A PRE-FEASIBILITY STUDY FOR THE PHOSPHATE DEVELOPMENT PROJECT

THE REPUBLIC OF ZAMBIA

ABRIDGEMENT

1985

JAPAN INTERNATIONAL COOPERATION AGENCY

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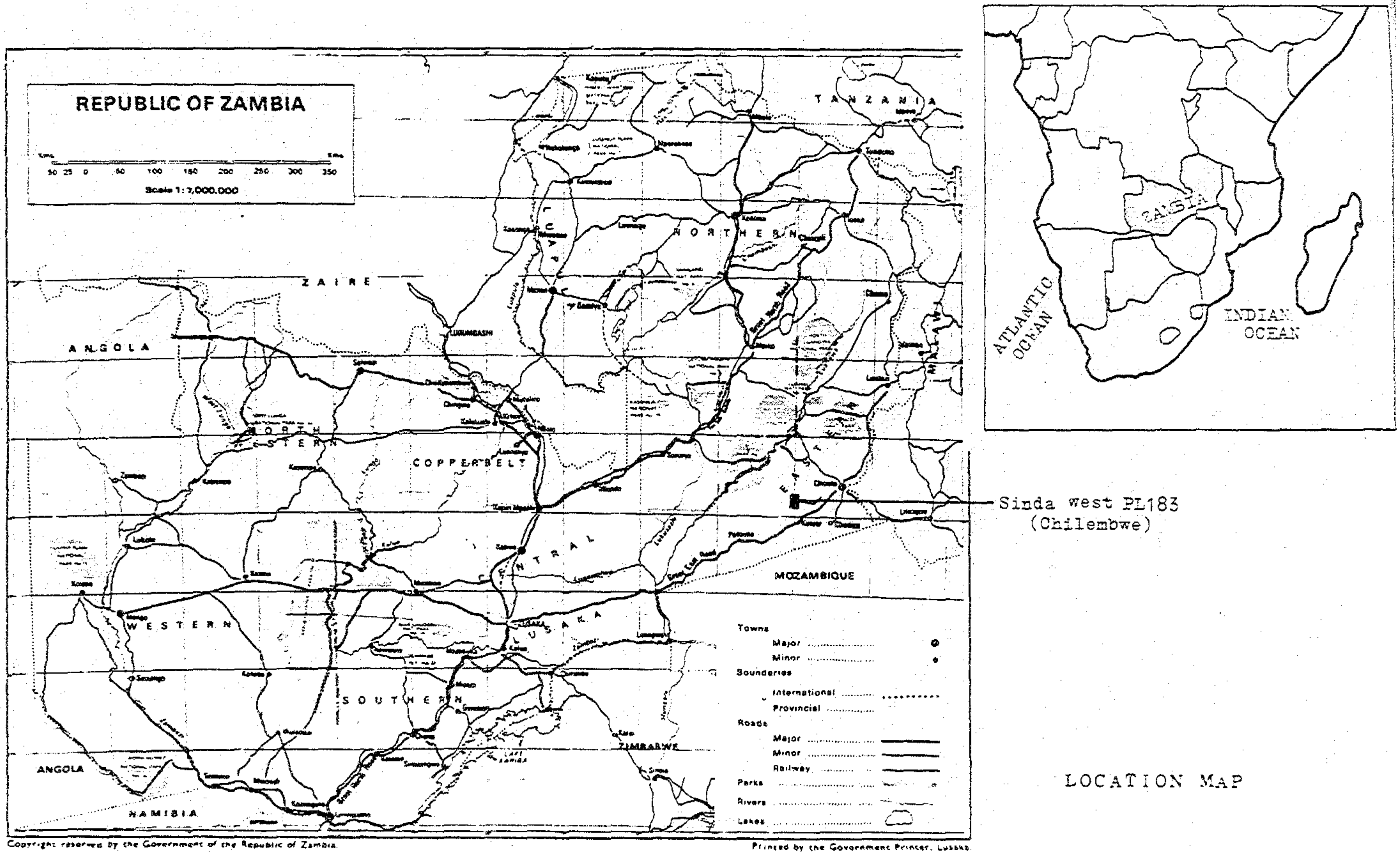

LOCATION MAP

Abstract

Since 1980, a detailed prospecting programme has been carried out by MINEX, over an area of apatite occurrence called the Chilembwe Prospect on the land of Sinda West in the Eastern Province.

Pursuant to the request made by the Government of the Republic of Zambia, the Japan International Cooperation Agency decided to implement a prefeasibility study for the phosphate development project.

After the field exploration work and the laboratory investigation, a mining plan was designed to produce annually a sum of 35,000 tonnes of 30% P_2O_5 concentrates over a period of fourteen years. With the estimation of the capital cost and the operation expenses, the internal rates of return were calculated in the financial and the economic evaluations. A marginal profit can be expected on a private mining enterprise and somewhat a higher contribution to the national economy is estimated.

Establishment of the apatite mining requires the existence of the phosphatic fertilizer plant in the country. The overall economic evaluation on the phosphatic industry should be made in connection with a feasibility study of the fertilizer plant.

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INTRODUCTION

It is well-known that Zambia has a great agricultural potential which can only be achieved with use of chemical fertilizers.

MINEX, the mineral exploration department of Zambia Industrial & Mining Corporation, Ltd., has been engaged in a search of phosphates. The Chilembwe Prospect is one of promising areas which has been located on the land of Sinda West in the Eastern Province.

Pursuant to the request made by the Government of Zambia, the Japan International Cooperation Agency dispatched a preparatory mission in October, 1983, and decided to undertake implementation of the study for

(1) confirmation of ore reserves with a research of ore dressing at the first stage, and

(2) designing of a mining plan at the second stage.

Preliminary investigation of magnesite deposits was also recommended for provision of raw material which might be necessitated for producing fused magnesium phosphate.

The results of the first stage are summerized in Chapters 2 to 4, and the production plan and the bases of designing are explained in Chapters from 5 to 10.

1. OBJECTIVES OF THE STUDY

The project intends to study the feasibility of development of apatite deposits in Chilembwe as a step toward the realization of the domestic production of phosphatic fertilizers.

Objectives of the study at the first stage were set

(1) to assess the phosphate reserves,

(2) to make out an investigation for ore dressing, and

(3) to confirm an occurrence of magnesium resources.

The Preparatory Mission selected two targets, namely, No. 2 and No. 4 Orebodies to be drilled and from these, the ore reserves exceeding 1.5 million tonnes are estimated and the apatite concentrates of 30% P_2O_5 are expected to be recovered with a rate of more than 80% recovery. In the second stage of the project, the designing of a mining plan and a pre-feasibility study of the project are carried out.

2. CHILEMBWE PROSPECT

2.1 Location

The prospect has been named after a village called Chilembwe which is situated some 8 km north west of the area.

The Chilembwe Prospect is in the vicinity of 13°59′S, 31°41′E, and its camp site is located some 28 km north of Sinda. All of the deposits are found within a distance of 9 km to the north of the camp. Sinda is situated 460 km east of Lusaka and both are connected by the paved Great East Road which runs to Malawi.

2.2 Physiography

The Prospect lies on a plain, moderately timbered with shrubs, dipping slightly westward at an elevation of 920 m. The average precipitation is about 1,000 mm annually, and most of the precipitation falls mainly as brief cloudbursts from November to March.

2.3 Mineral Deposits

The Chilembwe Prospect refers to a number of phosphate deposits which occur in the area of syenite mass. The plutonic rocks range in composition from mica syenite to monzonite and form a part of Sinda Batholith which is said to be of 500 m.y. in age.

Phosphate deposits occur as massive lenzes composed of apatite with association of quartz, alkali-feldspar, mica or amphibole. When apatite is associated with quartz and alkali-feldspar, ore is rather leucocratic, whereas ores which associate with ultrabasic rocks are deep green in color.

No. 2 Orebody occurs elliptically along a small hill and floats of ore are widely distributed at 190 meters north-south in length and 60 meters in an east-west direction. This area is adjoined to a possible satellite body of 60 meters in diameter to the west. A mineral deposit appears to be a large mass derived from late-magmatic segregation of alkali igneous rocks.

No. 4 Orebody indicates a form of sill-like. Apatite is accompanied with mica and amphibole and this orebody is probably, directly related to an intrusion of ultra-basic rocks.

2.4 Previous Work

The prospect is covered under Sinda West P.L. 183, formerly known as Lusandwa P.L. 144.

Since 1980, a detailed prospecting programme has been carried out by Minex, comprising geological, geochemical and geophysical surveys.

Fifteen holes totalling some 805 m were drilled in 1982 to investigate geophysical anomalies and downward extension of Nos. 1 to 3 Orebodies. Five holes, from DDH.1 to DDH.5, were drilled at No. 1 Orebody and DDH.13 was put down at No. 3 Orebody. Other nine holes were sunk over the area of No. 2 Orebody.

Apart from these, numerous pits and several trenches were excavated.

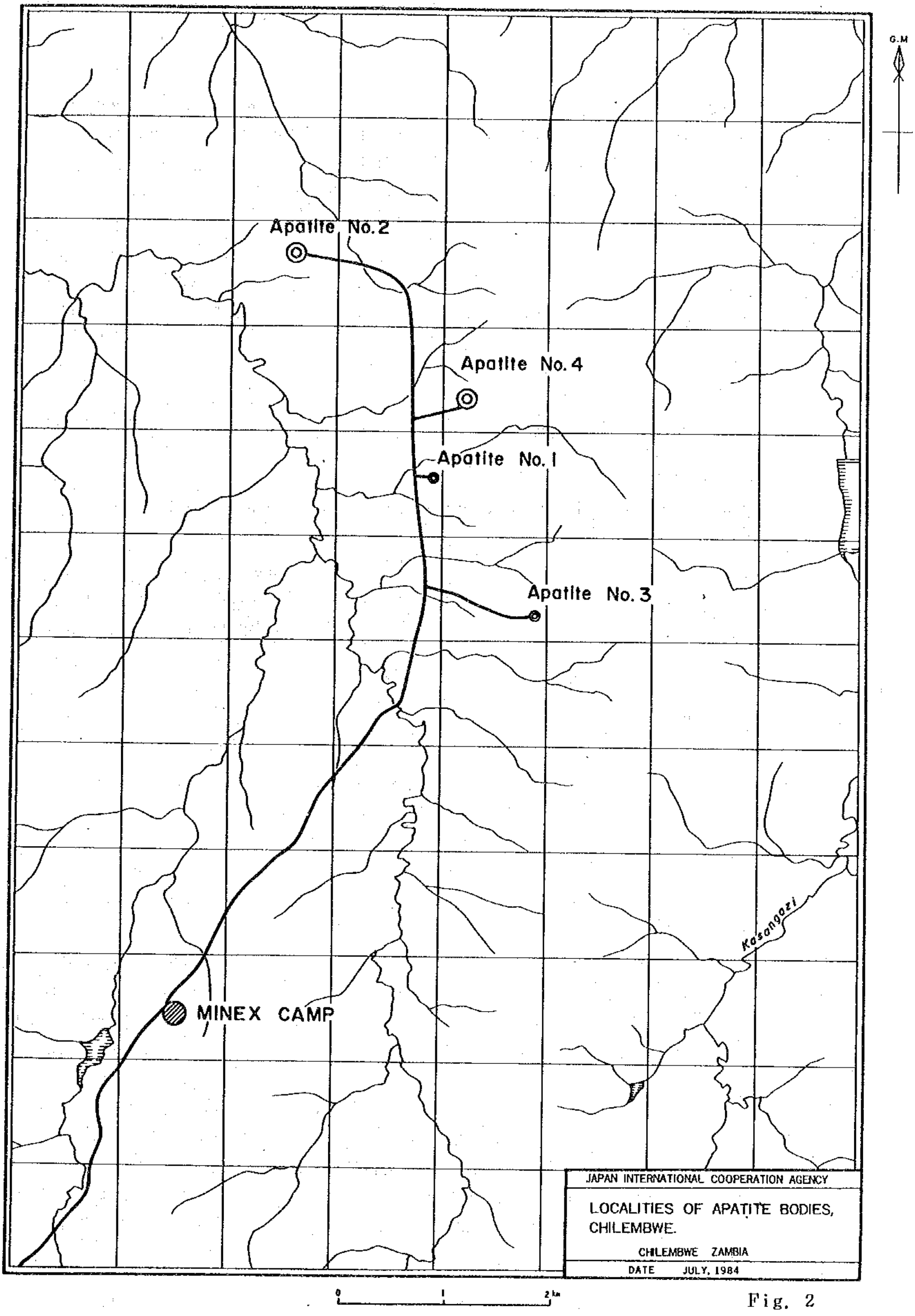

G.M
Apatite No.2
Apatite No.4
Apatite No.1
Apatite No.3
MINEX CAMP
Kasongozi
JAPAN INTERNATIONAL COOPERATION AGENCY
LOCALITIES OF APATITE BODIES,
CHILEMBWE.
CHILEMBWE ZAMBIA
DATE JULY, 1984
0
1
2 km

Fig. 2

2.5 Drilling

Tenders were invited in March, 1984, to undertake the drilling programme and a Romanian company, named Geomin, was selected by MINEX.

Drilling operation started on 28th of June and ended on 8th of August.

The drilling scheme of apatite deposit is tabled as follows:

Orebody	Initial: Number of Holes	Initial: Length Allotted	Actual: Number of Holes	Actual: Length Drilled
No. 2	6	360.0 m	10	534.8 m
No. 4	12	720.0	10	468.7
Total	18	1,080.0	20	1,003.5

2.6 Ore Reserves

Consequently, some 1.6 million tonnes of ore grading 11.8% P_2O_5 have been estimated as follows:

Orebody	Tonnage 1,000 t	Grade %P_2O_5	Contents 1,000 t
No. 2 Orebody	1,421	12.1	172
No. 4 Orebody, shallow horizon	107	10.3	11
No. 4 Orebody, deep horizon	113	9.5	10
Total	1,641	11.8	193

3. MAGNESIUM PROSPECT

3.1 Magnesium Resources

The fertilizer industry of fused magnesium phosphate consumes an apatite concentrate and serpentinite.

Dolomite can be substituted for magnesium resources, hence attention is given to the dolomites in the Basement Complex located north of Petauke and to the Lusaka Dolomites of the Katanga System. In the Lusaka district, several occurrences of dolomite have been recorded in the Cheta Formation and the overlying Lusaka Dolomite.

3.2 Scheme of Investigation

The purpose of the present investigation is to provide basic information, for full-scale prospecting of magnesium raw materials which will be required in case the choice of fused magnesium phosphate is made.

A reconnaissance survey was carried out by MINEX and a prospect named Kyindu Ranch has been taken up. The tentative drilling lengths of 180 m, being 3 holes of 60 m each, were allotted in the Scope of Work.

3.3 Kyindu Ranch Prospect

The prospect is in the vicinity of 15°32'S and 28°31'E, at an elevation of about 1,300 m. The prospect refers to a deposit at the site of Kyindu Ranch, a cattle station some 27 km southeast of Lusaka.

The area is underlain by a thick sequence of grey and white limestone. A dolomite bed crops out ellipstically with a width of 100 m in the north-south direction.

