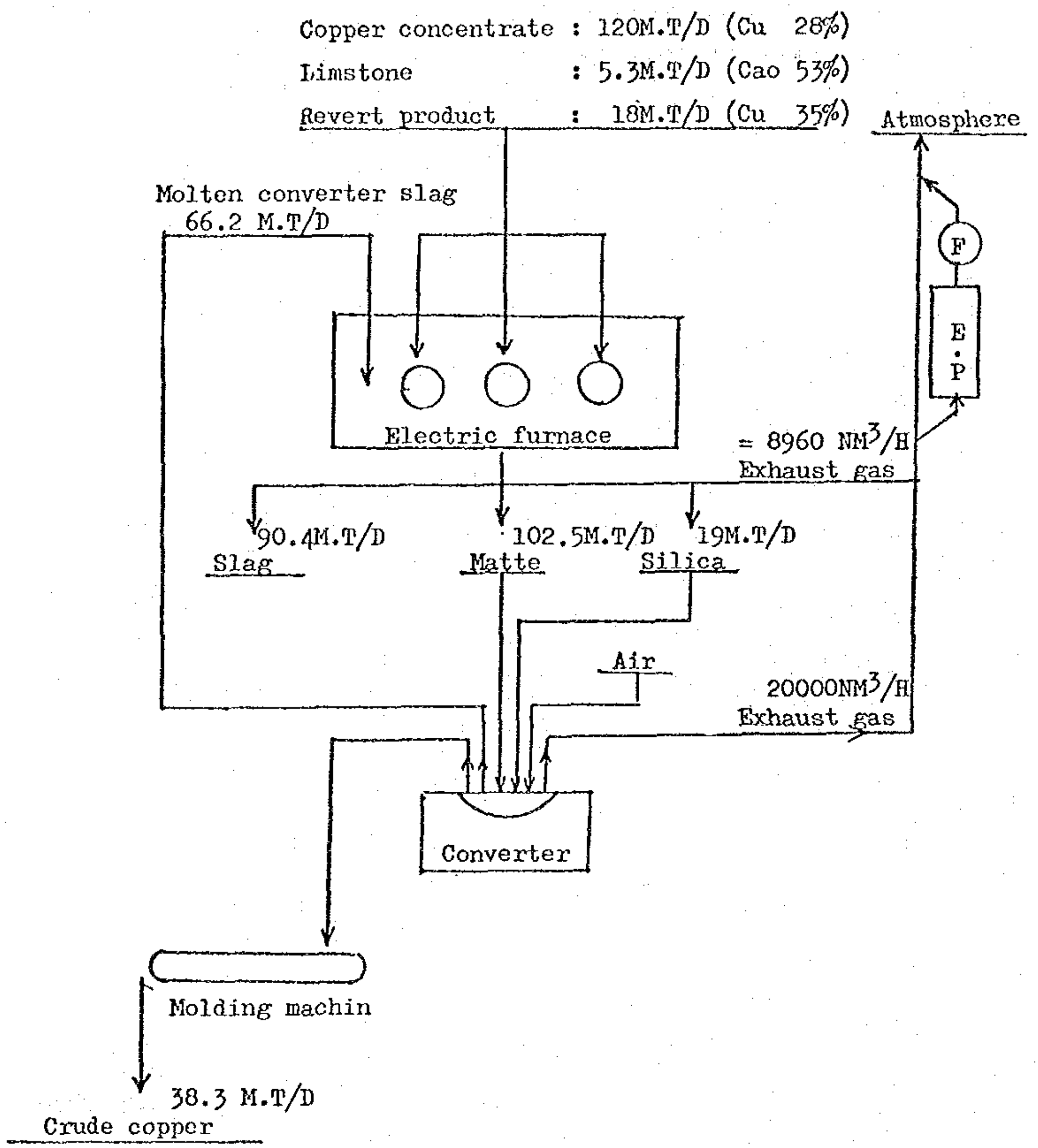
$$\therefore \quad 41.5=33.0+1.058X$$

$$\therefore \quad X = 8.0T/D$$

Composition:

Fe	41.5T	41.6%
O_2	12.4T	12.4%
SiO_2	36.1T	36.2%
CaO	2.8T	2.8%
Others	7.0T	7.0%
Total	99.8T	100.0%

Fig. 30 . FLOW CHART (PRESENT)



(Note) E.P. and F for exhaust gas are out of service due to deterioration

2-5 Electric Furnace Heat Calculation

Operating time 23 hrs

Charge

Copper concentrate 5.2 m tons/hr

Limestone 0.23 m tons/hr

Revert product 0.78 m tons/hr

Water(7%in ore) 0.39 m tons/hr

Conditions: Matte temperature 1.180°C

Slag temperature 1.260°C

Exhaust gas temperature 600°C

Required heat

(1) Heat of dissociation 383.23 Mcal/hr

FeS_2--- FeS + S -- 19.80Kcal/mol - FeS 64.35 Mcal/hr

5200kg x 7.5%/120 x19.8 x 1000

$2CuFeS_2$--- Cu_2S + FeS + S - 9.2Kcal/mol-$CuFeS_2$ 221.00Mcal/hr

5200kgx80.0% / 184x92x1000

$CaCO_3$--- CaO+CO_2 - 42.58Kcal/mol - $CaCO_3$ 98.93Mcal/hr

230kg/ 100x42.58x1000

(2) Water evaporation heat 312.39 Mcal/hr

Latent heat of evaporation 539.Kcal/kg

Speciifc heat 8.58Kcal/mol-J_2O -- 8.58/18x1000x550°C

--262Kcal/kg

Therefore (539+262) x 390kg

(3) Heat contained in matte 1,033.91 Mcal/hr

102.5M.T./23hrs x 232 Kcal/kg x 1000

(4) Heat contained in slag

28.2M.T./23hrs x 328 Kcal/kg'

(5) Heat for forming slag 34.83 Mcal/hr

1.03 M.T./23hrs x $\frac{56}{72}$ x 1000 x 100 kcal/kg-fe

(6) Heating of converter slag (20C) 21.24 mcal/hr

66.2M.T./23hrs x 7Kcal/kg

(7) Heat contained in exhaust gas 1.982.26 Mcal/hr

(6.23 x 1,500 NH_3/T) X$\frac{7.31}{22.4}$ x 650°C

(8) Oxidation heat of freed sulfur 1.098.80 Mcal/hr

(11400kg/23H)/32 x 70.94 x 1000

$S + O_2$ ---- SO_2

(9) Rediation heat 439 Mcal/hr

Side surface area $(6.55m + 15m) \times 2 \times 4.3m \times 115\% = 213m^2$

Roof and bottom area $(6.55 \times 15) \times 2 \times 115\% = 226m^2$

Total $439m^2$

Therefore, the required heat is:

$$(1) + (2) + (3) + (4) \quad (5) + (6) + (7) - (8) + (9) = 3440.61 \text{Mcal/hr}$$
$$= 4{,}000 \text{ K.W.H.}$$

Required transformer capacity:

at 95% power factor and O.C.B. limit of 90%,

4000/0.95/0.90 = 4,680 K.V.A.

2-6 Main Problems in Operation

The operation of the plant was surveyed for the first time on the 2nd of July, but due to lack of electrode paste, the entire plant was shut down from the 8th, and it never started operation again during the time we stayed there before moving over to Kampala on the 24th. However, during our stay, we were able to survey all the facilities in detail and also were able to discuss problems with the engineers in charge.

Frankly speaking, there is not ore unit free from problems. The diesel locomotives and all other transportation facilities are defective to some extent, and the crushing facility, machine tools and the electric units are all more or less defective.

Trouble develops continuously, but the major cause of operation interruption is trouble in the electric furnace, the heart of the whole plant. The breakdown of the plant shutdown hours related to the electric furnace troubles in 1976 by causes is as follows:

Electrode slip (broken suspension strip or chain)	194 hrs
Bus bar shorting	51 hrs
Water leak from holder	161 hrs
Repair of bus bar flexible cable	34 hrs
Repair of charge chute	40 hrs
Replacement and repair of holder	151 hrs
Molten material leak	71 hrs
Total	702 hrs

Besides the electric furnaces, cranes, converter blower, transportation facilities also constitute major factors for operation interruption, crane troubles causing 200 hours of operation interruption in 1976.