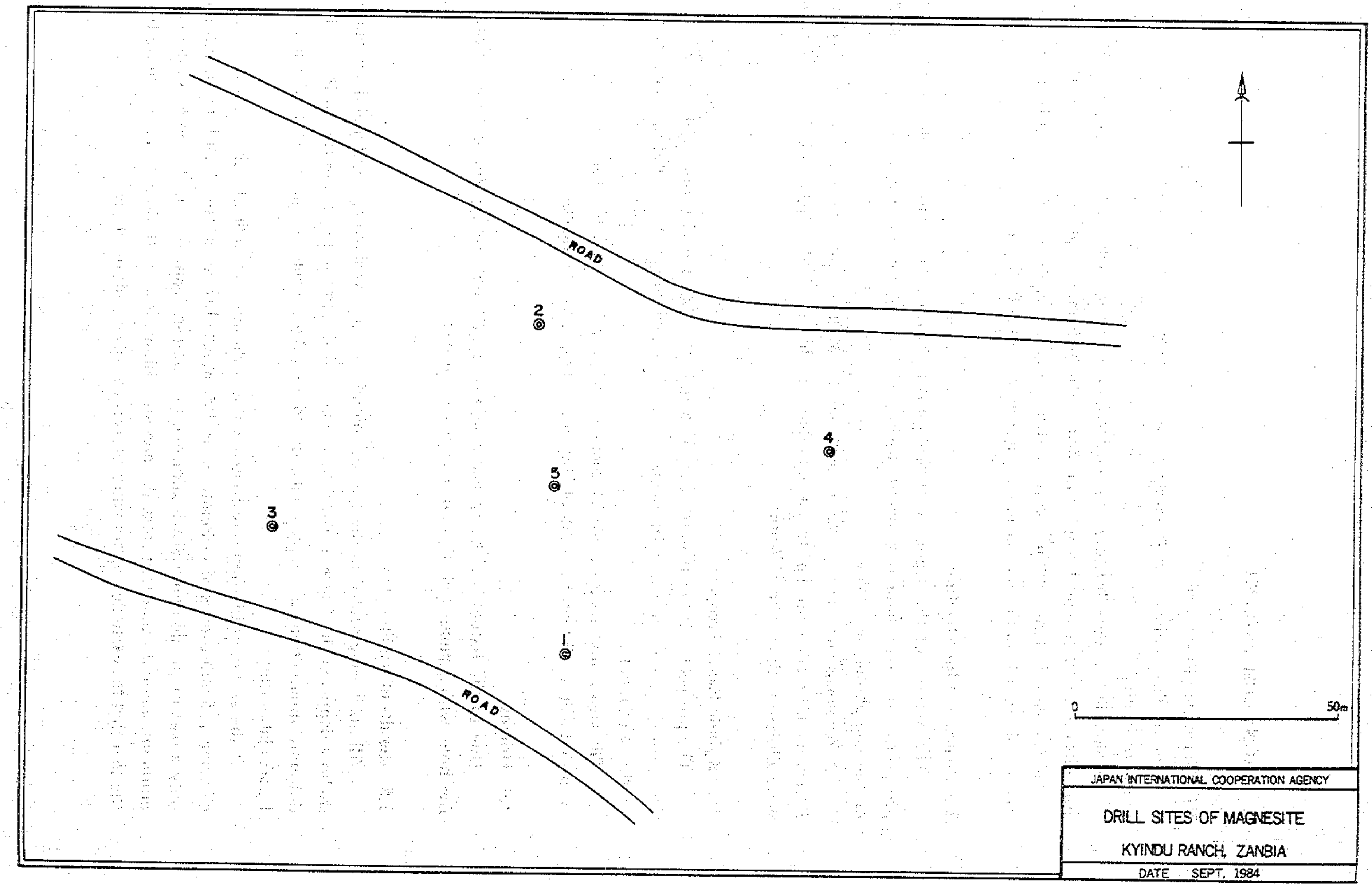
3.4 Drilling

Due to the flatness of the area, the depths of each hole were reduced. Consequently, five holes with a cumulative depth of 180 m were drilled.

3.5 Results of Drilling

All holes were drilled in the area of dolomite occurrences but encountered limestone at shallow depths in the western part. The dolomite bed seems to thicken in the northeast. The dolomite intersections are tabled with the mean values, using the data reported from the Lusaka laboratory.

If the area of estimation is restricted within a rhomb surrounded by these holes, reserves of some 125,000 tonnes of dolomite being 39% $MgCO_3$ can be obtained. But this area forms only a part of the dolomite distribution and the tonnage figure indicated is no more than the minimum amount calculated from the present drilling. The results do not indicate more than the fact that the existence of dolomite reserves has been confirmed.



ROAD
ROAD
2
4
5
3
1
0
50m
JAPAN INTERNATIONAL COOPERATION AGENCY
DRILL SITES OF MAGNESITE
KYINDU RANCH, ZANBIA
DATE SEPT. 1984

Fig. 3

Table 3. Dolomite Intersections

Hole No.	Form m	To m	Run m	$MgCO_3$ %
1	3.75	11.17	7.42	35.85
2	6.25	15.55	9.30	36.67
3	4.00	10.28	6.28	30.18
4	0.00	21.90	21.90	41.37
5	3.00	19.00	16.00	40.25

4. CONCENTRATION TEST

4.1 Sample

About 80 kg samples for a dressing test were taken from the trenches of No. 2 and No. 4 ore body and sent to Japan by air.

A feed sample for concentration tests was prepared by crushing it to a minus 14 mesh.

4.2 Mineralogical Examination

The phosphate mineral was identified as hydroxyl apatite $Ca_5(PO_4)_3OH$. The result of the chemical analysis of phosphate concentrate are as follows:

P_2O_5	T.Fe	Fe^{2+}	S	CaO	MgO	Al_2O_3
34.8%	0.90%	0.28%	<0.01%	48.3%	0.93%	0.74%

SiO_2	Na_2O	K_2O	CO_2	F	Cl
12.40%	0.22%	0.1%	<0.01%	0.78%	0.80%

4.3 Flotation

4.3.1 Comparative test for collectors

In order to select the most suitable collector for Chilembwe ore, five collectors were compared.

The Lilaflot BS#130, Keno-Gard (Sweden), was the most effective.

4.3.2 Flotation test

Aiming at the recovery of a coarser phosphate concentrate, some flotation schemes were applied.

Then, the following conclusions were conducted.

(1) Some coarse apatite particles, +28 mesh, are still recognized in the tailings.

(2) The slime is harmful in regard to stickiness of the froth.

From the results obtained by the preliminary tests, the flowsheet of the overall flotation was determined, for a case of screening with a 150 mesh screen.

Scheme I: Screening with 28 mesh and without regrinding

Scheme II: Screening with 28 mesh and with regrinding

The results are shown in following Table.

Test No.		P$_2$O$_5$ %				Recovery %		
		Feed	Rougher C	Cleaner C	Tailing	WT	R.C.	Cl.C.
Without regrinding	Z-1	21.31	33.51	34.56	8.49	51.3	80.6	80.0
	2	18.30	27.14	29.56	4.56	60.8	90.2	89.7
	3	12.87	28.08	30.16	1.89	41.9	91.5	91.1
	4	20.47	32.29	34.36	5.54	55.8	88.0	86.8
	5	20.11	33.40	34.72	5.53	52.3	86.9	85.2
	6	16.85	29.71	31.77	4.07	49.9	87.9	84.8
	7	18.23	33.30	35.01	10.12	35.0	63.9	63.4
	8	16.53	26.82	29.59	3.35	56.2	91.9	89.2
With regrinding	9	21.19	33.41	36.81	8.58	50.8	80.1	78.1
	10	18.48	28.28	36.96	4.36	59.0	90.3	87.2
	11	12.24	24.98	31.20	1.61	45.5	92.8	92.0
	12	20.59	33.65	38.81	7.26	50.5	82.6	79.6
	13	19.73	30.73	35.95	5.38	56.6	89.4	86.3
	14	17.18	30.14	34.42	3.73	50.9	86.1	87.5
	15	19.57	33.24	37.26	5.53	50.7	86.1'	85.0
	16	17.89	30.11	37.13	3.54	54.0	90.0	88.0
Ref.	F201	21.56	29.82	–	5.31	66.3	91.7	–
	F202	21.56	30.30	–	6.09	63.9	89.8	–

Remarks: Z-1 & 9 (No. 2), 2 & 10 (No. 4), 3 & 11 (Weathered), 4 & 12 (No. 2 3:1 No. 4), 5 & 13 (No. 2 + 10%W), 6 & 14 (No. 4 + 10%W), 7 & 15 (No. 2 + 20%W), 8 & 16 (No. 4 + 20%W), F201 & 202 (8 Min. & 5 Min. Grind)

The regrinding of rougher concentrate has little effect on the rate of recovery of the final concentrate. However, the regrinding has some positive effect on a final concentrate grade.

With regrinding, all the concentrates have a grade higher than 30% P_2O_5, generally exceeding 35%. On the contrary, this process has an unfavourable effect on the production of the –150 mesh fraction. On the average, the regrinding increases the –150 mesh fraction by about 4%.

Although the regrinding of rougher concentrate has some bad effect on the production of fine particles, this process may be indispensable to maintain the concentrate grade.

4.3.3 Medium size flotation test

A feed sample for this test was prepared by combining a sample from No. 2 Orebody with one from No. 4 Orebody. The combined size was –14 mesh, its weight 27 kg. The sample was then fed to a 300 X 300 mm F.W. Flotation machine (one cell). The feed sample was sieved with a 48 mesh screen, and the +48 mesh fraction was ground by a rod mill.

The minus 48 mesh fraction and mill product were fed to the flotation machine.

The results are given in following Table.

Circuit	Product	Weight (g)	$\% P_2O_5$	P_2O_5 (g)	Recovery %	
					Wt.	P_2O_5
Rougher (1)	Feed	23,877	17.55	4,191	88.4	89.9
	Conc.	10,630	38.62	4,105	39.4	88.0
	Tail.	13,247	0.65	0,086	49.0	1.9
Slime flot. (2)	Feed	3,123	15.15	0,473	11.6	10.1
	Conc.	1,577	29.17	0,460	5.9	9.8
	Tail.	1,546	0.86	0,013	5.7	0.3
Scavenging (3)	Feed	13,247	0.65	0,086	49.0	1.9
	Conc.	1,590	3.68	0,058	5.9	1.3
	Tail.	11,657	0.24	0,028	43.1	0.6
Total (1 + 2 + 3)	Feed	27,000	17.28	4,664	100.0	100.0
	Conc.	13,797	33.51	4,623	51.2	99.1
	Tail.	13,203	0.32	0,042	48.8	0.9

The grade and recovery of the rougher concentrates showed fairly good results; it grades 38.62% P_2O_5 with an 88.0% recovery.

The final concentrates including the concentrate from slime flotation and scavenger flotation have a grade of 33.5% P_2O_5 with 99.1% recovery.

4.4 Expected Performance

Based on the results of the overall flotation test, the medium sized laboratory test and the estimated feed grade from the calculation of minable ore, the following performance can be expected for Chilembwe ore.

	Assay %	Recovery %	
	P_2O_5	Wt	P_2O_5
Feed	11.50	100.0	100.0
Conc	30.07	33.8	88.4
Waste	2.01	66.2	11.6

Comparing this performance with that of the "Interim Report", the increase in recovery was about 5%.

The increase is made by the introduction of slime flotation. In the other report, P_2O_5 in the slime was disposed into the waste.

5. MINING

The production scale is based on;

(1) the estimation that the required amount of P_2O_5 is 20,000 T/year in Zambia.

(2) the proposed production which amounts to 50% of the annual required P_2O_5.

(3) the limited capacity for the water storage of Mankwala Dam at the end of the dry season.

Calculation:

Minable ore grade	11.5%
Processing recovery	88.4%
Required production	(20,000 × 0.5) ÷ (0.884 × 0.115) = 98,400 T/year (378.5 T/day)
Proposed production	400 T/day × 260 days/year = 104,000 T/year

It is impossible to obtain a scale merit because of the limited dam capacity. (The storage capacity of Mankwala Dam at the end of the dry season; 225,000 T)

However, recycled water is used as much as possible. (See Chap. 6 & 7)

5.1 Mining Method

Open pit mining is selected for the following reasons:

(1) The deposits are found near the surface.

(2) The maximum depth of the deposits is about 60 m from the surface.

(3) An adequate waste dump area is available nearby.

(4) The stripping ratio is within an economical allowance.

(5) Rain falls for only a short time even during the rainy season.

5.2 Selection of Equipment

Selection of equipment as to the make, model and size, and determination of the number of required units is based on their performances. Small size equipment is selected to suit the pit size and the scale of production.

5.3 Ultimate Pit Design (Refer to Fig. 5.1 ~ 5.4)

The following criteria are used in designing the ultimate pit.

(1) The final pit slope of 45° is decided because the geotechnical data is limited. However, the study of slope stability should be continued during the mining practices to seek an applicable steeper angle for the pit slope to minimize the overall stripping ratio and the operating costs.

(2) A bench height of 5 m and its slope of 70° are selected, considering the slope stability, the waste-ore ratio and the performance of the 2.2 cu.m. dozer shovel.

(3) Ore reserves of No. 2 Orebody and the shallow part of No. 4 Orebody are chosen for the present minable ore calculation. These figures are then converted to those of minable ore reserves, using the criteria as mentioned above: 1,551,000 T with an average grade of 11.5% P_2O_5 and a stripping ratio of 2.16. (Refer to Table 5.1)

5.4 Mining Plan

The mining term is divided into the following three stages:

(1)	Preparatory Stage	Pre-production stripping
(2)	Production Stage I	Production from the No. 2 Orebody above the 860 m level.
(3)	Production Stage II	Production from the No. 2 & No. 4 Orebody.

(1) Preparatory Stage

To avoid the rainy season, pre-production stripping will start from the first year prior to the commencement of production.

Within a year, an area for extracting crude ore for a 6 month period can be exposed. The amount of pre-production stripping is determined for minimizing the initial investment. The total volume of overburden to be removed is 124,000 cu.m., containing 52,000 T of ore which will be stockpiled separately for trial production.

(2) Production Stage I

This stage will last for 5 years. An average grade during this period is estimated to be 11.5% P_2O_5 with a stripping ratio of 2.1.

(3) Production stage II

This stage will last for 10 years. An average grade during this period is estimated to be 11.5% P_2O_5 with an overall stripping ratio of 1.95 (2.1 from 8th to 15th year, 1.83 in 16th and 0.27 in 17th year).

Exploration data indicates the existence of other orebodies such as No. 1 and No. 3 Orebodies and the deep horizons of No. 4 Orebody.

Due to the limitation of available data, these deposits are excluded from the present evaluation. However, the No. 1 Orebody seems to be minable without difficulties to a certain depth within an allowable waste-to-ore ratio.

The No. 3 Orebody has a narrow and steep dipping dyke form. As this orebody is of a high grade of ore, it may be possible to extract only a shallow part of this deposit, which helps easier control of the grade of feed even with its lesser amount. In this case, an appropriate pit slope should be determined through the mining practice, although an applicable steepness will limit the minable ore amount that can be economically extracted.

5.5 Operational Plan

For the concrete supply of ore and maintaining the grade to the concentrator, it is preferable to increase the number of benches. The greater the number of benches, the easier

Table 5.1. Minable Ore Reserve (No. 2 and No. 4 shallow Deposits)

( ) ore grade %

Block \ Level	Surface ~885L	~880	~875	~870	~865	~860	~855	~850	~845	~840	~835	~830	Total
DDH 6	(17.8) 23,410	(12.9) 13,770	(7.3) 13,770	(9.4) 13,770	(15.2) 13,770	(13.2) 13,770	(13.2) 13,770	(13.2) 13,770					(13.2) 119,800
DDH 7	(9.7) 22,630	(9.4) 12,570	(2.7) 12,570	(17.1) 12,570	(11.4) 12,570	(11.4) 12,570	(8.6) 12,570	(6.7) 12,570	(5.3) 12,570				(9.2) 123,190
DDH 9	(13.0) 41,180	(11.1) 25,740	(16.2) 25,740	(19.3) 25,740	(13.3) 25,740	(12.4) 25,740	(20.0) 25,740	(14.8) 25,740	(11.8) 25,740	(7.7) 25,740	(9.5) 25,740	(10.0) 25,740	(13.2) 324,320
DDH 11		(11.9) 15,330	(8.6) 15,480	(8.5) 15,480	(5.3) 15,480	(11.9) 15,480	(8.0) 15,480	(11.1) 15,480	(9.3) 15,480	(9.3) 6,190			(9.3) 129,880
DDH 12				(8.0) 5,700	(8.0) 5,700	(8.0) 6,040							(8.0) 17,440
II 2		(16.2) 8,400	(9.5) 16,470	(8.0) 16,470	(15.6) 16,470					(5.3) 5,280			(11.2) 63,090
II 4					(7.0) 14,880								(7.0) 14,880
II 5		(20.0) 7,940	(9.9) 19,370	(14.0) 19,370	(7.8) 19,370	(2.9) 19,370	(5.0) 19,370	(29.1) 19,370	(27.1) 19,370	(24.4) 19,370	(21.3) 19,370	(25.4) 21,300	(16.9) 203,570
II 6		(13.2) 15,780	(7.2) 23,420	(12.6) 23,420	(11.9) 23,420	(8.8) 23,420							(10.6) 109,460
II 7	(16.8) 9,800	(7.3) 13,880	(8.8) 13,880	(15.3) 13,880	(14.2) 13,880	(9.4) 13,880	(13.5) 13,880	(10.0) 13,880	(16.6) 13,880	(12.9) 13,880			(12.3) 134,720
II 8				(9.6) 38,130	(4.8) 38,130	(5.1) 38,130	(10.6) 38,130	(6.4) 38,130					(7.3) 190,650
II 10							(5.7) 11,720						(5.7) 11,720
IV (No. 4 Orebody)	(10.0) 108,260												(10.0) 108,260
Total	(11.8) 205,280	(12.1) 113,410	(9.4) 140,700	(12.4) 184,530	(10.0) 199,410	(7.9) 168,400	(11.2) 150,660	(12.7) 138,940	(14.6) 87,040	(13.3) 70,460	(14.6) 45,110	(17.0) 47,040	(11.5) 1,550,980T

it is to control the ore grade. However, increasing the number of benches increases the initial investment and operating costs by rising the waste ore ratio. Therefore, a three bench operation system is selected.

Daily production will be 400 T; ore blasting is every three days with the other days being for waste removal.

Stockpiled ore in the yard will be used in the concentrator during the non-extractable term of the rainy season.

Drilling Blast holes of 4"ϕ will be drilled by a hydraulic crawler-drill with spacings of 2.33 and 3.0 m in two rows and to a depth of 5.71 m, including 0.71 m of sub-drilling.

Blasting Most holes are expected to be dry, and a mixture of AN-FO is used. Slurry or dynamite explosives are provided only for the blast holes that cannot be dewatered. The powder factor for 4"ϕ holes is 267 g/t, including a 10% extra for secondary blasting.

Loading Blasted ore and waste are loaded by a dozer-shovel of a 2.2 cu.m. capacity.

Hauling The 20 t rear dump trucks are the main units for hauling the ore to the primary crushing plant and the waste to the dump area. Maintenance of all haulage roads in good condition at all times is one of the most important factors to reduce the costs of tires and truck maintenance. A fleet of the road maintenance equipments consisting of a grader, a bulldozer and a water cart will be provided for this purpose.

Drainage The greater part of the precipitation falls in a three month season. During the pre-stripping period, the surface water is drained through ditches. For the purpose of dewatering in the pit, a pump is used below the 885 meter leve.

Grade-Control The cuttings of drill holes should be collected and assayed prior to blasting, not only to discriminate ore from waste but also to find the grade of ores at each working face. This data will serve for the grade control of the daily operation.

Service Vehicle A service truck and pick-ups are provided for surveying, maintenance and supervision, etc., to support smooth and efficient operations.

Facilities Magazines to store a 6 month supply of explosives, AN-FO and detonators will be installed.

A repair shop for the heavy duty equipment will be constructed at the mine site.

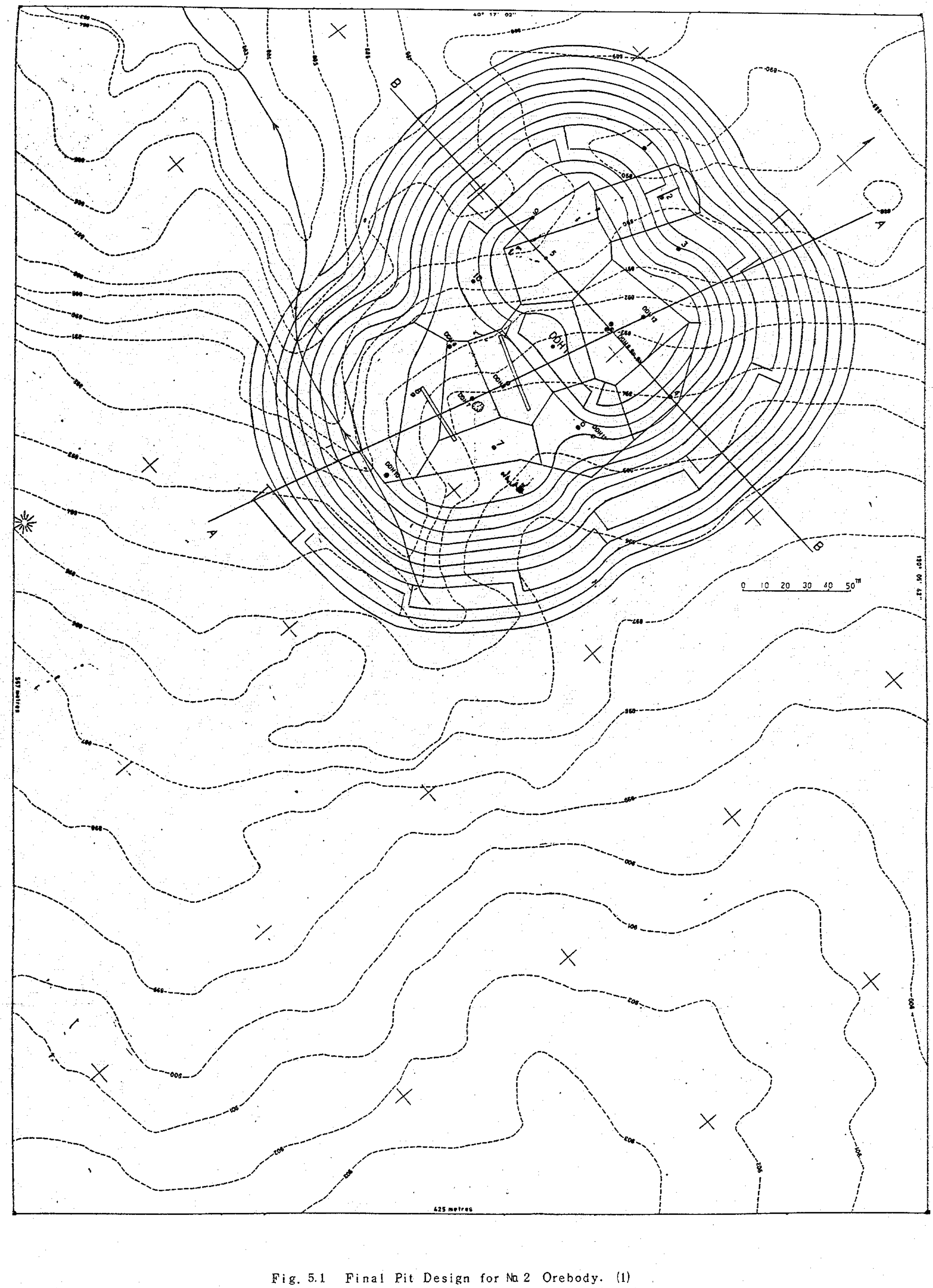
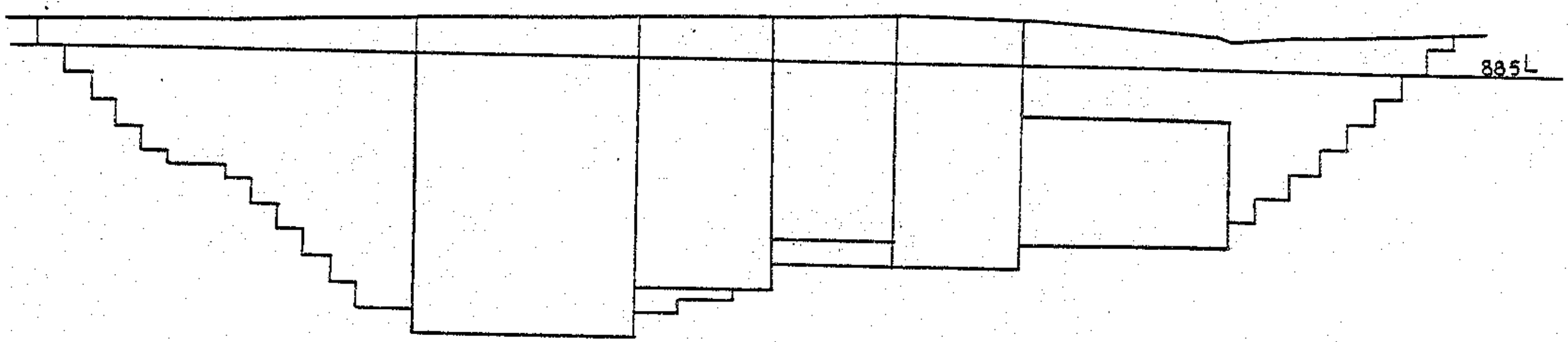


Fig. 5.1 Final Pit Design for № 2 Orebody. (1)

A—A SECTION



B—B SECTION

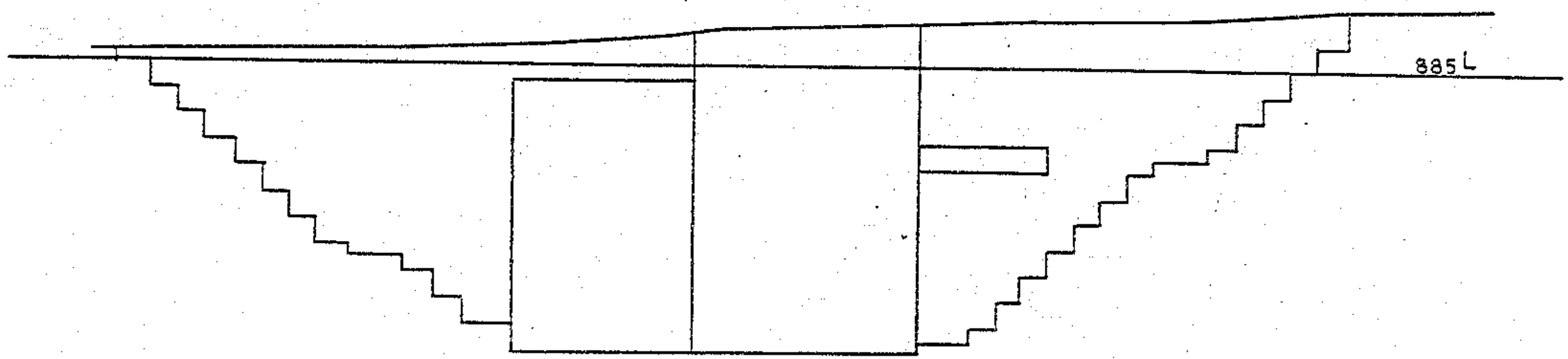


Fig. 5.2 Final Pit Design for No.2 Orebody (2)

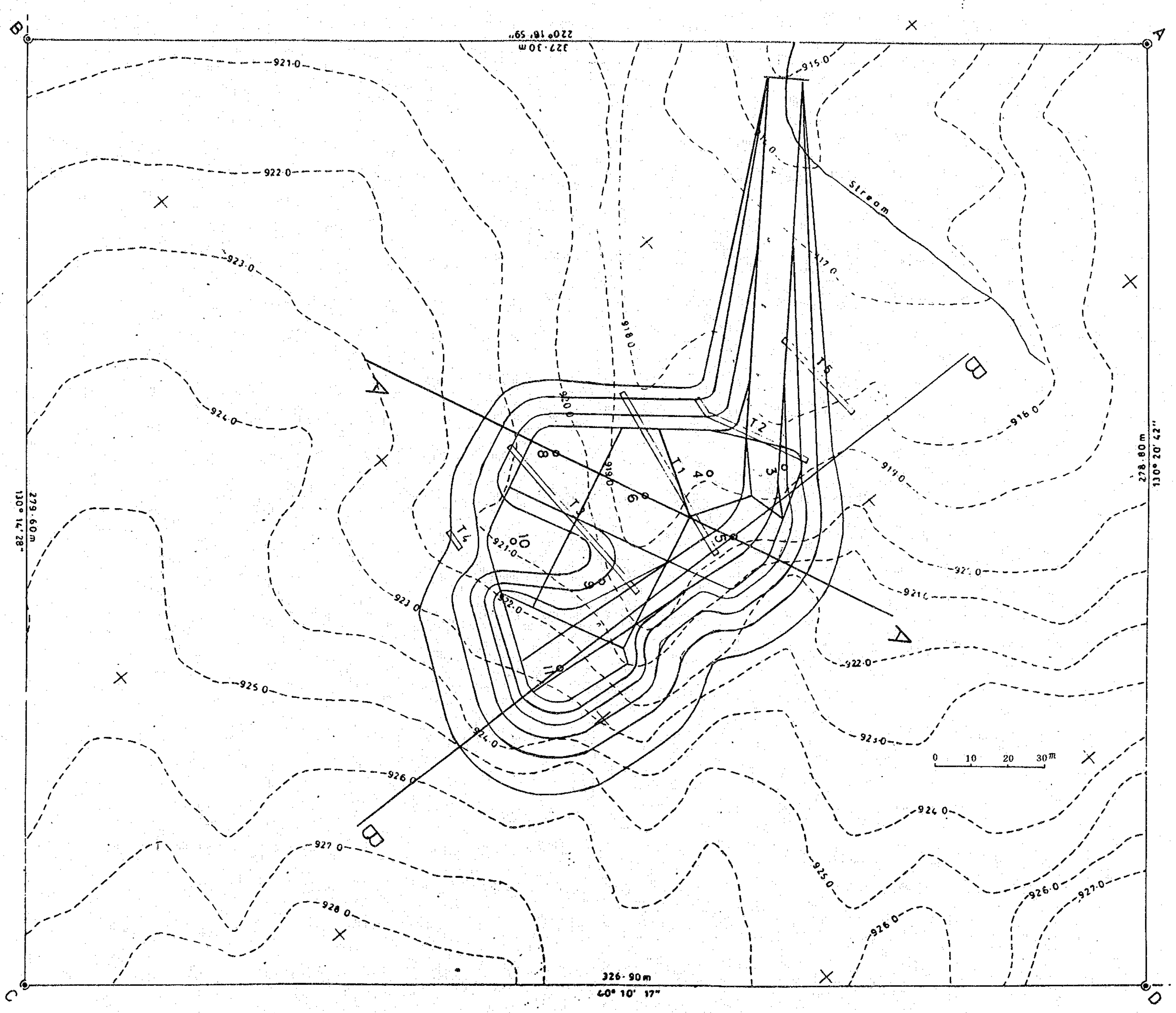
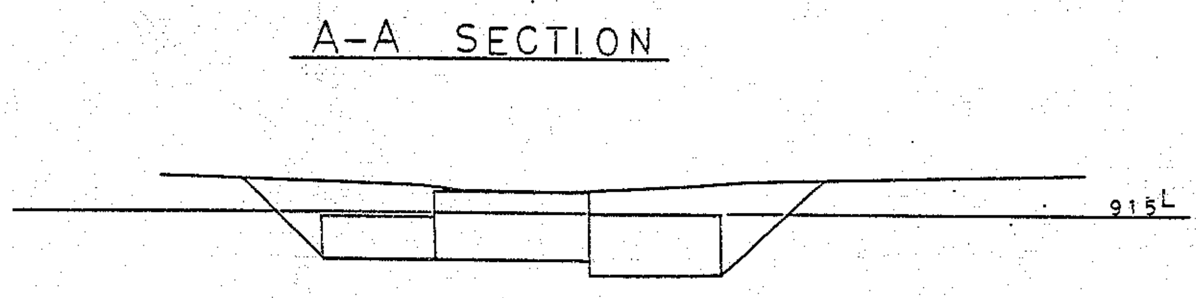


Fig. 5.3 Final Pit Design for № 4 Shallow Orebody. (1)



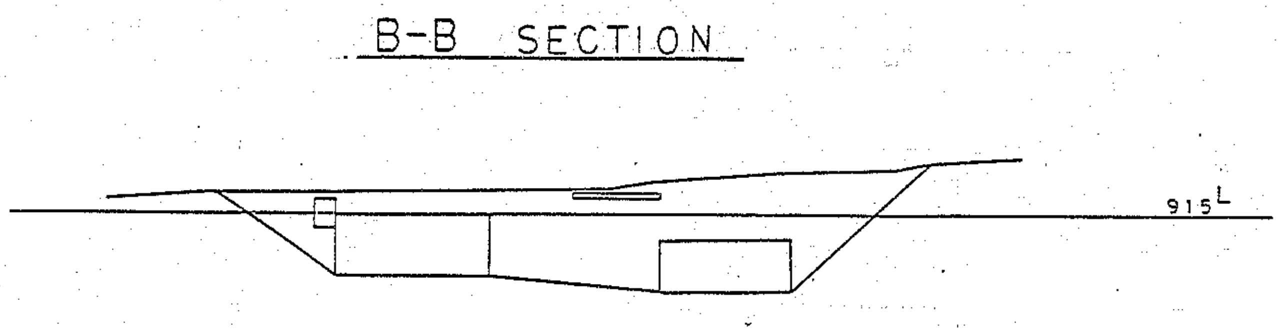


Fig. 5.4 Final Pit Design for No.4 Shallow Orebody (2)