Operation Data

A) The amount of ore accepted in 1976 and 1977 are as follows;

	1976	1977
January	1,828 MT	1,611 MT
February	2,195	1,256
March	1,808	960
April	2,388	1,201
May	1,498	552
June	1,465	963
July	1,506	1,175
August	1,403	1,240
September	956	2,135
October	823	1,860
November	1,232	1,538
December	1,131	273
Total	18,233	14,764
Average/Month	1,519	1,230

B) Monthly Crude Copper Production in 1976 and 1977

	1976	1977
January	855.062 MT	59.107
February	683.136	Nil
March	373.580	Nil
April	614.050	213.465
May	305.191	359.359
June	421.177	276.743
July	402.528	513.836
August	201.970	167.210
September	274.992	265.310
October	218.962	225.039
November	348.875	112.335
December	360.049	79.684
Average/Month	418.	189.

C) Yearly Production Data

	1974	1965	1976	1977
Treated Cu Conc.(M.T)	35,384	35,414	23,377	12,666
Cu % of Conc.(%)	28.71	28.19	27.082	25.970
Cu content in Conc.(M.T)	10,216.166	9,983.066	6,503.481	3,289.360
Cu from Other Sources(M.T)	2.898	3.392	26.924	-
Crude Cu Production(M.T)	8,914.693	8,266.535	4,977.775	2,255.529
Crude Cu Production/Day(M.T)	27.014	23.187	15.474	7.889
Treated Conc/Day(M.T)	117.996	109.000	72.151	46.415
Yield Ratio (%)	79.02	74.78	78.48	64.86

Note: $\text{Yield Ratio} = \dfrac{\text{Crude Cu Production/Day} \times 99.1\%}{\text{Treated Cu Concentrate/Day} \times \text{Cu\%}}$

2-7 SPARE REQUIRED

(Estimated amount based on Thousand Yen)

A.	Ore Receiving Equipment	5,310
B.	Ore Charging Equipment	12,950
C.	Electric Furnace	53,510
D.	Converter	29,700
E.	Casting Machine	1,500
F.	Flue System	1,940
G.	Crane	2,780
H.	Turbo blower & Air Compressor	6,600
I.	Work Shop	23,000
J.	Building	2,500
K.	Lavoratory	2,820
L.	Service Water Appliance	3,300
M.	Electric Appliance	8,600
N.	Subsidiary Material	20,830
O.	Steel Material	8,000
P.	Transportation Machinery	20,300
Q.	Other General Expense	6,000
R.	Management Expense	3,000
	Total	212,640
S.	Transportation Expense	8,520
T.	Others (Escalation etc.)	24,000
	Gland Total	245,160

DETAIL OF SPARE (Based on Thousand Yen)
(Annual estimation, unless noted otherwise)

A. Ore Receiving Equipment

1. Transportation Conveyor 3 Lines (230m)

Roller 30 %	690
600∅ Pulley 2 Lines	720
Motor 5.5 kW 1 set	150
Motor 2.2 kW 1 set	80
Chain, Sprocket	50
Belt fastener	200
Conveyor belt 30 % 150m	1,370
Sub Total	3,260

2. Crushing Equipment

V belt and other	100
Motor 55 kw 6P 1 unit	1,000
Conveyor part	200
Screen	150
Sub Total	1,450

3. Ore Bin		0
4. General Expense	Sub Total	600
	A. Total	5,310

B. Ore Charging Equipment

1. Feeding

500 W x 6,000 L B.F. 1 line	7,020
500 W x 1,000 L V.F. 1 line	1,300
Sub Total	8,320

2. Transportation Conveyor 3 lines (160 m)

Roller	30 %	490
600 ∅ Pulley	2 lines	720
Belt	30 %	1,000
Motor 15 kW	1 unit	350
Motor 2.2 kW	1 unit	80
Chain, Sprocket		70
	Sub Total	2,710