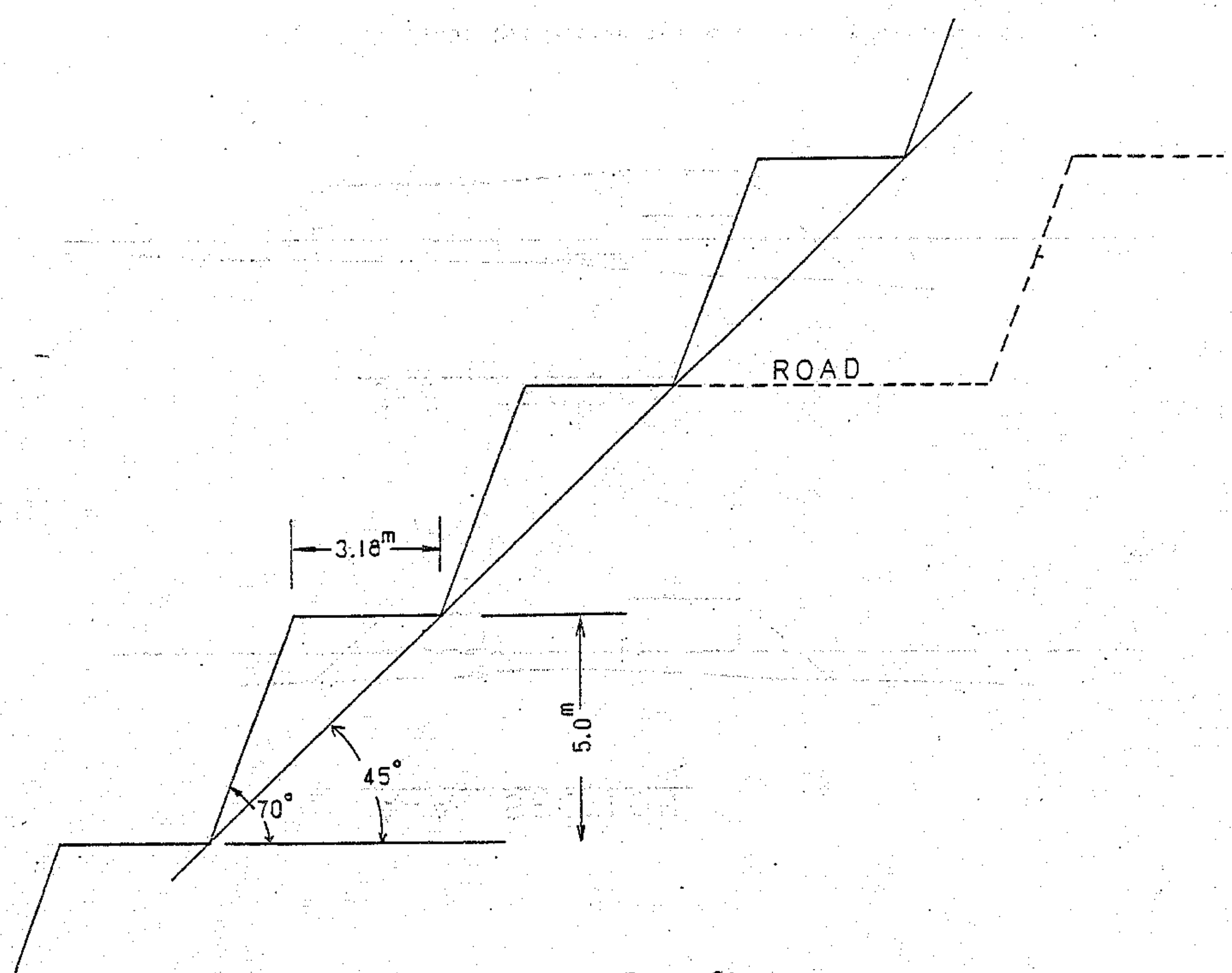


Fig. 5.5 Final Bench Slope

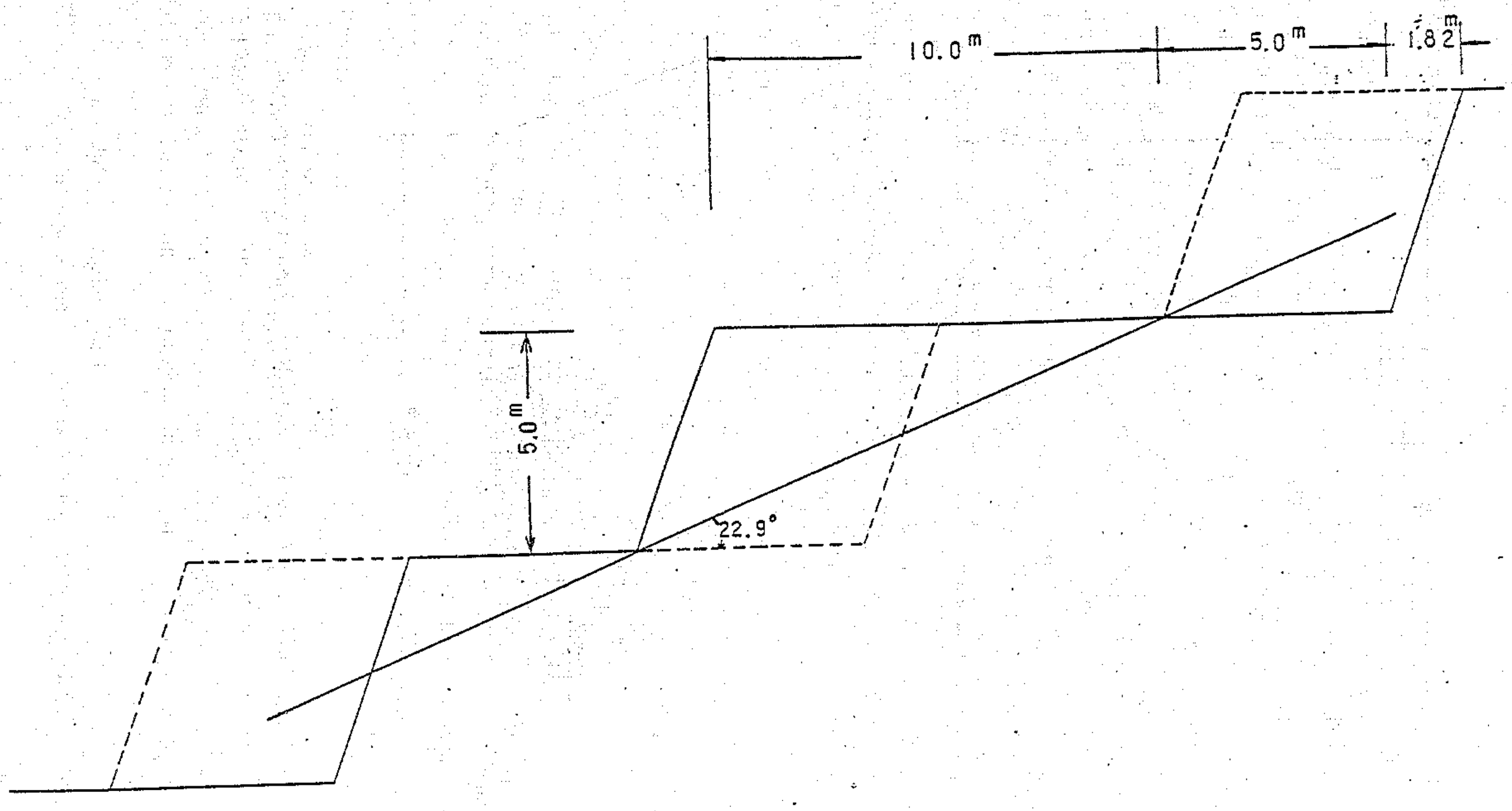
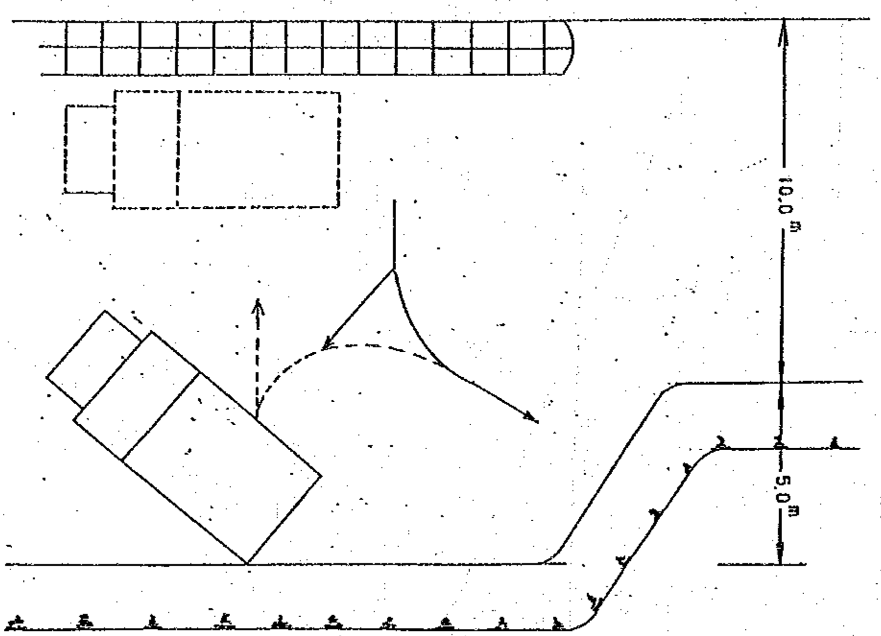


Fig. 5.6 Working Bench Slope



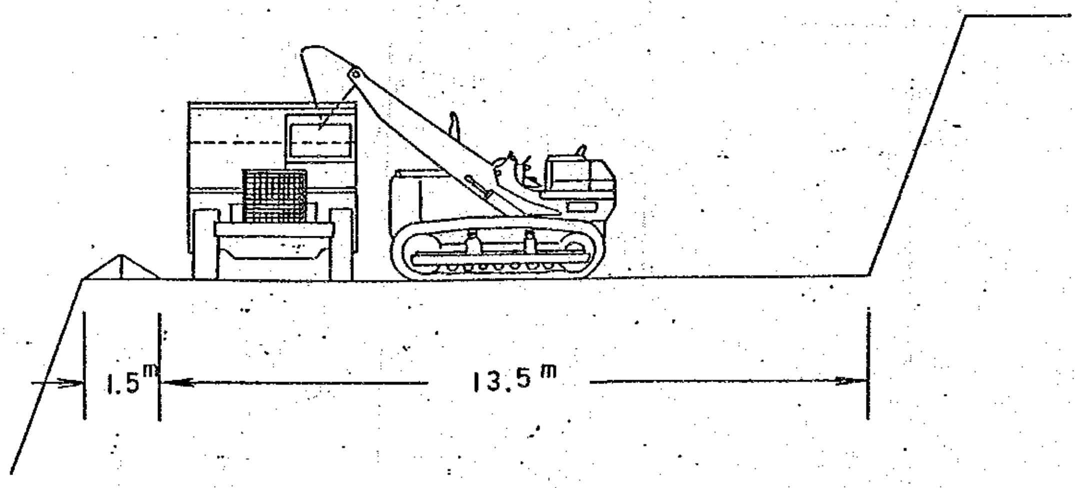


Fig. 5.7 Relationship of Equipment Size, Benchspacing and Operating Room for 20 ton Truck and 2.2 cu.m Dozer Shovel

6. CONCENTRATION

6.1 Outline

The concentrator is designed for all-slime flotation and will treat 104,000 T in 290 working days annually producing phosphorous concentrates.

To allow time for repairs, the maximum treatment capacity of the concentrator will be 400 T/day.

The annual working days of the extraction are fixed at 260 days because of provision against unworkable days during the rainy season.

The production will be stabilized by the supply from the stock pile installed between the pit and the primary crushing plant.

The concept of the concentration design is as follows; the pulp will be treated in high density on the flotation, and the primary slime will be eliminated to avoid a bad influence on the flotation. These two criteria will fairly reduce operation cost.

To recover the used water as much as possible because of water shortage in this area, the amount of re-used water should be increased.

The location of the concentrator is decided, considering the topography, the foundation of the rock, the direction of the wind, the ultimate pit location and the tailing pond location.

6.2 Design Parameter

According to the laboratory tests, pulp density and slime elimination were critical factors for the concentration design.

The followings are the design parameters to reduce initial and operating costs and facilitate the operation of the concentrator.

(1) Process will be as simple as possible.

(2) Instrumentation will be installed at principal points.

(3) The structure of the concentrator will be as simplified as possible, and the machinery is situated as efficiently as possible considering the aspect of its maintenance and control.

(4) The case of operation of the machinery and facilities will be given important consideration.

6.3 Process

6.3.1 Primary crushing plant

The run of mine will be transported from the pit to the primary crushing plant where it will pass over a grizzly.

The oversize ore is crushed by a primary crusher and combined with the undersize ore from the grizzly. The ore will be transported to the coarse ore storage bin.

6.3.2 Secondary-tertiary crushing plant

The ore from the coarse ore storage bin is drawn and screened by a secondary screen.

The oversize ore is crushed by a secondary crusher and transported to a tertiary screen.

The undersize ore is fed to a double deck washing screen with 15 mm and 5 mm openings.

The +15 mm ore is transported to a tertiary screen, while the −15/+5 mm ore is transported to the fine ore storage and the −5 mm ore (pulp) is fed to a spiral classifier.

The sand from the classifier is transported to the fine ore storage and the overflow from the classifier is sent to the flotation plant by a pump.

The oversize ore from the tertiary screen is crushed by a cone crusher and this product is recycled to the tertiary screen by a belt-coveyor. The undersize ore from the tertiary screen is transported to the fine ore storage.

6.3.3 Grinding

The grinding circuit comprises a ball mill and a spiral classifier which is operated in a closed circuit of a wet grinding system.

The 80% pass size of the ground ore is 0.3 mm and the pulp density is 50% solids.

6.3.4 Flotation

The flotation circuit is comprised of a rougher, a scavenger, a cleaner and a slime flotator.

The froth of scavenger is sent back to the rougher and the tailing becomes the final tailing.

The slime from the crushing circuit is fed to a cyclone. For the recovery improvement the underflow is fed to the slime flotation, and the clayey overflow is eliminated, because it causes some bad influences on the flotation.

6.3.5 Thickening and filtering

The P_2O_5 concentrate froth is thickened in a thickener.

The thickened concentrate is filtered by a drum-filter and the filtered cake is stored in the stock yard for concentrates.

6.3.6 Tailing

The tailing of the scavenger is sent to a tailing pond.

The overflow of the tailing pond is re-cycled and re-used in the concentrator.

6.4 Tonnage and grade control

The tonnage of the run of mine and the concentrate are weighed by the conveyor scale and the truck scale.

The grade control is obtained by the samples of the final concentrate and the tailing, according to which the crude ore grade is calculated.