3. Intermediate Receiving Bin & Charging Equipment

Intermediate Receiving Bin		0
Charging Equipment		1,920
	Sub Total	1,920
	B. Total	12,950

C. Electric Furnace

Oxygen 3 months	2,420
Lancing Pipe	15,000
Electric Casing	2,500
Electrode Paste	14,040
Wisdom Band	300
Brick 10 %	7,200
Electric Furnace Feed Pipe	1,100
Asbestos covered hose 200 m	900
1 1/2" Crip 1,000	150
Slag car Journal - bearing	300
Spout 4 ea.	600
Water Jacket 2 ea.	300
Gate valve 20 ea.	80
Electrode lifting part 10 %	680
Brick Tool	440
Electric Spare part	5,000
General Expense	2,500
C. Total	53,510

D. Converter

2 1/2 Steel ball	300
Steel Sheet	1,000
Steel Flange	200
Flexible Hose	300
Accessory	500
Universal joint	400
Asbestos code and sheet	300
Graphite coated Rope	100
Packing sheet for Flange	300
Others	500
Refractory 1 unit	22,000
Brick Tool	440
Tuyere Body	1,500
Gate valve	360
General Expense	1,500
D. Total	29,700

E. Casting Machine	E. Total	1,500

F. Flue System

Electric Furnace Fan part (Impeller, Shaft Bearing)	400
Convertor Fan part	900
Environment Fan and Motor assembly	540
Others	100
F. Total	1,940

G. Crane

Wire rope		250
Pinion 5 ea.		300
Lifting and Travelling gear	3 ea.	370
End shaft 4 pieces		200
Driving Pinion gear Box	2 ea.	410
Bearing		750
Others		500
	G. Total	2,780

H. Air Compressor and Turbo blower

Air Compressor part	10 %	1,500
Turbo blower	10 %	4,500
Others		600
	H. Total	6,600

I. Work Shop

Lathe 1800 1 set		5,600
Radial drilling machine	1 set	6,500
Sawing machine	1 set	1,200
Roll bender	1 set	1,800
Shearing machine	1 set	6,000
Welder	2 sets	550
Gas cutting apparatus	2 sets	150
Hand grinder	2 sets	100
Tool and Measuring instrument		1,100
	I. Total	23,000

J. Building

P. V. C. board		1,200
Steel material		1,000
Others		300
	J. Total	2,500

K. Laboratory

Ammonium solution	40
Sodium oxide	40
Sulfuric acid	80
Hydrochrolic acid	50
Ammonium	20
Urea	20
Filter paper	10
Platinum crucible	60
Glass implement	120
Paper, Rubber	220
Phosphoric acid	100
Screen	500
Thermo - couple	600
Stop watch	100
Blister sampling drill	50
Others	800
K. Total	2,820

L. Service Water Appliance

Main pump 1,000 ℓ/min H = 90m 1 set	1,200
Return pump 380 ℓ/min H = 20m 1 set	600
Electric furnace cooling water pump 600 ℓ/min H = 40m 1 set	800
Other cooling water pump 60 ℓ/min H = 10m 1 set	200
Piping Material	500
L. Total	3,300

M. Electric Appliance — M. Total 8,600

N. Subsidiary Material

Sand	1 month	870
Wood	6 months	750
Oxygen	3 months	100
Acetylene	6 months	350
Inert Gas		100
Welding rod		870
Gas cylinder guage		90
Dieseloil, Lube oil		16,700
Others		1,000
	N. Total	20,830

O. Steel Material

Ordinary steel plate	50 t	4,000
Checkered steel plate	15 t	1,300
Bar, Shaped steel		2,000
Others		700
	O. Total	8,000

P. Transportation machinery

Dump truck	14 t	1 car	6,000
Pick up truck		1 car	800
Forklift	8 t	1 car	4,500
Concrete breaker		1 car	9,000
		P. Total	20,300

Q. Other general expense	Q. Total	6,000
R. Management expense	R. Total	3,000

2-8 Recruiting Facilities for Smeltery Reconstruction(At Jinja)

(Rough Calculation) ¥ 252,000,000.

Conveyor balance (transportation into storage bin)	¥4,900,000
Repair shop expansion	¥10,900,000
Compressor 1 set	¥18,000,000
Diesel generator 1 set	¥8,000,000
Repair shop machines	¥80,400,000
Planer(3,000mm length x 1,500mm width, 1 set)	¥12,000,000
Milling machine(Stroke 300-350mm, 2 sets)	¥10,000,000
Shaving machine(Stroke 800mm, 2 sets)	¥6,500,000
Radial drill (1 set)	¥15,000,000
Facing lathe (1.5m dia. x 0.5m width, 1 set)	¥16,500,000
Roll bender(1.7m dia. x 2.0m length, 1 set)	¥3,600,000
Shearing machine (1 set)	¥6,000,000
Anvil (2 sets)	¥400,000
Electric welding machine (2 sets)	¥600,000
Gas cutting machine 95 sets)	¥300,000
Set of tools	¥2,000,000
Chain block (0.5-3.0 ton, 6 sets)	¥300,000
Hand grinder (4 sets)	¥200,000
Overhead travelling crane (1 set)	¥6,000,000
Others	¥1,000,000
Transportation facilities	¥59,500,000
Wheel loader (1.4m^3 x 1 set)	¥8,500,000
Crawler shovel (1.5m^3 x 1 set)	¥12,000,000
Diesel locomotive (100 ton x 1 set)	¥18,000,000
Crane truck (10 ton x 1 set)	¥12,000,000
Concrete breaker (rod dia.: 80-100 mm)	¥9,000,000
Cone Crusher	¥12,000,000
Crushing Conveyor and Support	¥6,100,000
Vibration Screen	¥1,200,000
Ladle (3m^3 x 8 sets)	¥40,000,000
Fence	¥9,000,000
Power Accepting Equipment	¥2,000,000

2-9 Overall Expenses (Rough calculation)

2-9-1 Slemtery Reconstruction Expenses (Jinja Smeltery) (¥1,000)

	Repair expense	Recruiting expense	Remodeling expense	Total
Transportation to storage bin	7,710	4,900	0	12,610
Storage bin	0	0	58,600	58,600
Transportation to a position above electric furnace	4,820	0	0	4,820
Electric furnace charging	300	0	16,100	16,400
Electric furnace equipment	0	0	200,070	200,070
Electric furnace exhaust gas equipment	0	0	14,000	14,000
Converter equipment	0	0	88,400	88,400
Converter blower equipment	0	0	3,900	3,900
Office and building	0	10,900	0	10,900
Blower room units	0	26,000	0	26,000
Repair shop machinery	0	80,400	0	80,400
Crane and moiding equipment	27,200	0	0	27,200
Main plant building	20,000	0	0	20,000
Transportation machinery	0	59,500	0	59,500
Crushing machinery	4,200	19,300	0	23,500
Power acceptance equipment	2,000	2,000	19,000	23,000
Others	1,000	49,000	4,500	54,500
Total	69,230	252,000	404,570	723,800

Transportation expense	29,000
Erection expense	55,000
Reserve expense	81,000
Others	81,000
Grand Total (¥1,000)	969,800

2-9-2 Overall Expense (¥1,000)

Jinja Smeltery Reconstruction

	¥1,000/Month
Transportation expense	22,760
Copper concentrate	13,880
Blister(to monbasa)	8,190
Limestone	690
Silica	0
Operation expenses	36,750
Depreciation	7,270
Interest	4,040
Total	70,820

2-9-3 Smeltery New Construction Expenses (Kilembe)