Table 6 Legend for Flow Sheet

No.	Equipment	Size & Spec.	Number	Power (kW)	Remarks
1	HOPPER	50t	1		
2	GRIZZRY FEEDER	900 x 2,400	1	7.5	100mm
3	PRIMARY CRUSHER	900 x 600	1	55	Single toggle
4	B.C.	600	1	11	Belt conveyor
5	ORE BIN	300t	1		
6	APRON FEEDER	900 x 3,000	1	2.2	
7	B.C.	400	1	3.7	
8	2ND SCREEN	900 x 1,800	1	5.5	60mm
9	2ND CRUSHER	760 x 300	1	37	Single toggle
10	B.C.	400	3	7.4	
11	3RD SCREEN	1,200 x 2,400	1	7.5	15mm
12	3RD CRUSHER	900ϕ	1	55	Cone crusher
13	B.C.	400	2	3.7	
14	B.C.	400	3	5.2	
15	WASHING SCREEN	1,200 x 2,400	1	7.5	15mm, 5mm Double
16	SPIRAL CLASSIFIER	600ϕ x 4,500	1	2.2	
17	B.C.	400	1	1.5	
18	B.C.	400	1	1.5	
19	B.C.	400	1	5.5	
20	ORE BIN	400t	1		
21	BELT FEEDER	400	1	1.5	
22	B.C.	400	1	2.2	
23	BALL MILL	2,400 x 1,800	1	150	
24	SPIRAL CLASSIFIER	1,050ϕ x 6,800	1	3.7	

No.	Equipment	Size & Spec.	Number	Power (kW)	Remarks
25	PUMP	3/2 WP	2	3.7 x 2	
26	CONDITIONER	2,000ϕ x 2,000	2	5.5 x 2	
27	ROUGHER	#24 FW	8	11 x 8	
28	PUMP	3/2 WP	2	5.5 x 2	
29	CYCLONE	200ϕ	2		
30	SCAVENGER	#24 FW	6	11 x 6	
31	PUMP	3/2 WP	2	2.2 x 2	
32	CLEANER	#24 FW	8	11 x 8	
33	PUMP	3/2 WP	2	3.7 x 2	
34	PUMP	1 WP	2	2.2 x 2	
35	THICKNER	10mϕ	1	1.5 + 0.4	
36	PUMP	3/2 WP	2	5.5 x 2	
37	CYCLONE	200ϕ	2		
38	CONDITIONER	2,000ϕ x 2,000	1	3.7	
39	SLIME FLOTATOR	#21 FW	6	3.7 x 6	
40	PUMP	3/2 WP	2	3.7 x 2	
41	FILTER	2,400ϕ x 3,600	1	2.2 + 5.5	
42	STOCK YARD	600t	1		
43	PUMP	3/2 WP x 2	4	22 x 4	
44	PUMP	3/2 WP	2	15 x 2	

Fig. 6.1(1) FLOW SHEET(Crushing)

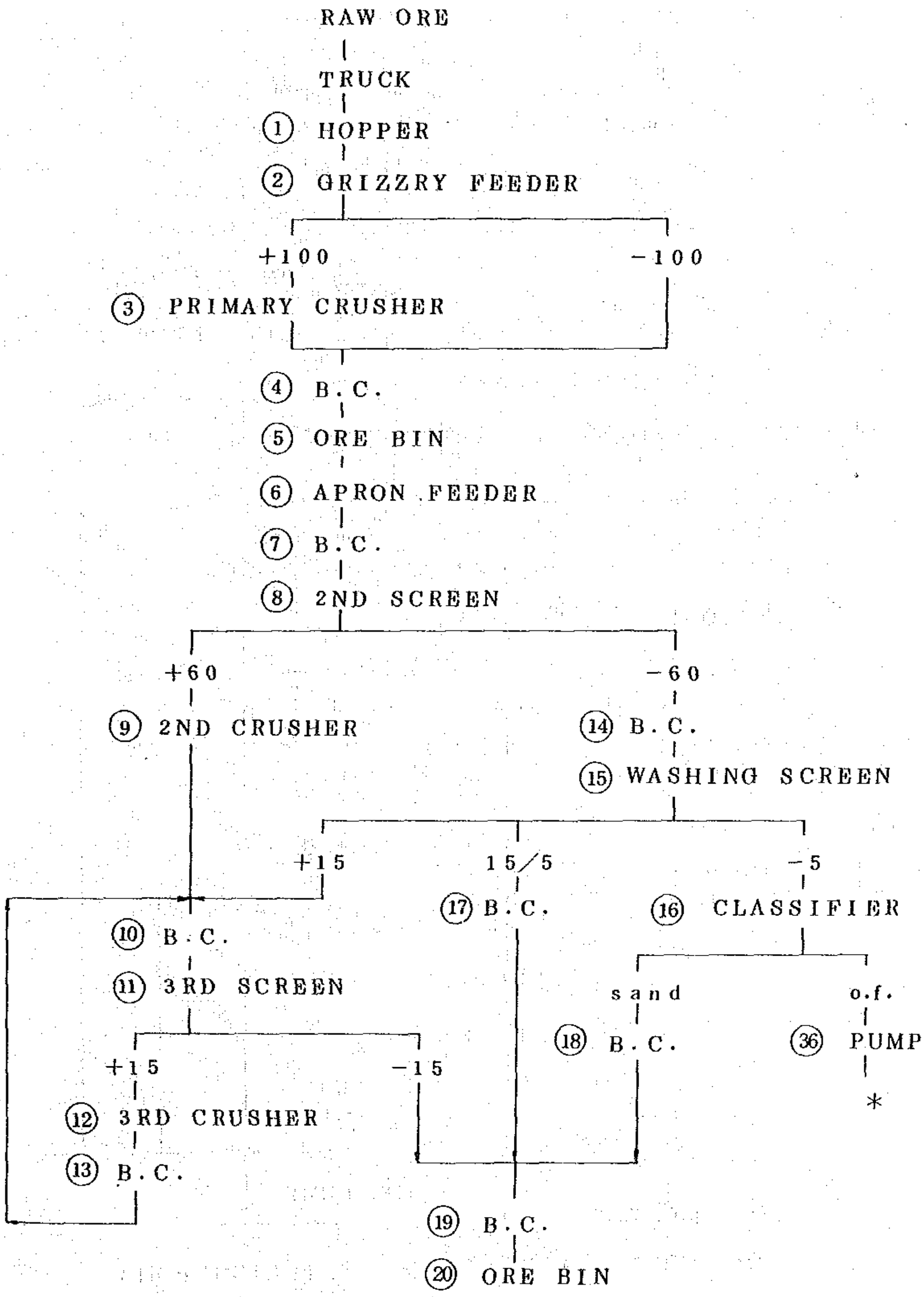
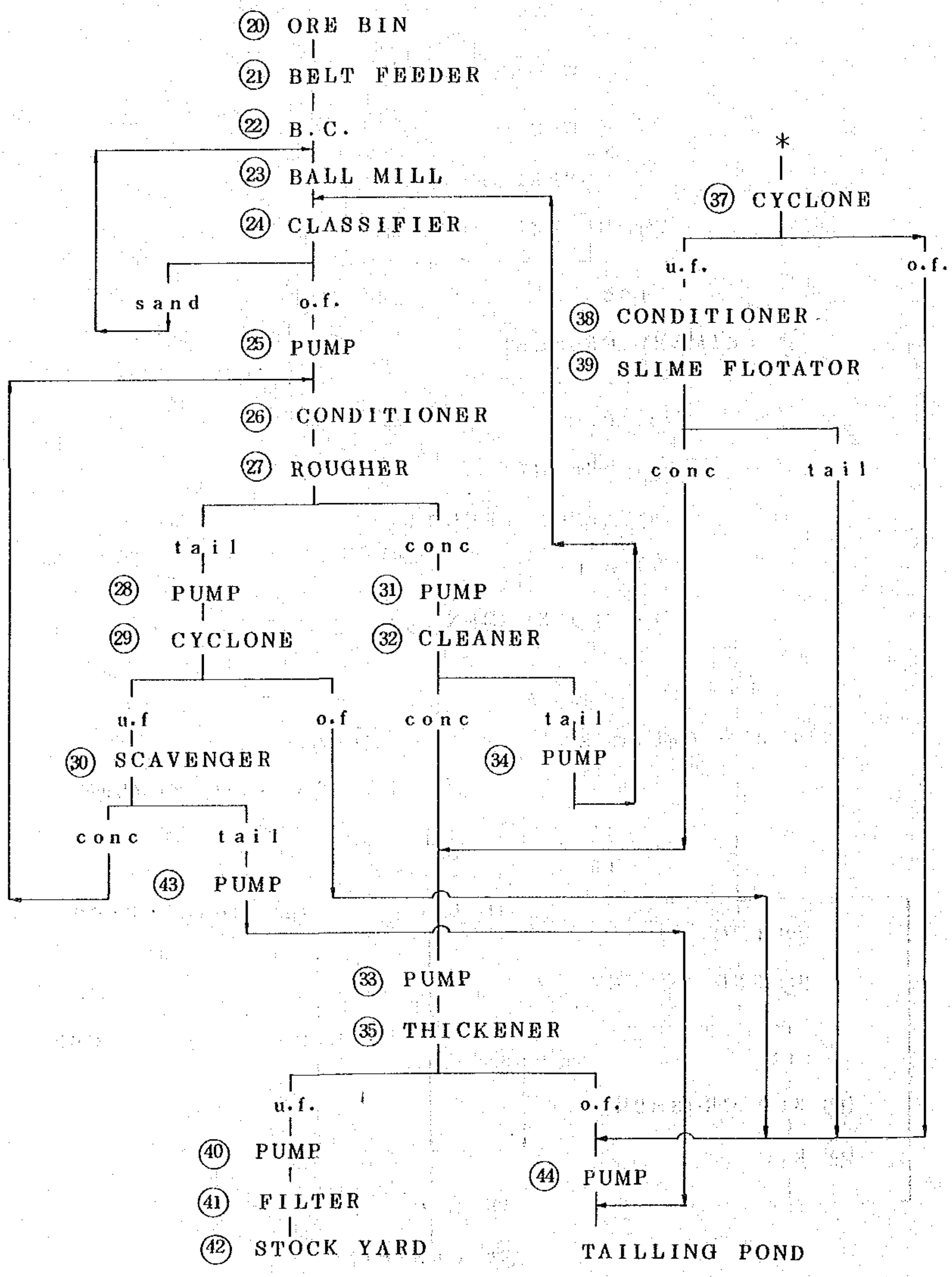


Fig. 6.1(2) FLOW SHEET (Grinding and Flotation)



6.5 Process Water

The process water consists of the overflow of the tailing pond and the fresh water. To recover the used water as much as possible because of water shortage in the area, the amount of re-used water should be increased.

The recycled water recovery from the tailing pond is possible because;

(1) The settling velocity of fine particles is so slow that the big thickener is necessary to settle the fine particles. The tailing pond serves this purpose.

(2) The reagent in the water is decreased by the absorption, resolution and dilution in the tailing pond.

(3) The tailing pond is constructed close to the concentrator.

6.6 Reagents

The reagents for the flotation process are caustic soda, Lilaflot and water glass.

6.7 Instrumentation

Instruments for monitoring, indicating and recording will be installed at each necessary point.

(1) For the weighing of ore drawn from storage, three weight meters will be installed.

(2) For the determination of the volume in the washing and grinding section, two flow meters will be installed.

(3) For the measurement of a pulp pH value in the flotation process, two pH meters will be used.

(4) For the determination of pulp density in the grinding section, one density meter.

6.8 Miscellaneous

The dust arresters, samplers, overhead cranes, store rooms and an office will be provided.

6.9 Analysis and Laboratory

An analysis and laboratory room will be built adjacent to the concentrator; these facilities will be used for other purposes as well as for the concentrator.

7. ANCILLARIES

7.1 Power Supply

7.1.1 Outline

There will be two power distribution systems, one for mine area where the pit, concentrator, and various facilities will be located and the other for Mankwala Dam where the pump station for the water supply will be located. The power line to Mankwala Dam will be diverged from the substation at the mine site. A cost of generating electricity in a private sector exceeds a cost of power purchased from an electric company. The power is deemed to be introduced from outside.

7.1.2 Power source

Power will be purchased from Zambia Electricity Supply Corporation Limited (ZESCO) at the prevailing rates.

Between the mine site and Katete substation, the power line with 33 kV will be constructed, and maintained by ZESCO.

7.1.3 Power distribution

(1) Power requirement:

The maximum power required for the production and resident facilities will be 790 kw and the total annual requirement will be 3,317,800 kwh.

(2) Voltage used:

The power line from Katete to the mine site substation will be of 33 kV. At the mine site substation, the voltage will be stepped down to 6,600, 440 and 220 V.

(3) Emergency power supply:

Emergency power facilities will not be constructed, because the demand of electric power is not so great as to disturb production.

7.1.4 Cost of purchased power

Applicable code:

Section 12 of Electricity Act. Chapter 811 of the Laws of Zambia. Tariff D 2 (effective 1st May, 1983)

Estimated unit cost	0.015 US$/kwh
Annual power cost	50,900 US$/year

7.2 Water Supply

7.2.1 Outline

The facilities will supply water both for the process and the domestic use, comprising of intake, heading-water, purification and distribution.

7.2.2 Water requirement

Required quantity is 1,820 m^3 per day for all purposes. But the necessary amount of intake water is 855 m^3 per day. (Refer to Fig. 7.2.1)

(1) Process water

Total process water requirement will amount to 1,700 m^3, out of which about 99% will be used for the concentration alone.

About 57% of the required amount will be recycled in the plant, because the Mankwala source has not enough capacity. Therefore, required amount of fresh water is 735 m^3/day.

(2) Water for domestic use

Required quantity is 120 m^3/day. And 83% of it will be used at the resident area. The proposed population of the resident area will be 500 including families and its daily consumption is set at max. 200 ℓ per head.

7.2.3 A method of supply and installation

(1) Source of water (Refer to Fig. 7.2.2)

The Mankwala is selected because of its great advantage of a large capacity.

However, the capacity of the Mankwala Dam at the end of the dry season, the Kasangazi River being a dry swamp during dry season, is less than the necessary amount of water for the operation and there is no water supply except precipitation.

Therefore, the process water will be recycled in the plant.

The required amount of process water will coordinate with the maximum capacity of Mankwala Dam using the recycled water.

(2) Intake of water

The water will be taken in by the diver pump at the upper reaches of the pond, and sent to a pump sump near the pond.

(3) Heading of water

The water will be sent through a 4"ϕ steel pipe for a distance of 7,800 m from the pump station to the 1,000 ton tank in the plant.

After purification, the process water will be pumped up to an adjacent 80 ton elevated tank and also the water for domestic use will be pumped up to an 80 ton elevated tank in the resident area which is 650 m from the purifier after being sterilized.

(4) Purification of water

The object of this installation is to remove foreign and fine impurities.

This installation will be furnished with an 1,000 ton concrete tank.

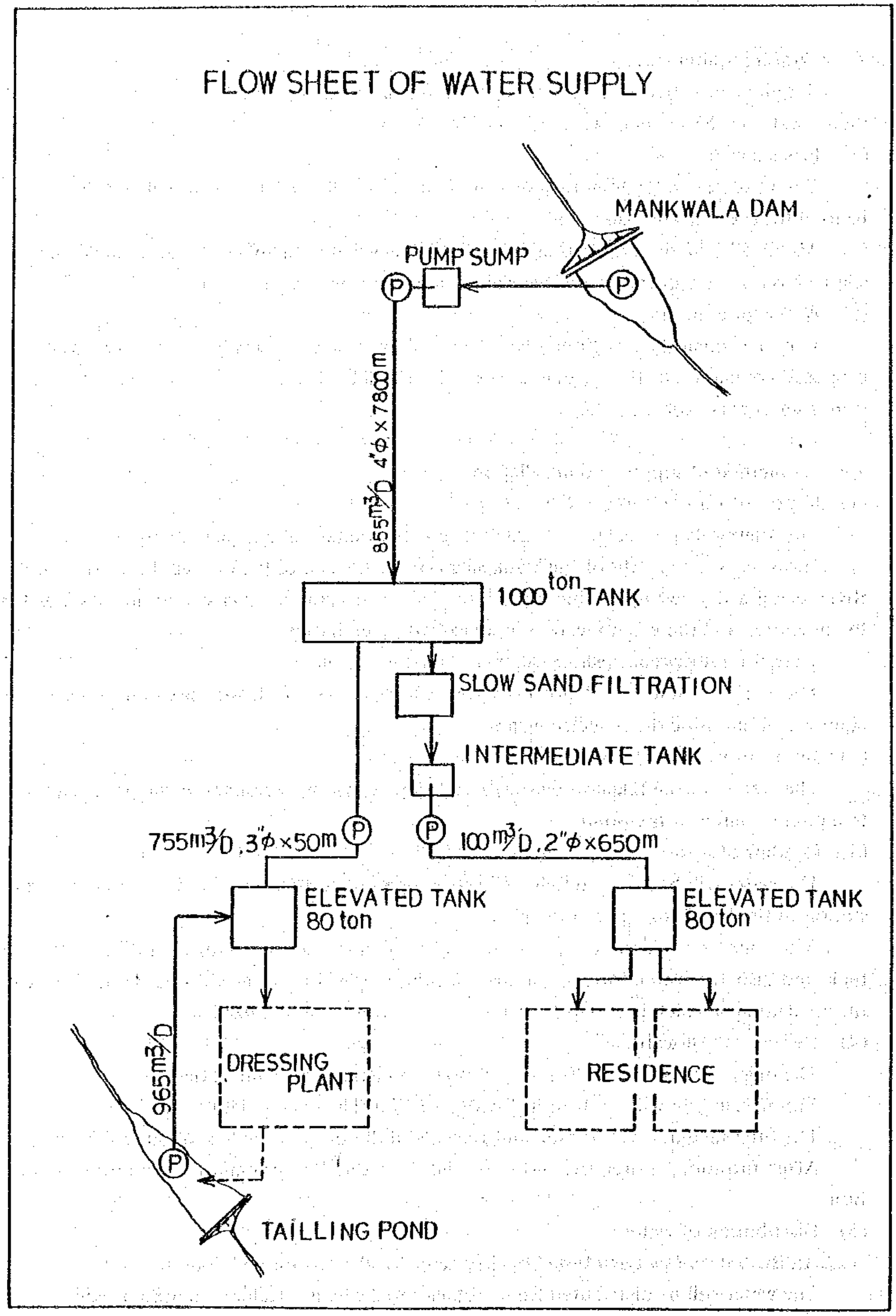
The filter medium will be sand and gravel, and the thickness will be about 1.0 m.