¥1,000 -

	Repair expense	Recruiting expense	Remodeling expense	Total
Transportation to storage Bin	0	4,900	36,500	41,400
Storage Bin	0	0	125,400	125,400
Transportation to a position above electric furnace	0	0	26,800	26,800
Electric furnace changing	0	0	23,400	23,400
Electric furnace equipment	0	0	304,100	304,100
Electric furnace exhaust gas equipment	0	0	33,000	33,000
Converter equipment	20,000	0	287,800	307,800
Converter blower equipment	500	0	3,600	4,100
Office and Building	0	0	256,900	256,900
Blower room unit	0	26,000	0	26,000
Repair shop machinery	2,000	80,400	0	82,400
Crane and equipment	22,000	0	24,600	46,600
Main Plant Building	0	0	412,700	412,700
Transportation Machinery	1,000	59,500	0	60,500
Crushing Machinery	4,200	19,300	0	23,500
Power Acceptance Equipment	0	0	302,200	302,200
Others	0	40,000	96,500	136,500
Total	49,700	230,100	1,933,500	2,213,300

Transportation Expense	89,000
Erection Expense	350,000
Reserve Expense	265,000
Others (escalation, etc)	265,000
Grand Total	3,182,300

2-9-4 Overall Expenses (Kilembe Smeltery)

	Kilembe New Smeltery Construction ¥1,000/Month
Transportation Expense	14,530
copper concentrate	0
Blister to Mombasa	11,760
Limestone	0
Silica	2,770
Operation Expense	33,740
Depreciation	23,900
Interest	13,260
Total	85,430

As can be dedcucted from a comparison of the overall expenses for the Jinja smeltery re-construction and that of Kilembe smeltery new construction, the per ton cost for crude copper of the new Kilembe smeltery will be ¥19,000 higher than that of the re-constructed Jinja smeltery:

Newly Built Kilembe smletery

¥85,530,000 ÷ 773 = ¥110,520/ton of crude copper

Re-constructed Jinja smeltery

¥70,820,000 ÷ 773 = ¥91,620/ton of crude copper

Also the fund requirement, including the fund for spare parts, will be far larger for a new smeltery ocnstruction as follows:.

	Jinja smeltery reconstruction	Kilembe new smeltery construction
	¥1,000	¥1,000
Re-construction or new construction expense	969,800	3,182,000
Spare parts expense	245,160	245,160
Total	¥1,214,960	¥3,427,460

Although the new construction of a smeltery at Kilembe was first considered to be ultimately more profitable on account of the reduced transportation expenses, resulting from the copper concentrate transportation to Jinja being replaced by less expensive crude ore transportation, and the expected reduction of labour expense resulting from the unification of maintenance operations with those of the mine, the above calculation of the overall expenses and fund requirement seem to indicate that re-construction of the

Jinja smeltery would be more advantageous.

(Note 1) Monthly concentrate and crude copper amounts in the overall expense calculation

Crude ore 50,000 tons/month Copper content: 1.75%

Mill Recovery 92%

Copper content of copper concentrate 28%

Therefore $\frac{50,000 \times 0.0175 \times 0.92}{0.28} =$ 2,875 tons/month

Wet ore 2,875 x 1.10 = 3,163 tons/month

Crude copper

Yield ratio 95%

Copper content in crude copper 99.0%

Therefore $\frac{2,875 \times 0.28 \times 0.95}{0.99} =$ 773 tons/month

(Note 2) Amount of copper concentrate in flow chart

In actual operation, the total elimination of load variation is impossible. Therefore, on the assumption of a max. load factor if 115%, the above copper concentrate amount was determined $\frac{2,875 \text{ tons} \times 12 \text{ months}}{330 \text{ days}} \times 1.15$

120 tons/day

140

(Note 3) Transportation cost (Kilembe-Mombasa)

Concentrate Transportation Cost (Kilembe-Mombasa)

Kilembe-Jinja	U.Sh 155.5	¥4,389/ton
Jinja-Malaba	U.Sh 89.0	¥2,512/ton
Malaba-Mombasa	K.Sh 170.0	¥5,175/ton

Blister Transportation Cost (Kilembe-mombasa)

Kilembe-Jinja	U.Sh 163.5	¥4,615 /ton
Jinja-Malaba	U.Sh 95.0	¥2,681/ton
Malaba-Mombasa	K.Sh 260.0	¥7,914/ton