After filtration, moreover, the water will be treated by calcium hypochlorite sterilization.

(5) Distribution of water

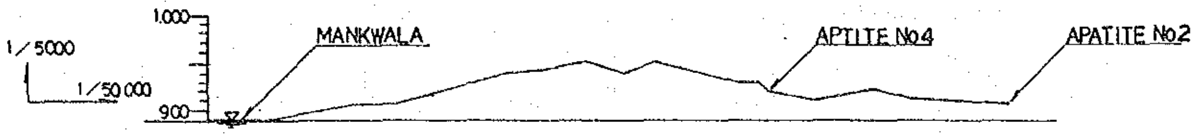
A distribution is accomplished by 2 systems for the process and domestic water.

The water will be distributed from each elevated exclusive tank by natural head.


FLOW SHEET OF WATER SUPPLY
MANKWALA DAM
PUMP SUMP
P
855m3/D 4"φ×7800m
1000ton TANK
SLOW SAND FILTRATION
INTERMEDIATE TANK
755m3/D, 3"φ×50m
100m3/D, 2"φ×650m
ELEVATED TANK 80 ton
ELEVATED TANK 80 ton
965m3/D
DRESSING PLANT
RESIDENCE
TAILLING POND

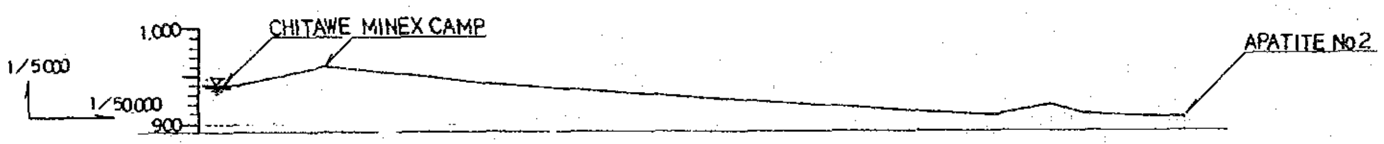
NO.	REVISIONS	DATE	DRAFT	CHK'D	APP'D
1					
2					
3					

MANKWALA — APATITE No 2



No	DISTANCE (m)	CUMULATIVE DISTANCE (m)	ELEVATION (m)
0	000.0	000.0	890.0
1	200.0	200.0	
2	200.0	400.0	
3	600.0	1000.0	905.0
4	400.0	1400.0	
5	400.0	1800.0	915.0
6	400.0	2200.0	928.0
7	400.0	2600.0	938.0
8	300.0	2900.0	941.0
9	500.0	3400.0	952.0
10	400.0	3800.0	940.0
11	300.0	4100.0	952.0
12	900.0	5000.0	928.0
13	200.0	5200.0	928.0
14	100.0	5300.0	920.0
15	500.0	5800.0	910.0
16	600.0	6400.0	921.0
17	400.0	6800.0	910.0
18	1000.0	7800.0	905.0

CHITAWE — APATITE No 2



No	DISTANCE (m)	CUMULATIVE DISTANCE (m)	ELEVATION (m)
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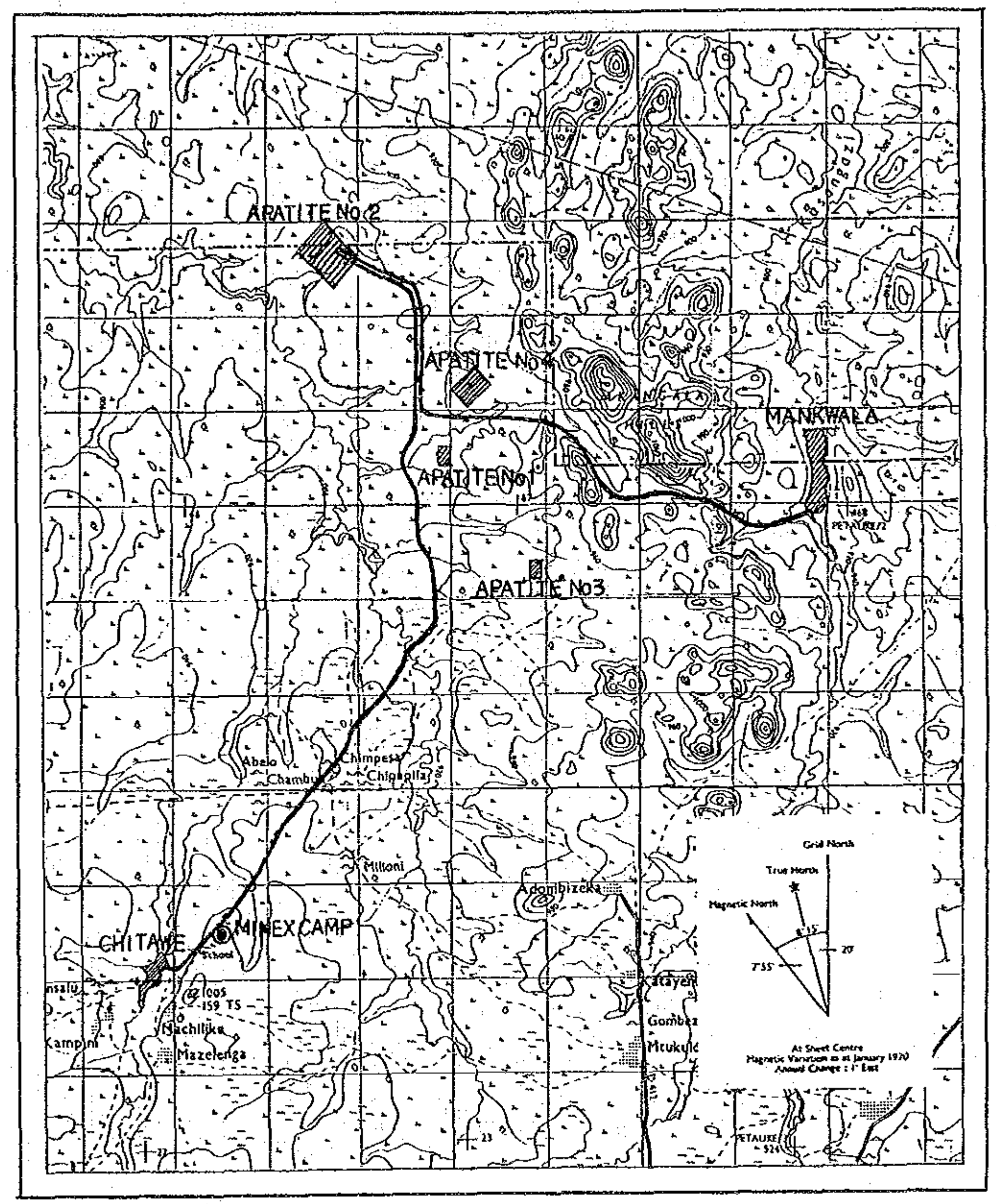
1	900.0	900.0	960.0
	500.0	1400.0	
2	600.0	2000.0	950.0
3	700.0	2700.0	942.0
4	2000.0	4700.0	
5	700.0	5400.0	
6	500.0	5900.0	
7	200.0	6100.0	
8	800.0	6900.0	
9	200.0	7100.0	913.0
10	100.0	7200.0	
11	700.0	7900.0	910.0
12	600.0	8500.0	921.0
13	400.0	8900.0	910.0
14	1000.0	9900.0	905.0

(N.B) TEMPORARY ELEVATION : THE CHITAWE DAMCREST EL = 945.000 ML



HAPPY

Fig. 7.2.2 Route Map of Pipe Line

7.3 Road Construction

7.3.1 Main access road (Refer to Fig. 7.3)

The established road (width 3.2 m, length 35 km) is to run through at about the ridge line from the Great-East Road to the mine site.

The maximum grade is about 1.2%. Therefore, the work will be to widen and improve the established road except for a 2 km section close to the mine site which will have to be of new construction.

7.3.2 The inside roads

The roads will be constructed with a gravel surface and will connect the office, the pit, the concentrator, the magazine, and the tailing-pond.

7.3.3 Approach for the water supply

This approach will be constructed with a gravel surface, and will connect the plant to the Mankwala Dam which will be an important source of water supply. This will facilitate dam patrol during operation.

7.4 Tailing Pond

The purpose of the tailing pond is to allow the solids to settle out of the tailing discharge of the plant, and because of the shortage of dressing water, to recycle the separated water to the plant.

7.4.1 Site selection

As the result of the investigation, the valley of the Luwanda River, 600 m south from the proposed site of the concentrator is selected.

7.4.2 Topography

The site is of a comparatively deep valley forming a distinct basin and a big tributary. Accordingly, it will have an advantageous pocket.

7.4.3 Catchment area

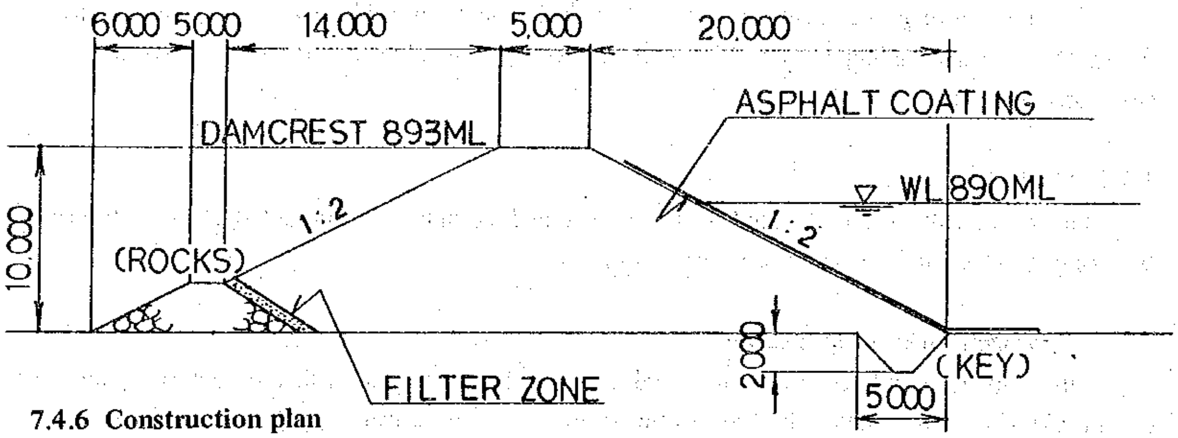
The catchment area of the site is 80.6 km^2, reaching 15 km south and extending 7 km in an east-west direction. In the dry season, it has a dry river bed.

7.4.4 Type of tailing pond

The pond will be formed by closing the opening on the north-west side. The tailing transported by the pipeline from the concentrator to the pond will be discharged on the upper side without any treatment and will be piled upto an elevation of 890 m above sea level.

7.4.5 Embankment

The dam will be an earth filled dam with asphalt coating on the inner slope to store water coming from the upper stream. At the dam toe, a drain which has good permeability will be constructed to lower the seapage line.



7.4.6 Construction plan

(1) Volume balance of tailing

Annual production of crude ore	104,000 T
Tailing ratio	66.2%
Tailing volume	68,848 T/year (specific gravity: 1.0)
Total tailing volume	68,848 T × 15 year = 1,033,000 m^3

(2) Construction plan

To minimize the initial investment the dam construction is separated into three 5 year periods. By the cumulative curve, dam heights are as follows:

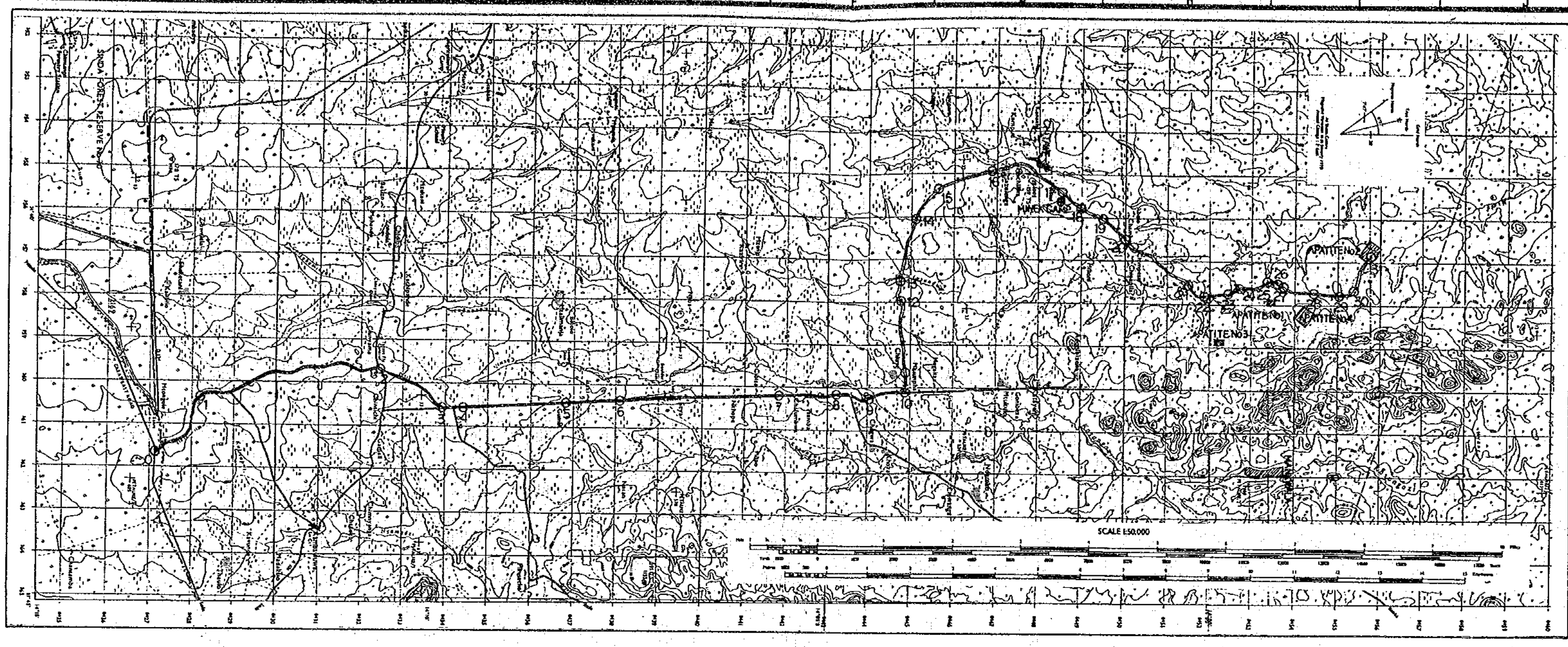
Year	Dam Height (mL)	Dam capacity (m^3)
1	890.5	348,000
7	892.0	688,200
12	893.0	1,033,100

Note: Height of extrabanking between the top of the dam and water level is 3 m.