Ocean Freightage (Mombasa-Yokohama)

Cu-Concentrate	(in drums)	US.$34.15	¥7,684/ton
	(in bulk)	US.$30.00	¥6,750/ton
Blister		US.$38.60	

Currency Fare Charge

38.6, 12% US.$55.232, ¥12,427/ton

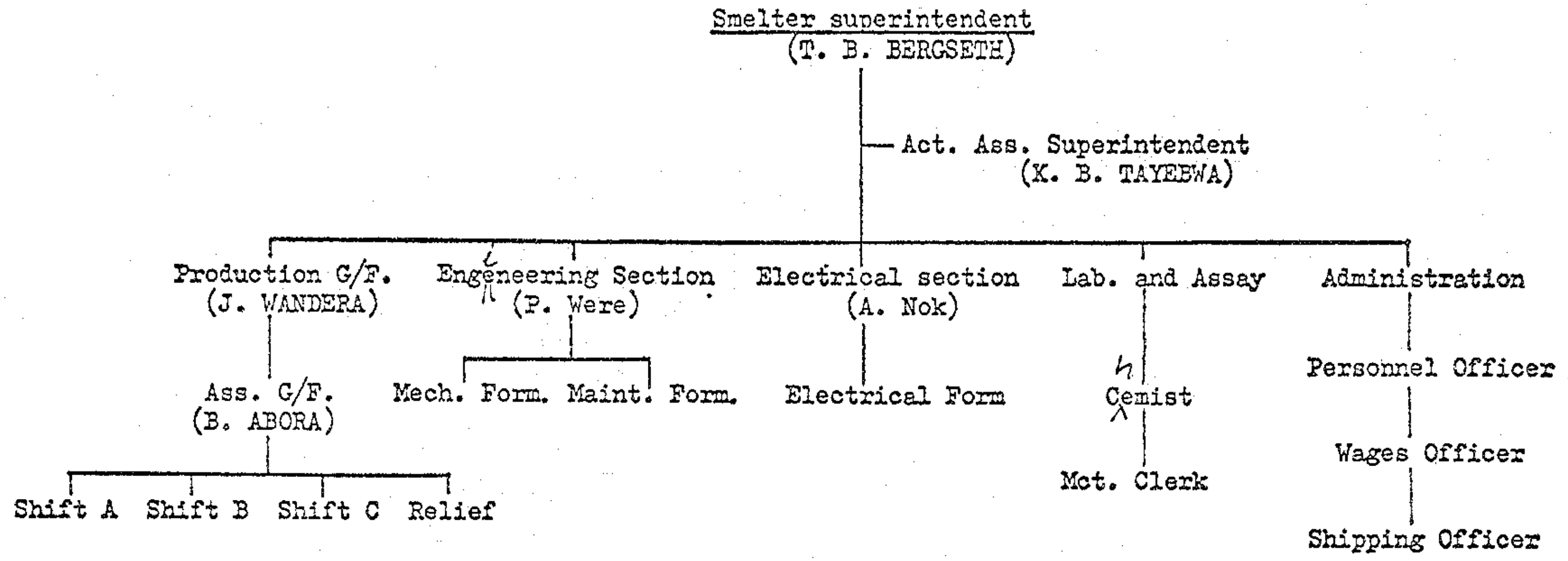
Banking Expense 12.0

Exchange Rate US.$1.0 = U.Sh 7,972 = J.Sh 7.392 = ¥225

(Note 4)

	Jinja(U.Sh/month)	Kilembe(U.Sh/month)
Labour Expense	292,000	205,400
Material Expense	700,000	700,000
Power Expense	190,000	190,000
Miscellaneous Expense	120,000	100,000
Total	1,302,000 (¥36,750,000/month)	1,195,400 (¥33,740,000/month)

Fig. 31 ORGANIZATION



Overall supervisors	4
Senior supervisors	12
Junior supervisors	28
Workers	278
Casual workers	41
Total	363