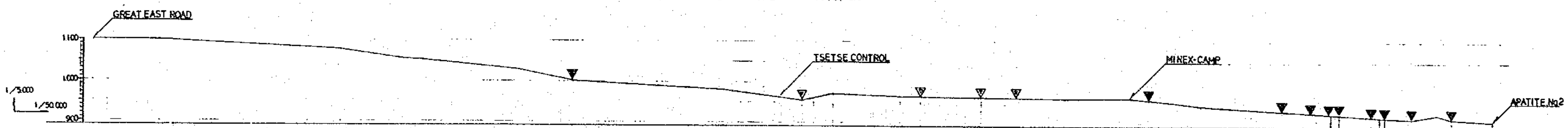
7.4.7 Drainage

The catchment area of 80.6 km^2 is very spacious in comparison with the dam area. As this dam is utilized for an impounding reservoir, storm sewage is gathered and impounded.

For the purpose of water level adjustment, an open channel at the left side of dam (3.5 m wide, 1.8 m high, 150 m long) will be constructed with concrete to discharge down-stream.



GREAT EAST ROAD — APATITE No.2



No	DISTANCE (m)	CUMULATIVE DISTANCE (m)	ELEVATION (m)
0	000.0	000.0	1102.0
1	1600.0	1600.0	1100.0
2	4500.0	6100.0	1078.0
3	1600.0	7700.0	1055.0
4	500.0	8200.0	1052.0
5	2400.0	10600.0	1028.0
6	1300.0	11900.0	1010.0
7	3800.0	15700.0	980.0
8	1400.0	17100.0	962.0
9	600.0	17700.0	953.0
10	300.0	18000.0	960.0
11	500.0	18500.0	972.0
12	1700.0	20200.0	964.0
13	500.0	20700.0	
14	1500.0	22200.0	
15	900.0	23100.0	
16	1200.0	24300.0	958.0
17	1600.0	25900.0	960.0
18	500.0	26400.0	
19	600.0	27000.0	950.0
20	700.0	27700.0	942.0
21	2000.0	29700.0	
22	700.0	30400.0	
23	500.0	30900.0	
24	200.0	31100.0	
25	800.0	31900.0	
26	200.0	32100.0	913.0
27	100.0	32200.0	
28	700.0	32900.0	910.0
29	600.0	33500.0	
30	400.0	33900.0	910.0
31	1000.0	34900.0	905.0

(NB-1) TEMPORARY ELEVATION : THE CHITAWE DAM CREST EL = 945.000 ML

(NB-2) BRIDGE CONSTRUCTION : ▼
BRIDGE REPARATION ... : ▽

Fig. 7.3 Road Profile

7.5 Auxiliary Facilities

These facilities are composed of repair shop, office, warehouse, and other facilities.

Item	Nos.	Dimension	m^2	Specification
Repair shop	1	5 X 30	150	Mechanic, electric
Pit shop	1	9 X 24	216	Tire parts, office
Magazine	2	4 X 5	20	Dynamite 43 t for 6 months
		3 X 4	12	
Ware house	1	5 X 30	150	Spare parts, general goods
Office	1	10 X 20	200	Office, technical staff, capacity 120
Canteen	1	10 X 25	250	Capacity 120 persons
Change house	1	3 X 6	18	Personnel locker 45
Fuel station	1	3 X 4	12	For dump truck, machine
Security office	2	3 X 4	12	South gate, east gate

7.6 Welfare Facilities

7.6.1 Outline

An entirely new mine town will be built to accommodate all the mine personnel and their families and to provide necessary services for their daily needs.

The estimated population of the mine town belonging to the company is 500 including the families.

7.6.2 Estimate of resident personnel

In this estimate the percentage of unmarried personnel among the workers is to be 50%. Each family consists of 5 members; a wife and 4 children.

7.6.3 Location and development

The mine town will be built 300 m east from the gate of mine. The staff area and worker's area will be separated. Development area is 6.6 hectares. In order to leave as many trees as possible only 70% of the area will be cleared.

7.6.4 Residence

Housing to be built will be 5 single units (116 m^2/unit) for the mine manager and superintendents, and 22 single units (35.25 m^2/unit) for the other staff.

Housing for the married personnel among the workers will be 12 quadruple units (4 × 26.25 m^2/unit).

Housing for bachelor workers will be 6 octuple units.

Total	45 units
Const:	Concrete block

7.6.5 Service facilities

The service facilities will be built as follows:

Const: Concrete block for all

Facilities	Unit	Area (m^2)	Specification
Church	1	96	Capacity 120 persons
School	3	96	Capacity 100 students
Clinic	1	96	
Guest house	1	100	Reception and accommodation
Ware house & store	1	300	Supply of maize and daily needs
Meet	1	200	Meeting, amusement, bachelor's dining room

7.6.6 Domestic water supply and sewage disposal

Domestic water will be filtered, sterilized, and supplied to each place by steel pipes from an elevated tank.

The sewage that drains from each family unit will be discharged in a channel at the edge of the road which also serves for rain drainage.

The sewage water will be sent by a polyethylene pipe to the purification tank which will be built at the south of the residence area, and discharged into the Luwanda River.

7.7 Maintenance and Repair Section

This section has the pit shop, the machine repair shop, the electrical repair shop and the carpentry workshop which have all the necessary facilities to undertake mechanical and electrical repairs, including major overhauls.

7.8 Administration

This section consists of the purchasing, personnel, accounting, general affairs, training and security control.

8. PRODUCTION PLAN AND MANPOWER REQUIREMENT

8.1 Production Plan

The concentrator will treat 104,000 T of crude ore to produce P_2O_5 concentrate.

Annual average grade of ore treated will be 11.5%.

Therefore, the concentrate will average 35,181 T/year (30.07% P_2O_5) and the P_2O_5 recovered will average 10,578 T/year.

8.1.1 Production in year 3

After the construction work, start up operation and test-run for 3 months will be completed according to the project schedule, and commercial production will start.

A 6 month supply of ore will be produced during the pre-stripping period and the test-run operation at concentrator.

The new equipment can be expected to operate smoothly and normally from the beginning.

Therefore, it will operate at a full capacity of 104,000 T/year from the beginning.

8.1.2 Operation

Annual working days will be 260 days for mining, 290 days for other departments.

Mining will be operated in one shift, milling in three shifts.

Item	Mining	Milling	
Annual crude ore	104,000 T (11.5%)	104,000 T (11.5%)	
Daily crude ore	400 T	360 T	
Annual concentrates		35,181 T (30.07%)	
Annual waste or tailing	218,400 T	68,819 T	
Annual working days	260	290	
Shift per day	1	Primary crushing	1
		Secondary, tertiary crushing	2
		Grinding, flotation	3
Working hours per shift	8	8	

8.2 Manpower Requirement

The proposed organization and manpower distribution are shown in Table 8.1 and Fig. 8.1.

All production departments will be directly under the administration of the company management.

Table 8. Manpower

	Mining		Milling		Maintenance & repair		Administration		Total
	Item	Nos.	Item	Nos.	Item	Nos.	Item	Nos.	Nos.
Staff	Mining engineer	1	Superintendent	1	Superintendent	1	Mine manager	1	
	Mining foreman	1	Operating foreman	3	Pit shop foreman	1	Superintendent	1	
	Surveyor	1	Assayer	1	Plant maintenance foreman	1	Purchase section chief	2	
	Geologist	1	Metallurgist	1			Personnel 〃	1	
	Clerk	1	Clerk	1	Electrical foreman	1	Account 〃	1	
							General affair 〃	1	
							Training 〃	1	
							Security 〃	3	
Staff Total		5		7		4		11	27
Worker	Driller	4	Operator	16	Pit shop mechanics	2	Purchase section	6	
	Blasting crew	3	Assayer	2	Plant maintenance	2	Personnel 〃	4	
	Shovel operator	4	Metallurgical test	1	Electrician	3	Account 〃	2	
	Truck operator	5	Loader operator	1	Operator	2	General affair 〃	9	
	Bulldozer operator	2	Pump control	2	Civil work	2	Training 〃	2	
	Driver	2					Security 〃	9	
	Helper of surveyor	3							
	Helper of geologist	2							
Worker Total		25		22		11		32	90
Grand Total		30		29		15		43	117

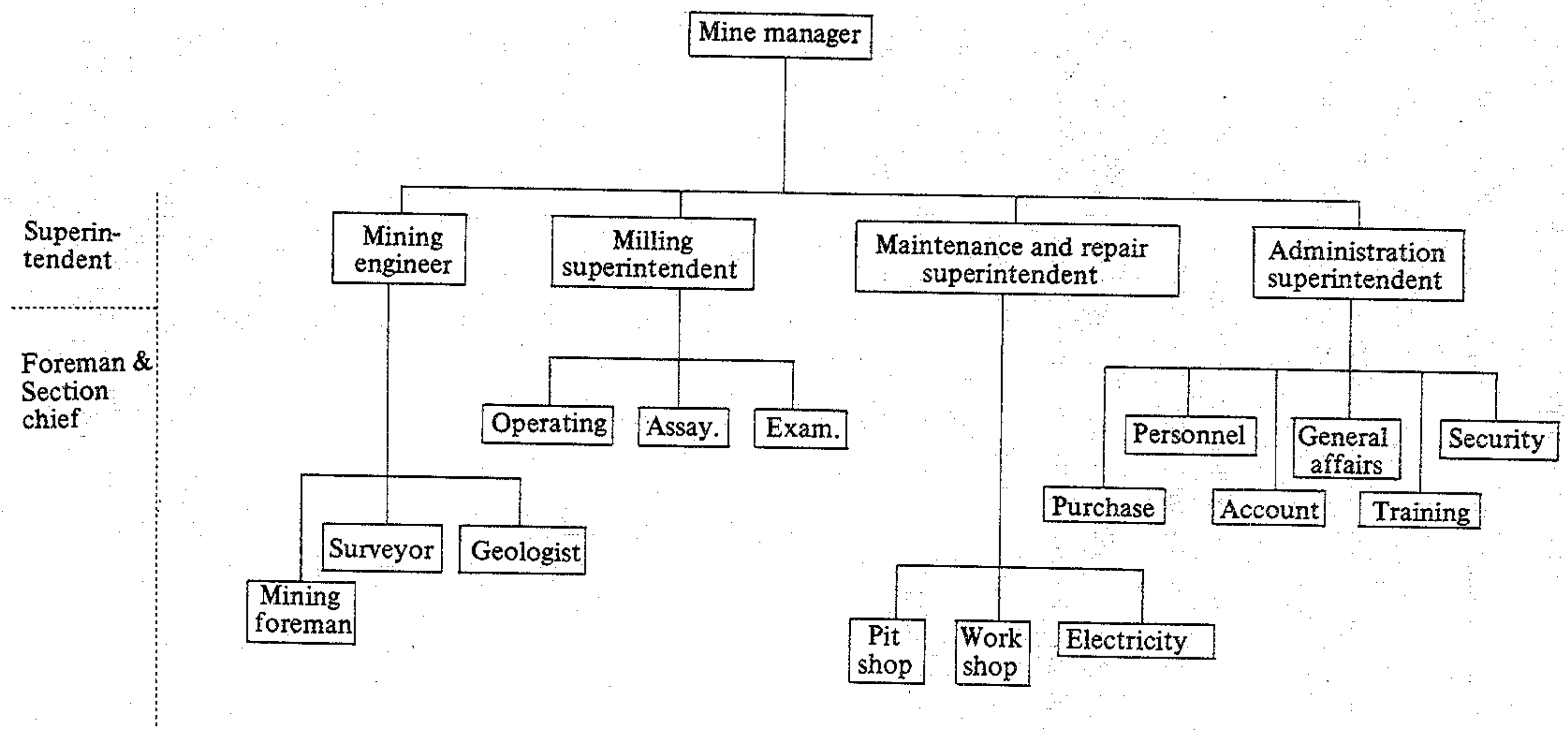


Fig. 8 Organization Chart

Classification of staff and workers by department is as follows.

Classification by department	Staff	Worker	Total
Mining	5	25	30
Concentrator	7	22	29
Maintenance and repair	4	11	15
Administration	11	32	43
Total	27	90	117

Note: Including the mine manager in Administration department.

The number on the table above shows only operating staff on the mine site, and does not include managing staff in Lusaka, because they should be considered in connection with the fertilizer plant.

9. PROJECT SCHEDULE

The project schedule allows three years to bring the Mine into production, which is shown in Table 9.

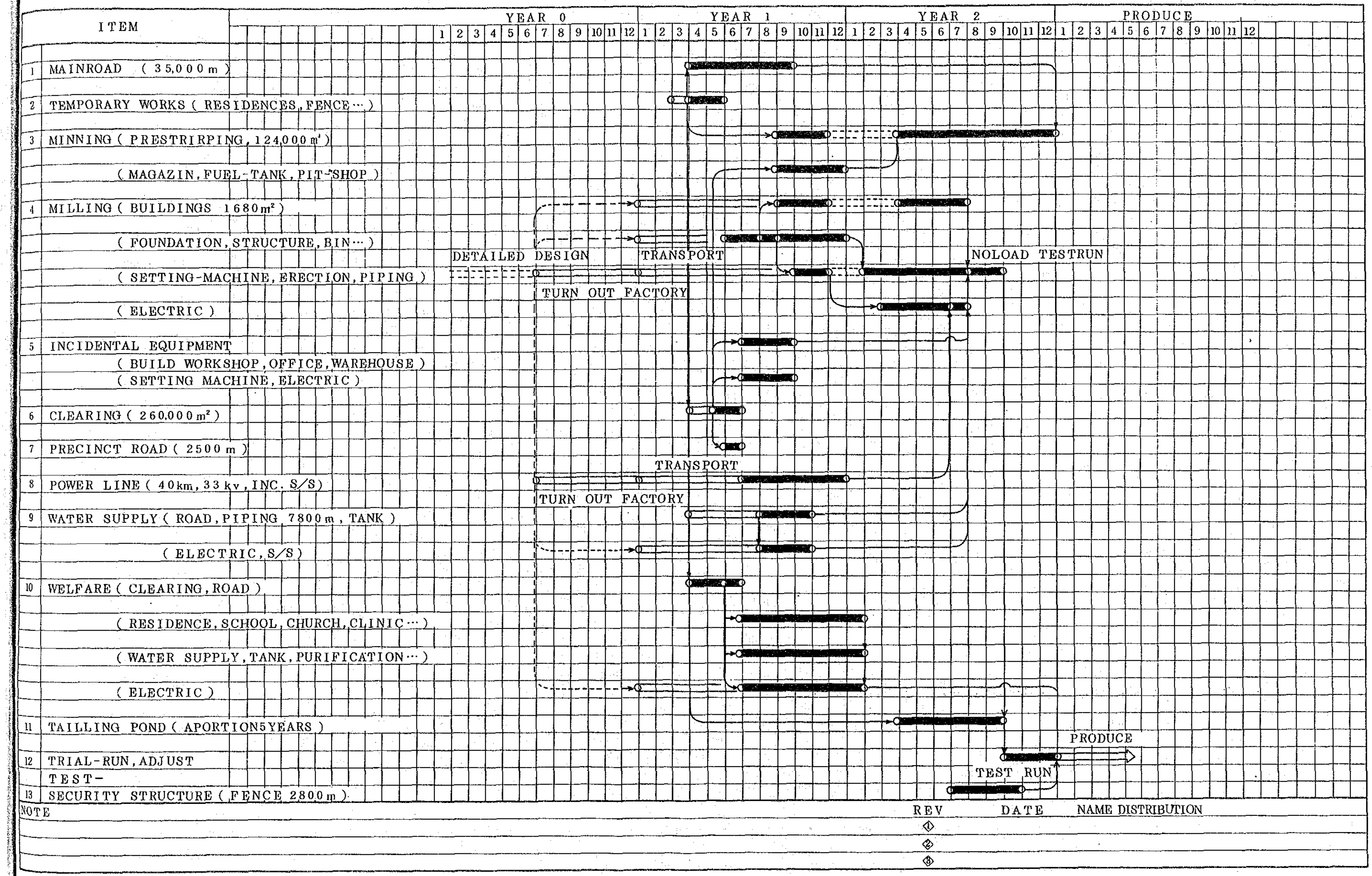
The work in year (0) is only preparatory work such as engineering, ordering equipment and machinery, etc. The main construction work will commence in year (1) and will last for two years. Civil work will stop during rainy seasons. Key dates of the schedule are:

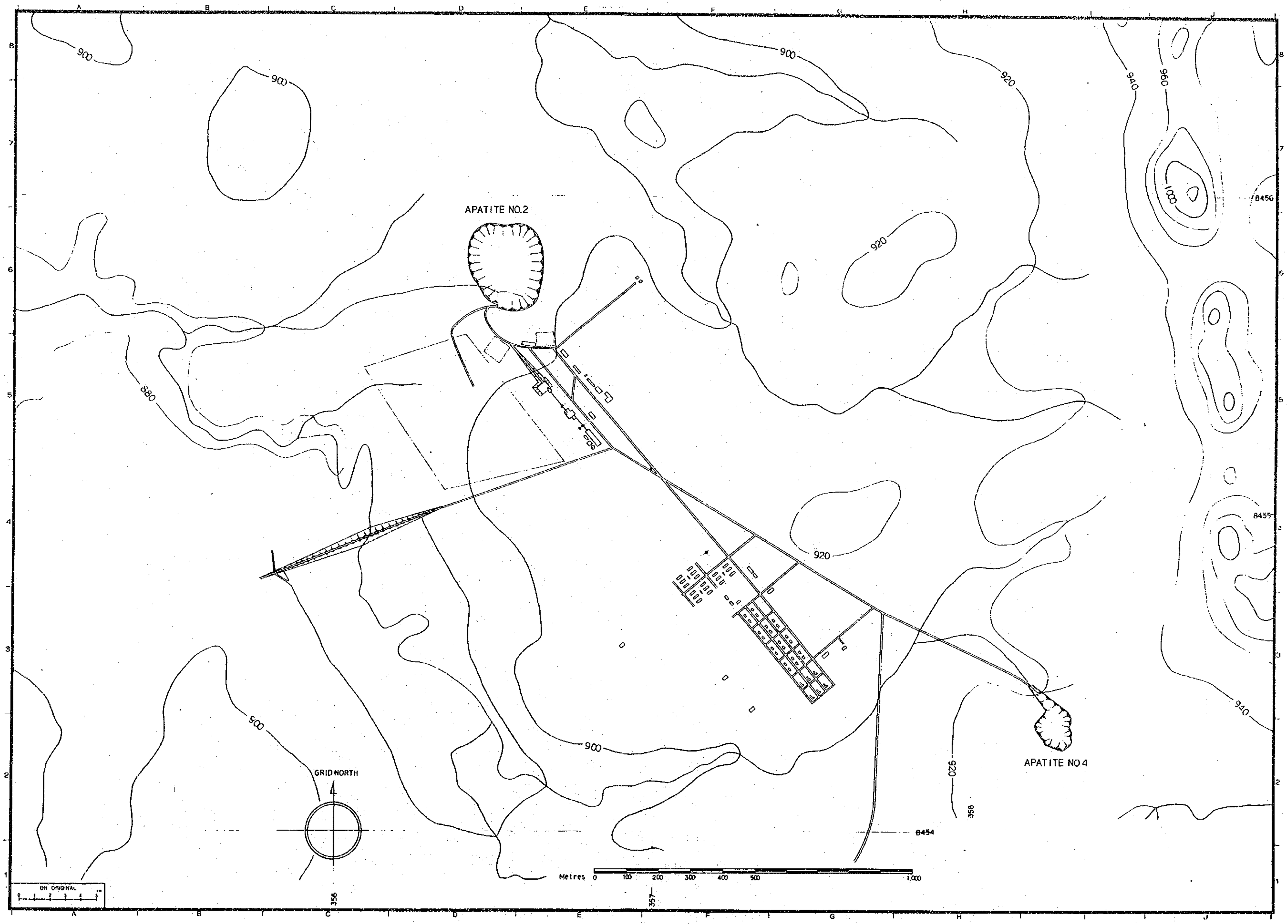
Year 1	April	Civil work starts
Year 1	September	Pre-stripping starts
Year 2	August	No-load test run starts
Year 2	October	Test-run with load starts
Year 3	January	Production commences

Test-run without load will commence from August of year (2), while a test with load from October will end by December, treating 26,000 T.

Production will commence in January, so that Mankwala Dam and the tailing pond will be full of water, to avoid initial trouble.

Table 9.1 CONSTRCTION SCHEDULE





General Layout

10. CAPITAL COST AND OPERATING COST

10.1 Capital Cost Estimates

10.1.1 General

The following items are the capital cost estimates at the start of production.

		Amount (1,000 US$)
(1)	Production and auxiliary facilities (Mining, concentrator, tailing pond, water supply, power distribution, main road, etc.)	9,180.0
(2)	Welfare facilities	722.8
(3)	Common construction (Temporary facilities, etc.)	51.1
(4)	Management cost and engineering fee	858.7
(5)	Inventories (General stocks and spare parts)	780.6
(6)	Working capital	86.2
(7)	Contingency	1,119.8
	Total	12,799.2

Working capital is based on a 3 month operation cost of labour and electricity.
Inventories are estimated;

Imported materials — 6 month supply
Local materials — 1 month supply

10.1.2. Basis for capital cost estimates

Applicable laws, working condition, salary, wage, equipment purchase cost, commodities prices, etc. are those on September-November, 1984.

Currency conversion rate: The rates used are;

US$ = K 1.8
US$ = ¥245 (Sept. 1984)

All amounts in the estimate are expressed in US dollar.

Construction work: Pre-stripping work except clearing of the top soil is to be done by the company. All the rest of the construction work is to be done by constractors.

The average number of construction workers is estimated to be 150 per day.

Imported equipment: A transportation charge in marine and inland with an insurance is estimated at 30% of an equipment cost.

Prices of principal construction materials at the mine:

Diesel fuel	US\$ 0.56/ℓ
Gasoline	US\$ 0.75/ℓ
Cement	US\$ 72/T
Dynamite	US\$ 500/T
AN-FO	US\$ 460/T
Wooden	US\$ 417/m^3

10.2 Additional Investment and Replacement Cost

Additional investment and replacement cost after the start of production are;

(1) Drainage pump set at the bottom of open pit

(2) Additional work on tailing pond

(3) Replacement cost including mining and other equipments

Table 10.2 shows the estimated additional investment and replacement cost by year up to year 17 of production stage.

10.3 Operating Cost Estimates

The operating cost consists of the direct operating cost of each department; mining, milling and maintenance.

10.3.1 Average annual operating cost of each department

	Year 3 ~ 15	Year 16	Year 17
Ore treated/year	104,000 T	104,000 T	69,000 T
Operating cost (\$)	1,148,100	1,120,460	638,300
Mining	390,420	362,780	135,610
Concentrator	452,040	452,040	299,910
Maintenance	96,600	96,600	64,090
Administration	158,140	158,140	104,920
Electricity	50,900	50,900	33,770
(US\$/T ore)	(11.04)	(10.77)	(9.25)
By foreign currency	44.84%	44.84%	46.50%
By domestic currency	55.16%	55.16%	53.50%

The material costs of maintenance and repair for equipment and vehicle are included in each department.

Table 10.1. Breakdown of Capital Cost

(Units: 1,000 US$)

Item	Total			Year 1			Year 2			Remarks
	Total	K	$	Total	K	$	Total	K	$	
MINING	2314.8	290.5	2024.3	1637.1	58.3	1578.8	677.7	232.2	445.5	
Pit equipment	*1914.7		1914.7	1577.9		1577.9	336.8		336.8	Preproduction stripping 350,670 t
Preproduction stripping	383.2	274.5	108.7	42.3	42.3		340.9	232.2	108.7	
Magazine	16.9	16.0	0.9	16.9	16.0	0.9				
MILLING	4029.7	1105.5	2924.2	3612.5	817.2	2795.3	417.2	288.3	128.9	
Loader	* 128.9		128.9				128.9		128.9	
Equipment	2181.3		2181.3	2181.3		2181.3				
Installation	990.5	785.2	205.3	920.9	715.6	205.3	69.6	69.6		
Building construction	422.4	312.8	109.6	211.2	101.6	109.6	211.2	211.2		
Electric work	306.6	7.5	299.1	299.1		299.1	7.5	7.5		
WATER SUPPLY	465.6	219.9	245.7	465.6	219.9	245.7				4" pipe line 7.8 km
Equipment	15.5		15.5	15.5		15.5				
Pipe line & building	250.2	127.9	122.3	250.2	127.9	122.3				
Road	70.2	70.2		70.2	70.2					
Electric work	129.7	21.8	107.9	129.7	21.8	107.9				
MAIN ROAD	503.9	502.1	1.8	503.9	502.1	1.8				
G.E.R-Minesite	471.9	471.9		471.9	471.9					Great East Road-minesite 35km
Bridge	32.0	30.2	1.8	32.0	30.2	1.8				No. of bridge 14
POWER LINE & DISTRIBUTION	831.5	389.3	442.2	767.1	383.5	383.6	64.4	5.8	58.6	
Power line (Sinda-Minesite)	361.1	361.1		361.1	361.1					Power line 40 km 33 kV
Sub-station	364.1	13.1	351.0	364.1	13.1	351.0				Capacity 900 kVA
Auxiliary facilities	83.3	14.1	69.2	41.9	9.3	32.6	41.4	4.8	36.6	
Communication	23.0	1.0	22.0				23.0	1.0	22.0	
TAILING POND	180.3	175.7	4.6				180.3	175.7	4.6	1st stage (life 5 years)
MAINTENANCE & REPAIR	652.1	187.9	464.2	536.4	187.9	348.5	115.7		115.7	
Maintenance equipment	224.6		224.6	224.6		224.6				
Building construction	72.1	64.9	7.2	72.1	64.9	7.2				
Vehicle	* 232.4		232.4	116.7		116.7	115.7		115.7	
Civil work	123.0	123.0		123.0	123.0					Including clearing & road construction in minesite
SUB TOTAL	8977.9	2870.9	6107.0	7522.6	2168.9	5353.7	1455.3	702.0	753.3	

(Units: 1,000 US$)

Item	Total			Year 1			Year 2			Remarks
	Total	K	$	Total	K	$	Total	K	$	
AUXILIARY FACILITIES	109.7	70.6	39.1	96.9	70.6	26.3	12.8		12.8	
Vehicle	* 36.1		36.1	23.3		23.3	12.8		12.8	
Building construction	53.2	50.2	3.0	53.2	50.2	3.0				
Furnitures & fixtures	20.4	20.4		20.4	20.4					
MINE WELFARE FACILITIES	722.8	660.1	62.7	616.5	553.8	62.7	106.3	106.3		Population 500
Site preparation	9.2	9.2		9.2	9.2					
House & quarters	515.0	485.9	29.1	432.6	403.5	29.1	82.4	82.4		
Road	52.5	52.5		52.5	52.5					
Water supply & sewage	129.5	108.9	20.6	105.6	85.0	20.6	23.9	23.9		
Power distribution	16.6	3.6	13.0	16.6	3.6	16.6				
SECURITY FACILITIES	92.4	28.0	64.4				92.4	28.0	64.4	Fencing around the mine site
TEMPORARY FACILITIES	51.1	19.4	31.7	51.1	19.4	31.7				
Portable generator	26.5		26.5	26.5		26.5				
Temporary housing	5.6	5.6		5.6	5.6					
Warehouse stockyard	12.4	11.8	0.6	12.4	11.8	0.6				
Fence	6.6	2.0	4.6	6.6	2.0	4.6				
CONSTRUCTION MANAGEMENT	152.7	152.7		60.8	60.8		91.9	91.9		
EDUCATION AND TRAINING	111.1	111.1					111.1	111.1		
INVENTORY	* 780.6	24.0	756.6	327.3		327.3	453.3	24.0	429.3	Import material 6 months Local material 1 month
TEST-RUN	113.7	42.2	71.5				113.7	42.2	71.5	
WORKING CAPITAL	* 86.2	86.2					86.2	86.2		
SUB TOTAL P.1 + P.2	11198.2	4065.2	7133.0	8675.2	2873.5	5801.7	2523.0	1191.7	1331.3	
CONTINGENCY (p.1 + p.2) *10%	1119.8	406.5	713.3	867.5	287.4	580.2	252.3	119.2	133.1	
ENGINEERING FEE	481.2		481.2	397.8		397.8	83.4		83.4	6% of capital cost without * mark
TOTAL	12799.2	4471.7	8327.5	9940.5	3160.9	6779.7	2858.7	1310.9	1547.8	
INTEREST DURING CONST.										

Amount Subject to Depreciation

Unit: 1,000$

Half Term	1st	2nd	3rd	4th
Mining	0.9	1,636.2	108.7	569.0
Concentrator	2,347.6	1,264.9	128.9	288.3
Water supply	245.7	219.9		
Road	1.8	502.1		
Power supply	383.6	383.5	58.6	5.8
Tailing pond			4.6	175.7
Maintenance & repair	123.9	412.5		115.7
Auxiliary facilities	3.0	93.9	12.8	
Welfare facilities	62.7	553.8		106.3
Security facilities			64.4	28.0
Temporary facilities	31.7	19.4		
Construction management		60.8		91.9
Education & training				111.1
Sub-total	3,200.9	5,147.0	378.0	1,491.8
Contingency	320.1	514.7	37.8	149.2
Engineering fee	397.8		83.4	
Total	3,918.8	5,661.7	499.2	1,641.0

Table 10-2. Additional Investment and Replacement Cost

(US$1,000)

Year	Additional Investment			Replacement Cost			Grand total
	Tailing pond	Mining equipment	Total	Heavy equipment	Vehicles	Total	
3	–	13.5	13.5	–	–	–	13.5
4	–	–	–	–	–	–	–
5	–	–	–	–	14.6	14.6	14.6
6	–	–	–	–	–	–	–
7	113.8	–	113.8	190.8	41.6	232.4	346.2
8	–	–	–	162.8	14.6	177.4	177.4
9	–	–	–	171.6	–	171.6	171.6
10	–	–	–	347.4	–	347.4	347.4
11	–	–	–	162.8	14.6	177.4	177.4
12	70.7	–	70.7	663.0	32.6	695.8	766.5
13	–	–	–	–	–	–	–
14	–	–	–	–	14.6	14.6	14.6
15	–	–	–	–	–	–	–
16	–	–	–	–	–	–	–
17	–	–	–	–	–	–	–
Total	184.5	13.5	198.0	1,698.4	132.8	1,831.2	2,029.2

Table 10-3. Replacement (Heavy Equipment and Vehicle)

Year	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	Total
Crawler drill										1						1
Dozer shovel										1						1
20 T dump truck					1	1	1	1	1	1						6
Bulldozer								1								1
Wheel loader										1						1
Water cart					1					1						2
Pick-up					1					1						2
2 T truck					1		1			1						3
Light van					1					1						2
Ambulance			1			1			1			1				4
Vehicle for manager					1					1						2

Table 10-4. Operating Cost per Year

(US$)

Year	Mining	Concentrator	Maintenance	Administration	Electricity	Total	Foreign currency	Domestic currency
3	390,420	452,040	96,600	158,140	50,900	1,148,100	514,800	633,300
4	"	"	"	"	"	"	"	"
5	"	"	"	"	"	"	"	"
6	"	"	"	"	"	"	"	"
7	"	"	"	"	"	"	"	"
8	"	"	"	"	"	"	"	"
9	"	"	"	"	"	"	"	"
10	"	"	"	"	"	"	"	"
11	"	"	"	"	"	"	"	"
12	"	"	"	"	"	"	"	"
13	"	"	"	"	"	"	"	"
14	"	"	"	"	"	"	"	"
15	"	"	"	"	"	"	"	"
16	362,780	"	"	"	"	1,120,460	502,420	618,040
17	135,610	299,910	64,090	104,920	33,770	638,300	296,810	341,490
Total	5,573,850	6,628,470	1,410,490	2,318,880	746,370	16,684,060	7,491,630	9,192,430

10.3.2 Basis for operating cost estimates

The cost is estimated taking into account the variations in mine operation and the price level on September – November, 1984.

Currency conversion rate:

US$ 1.0 = K 1.8
US$ 1.0 = ¥245

All amount in the estimate are expressed in US dollar.

Salary and Wage

Salary and wage including the basic pays, social security, bonus, retirement allowance, etc. are as follows:

Staff:	
Mine manager	7,200 US$/year
Mining engineer and superintendent	5,400
Foreman	3,960
Surveyor and geologist	4,320
Worker:	
Operator, mechanic and electrician	2,400 US$/year
Technical worker	2,100
Non-technical worker	1,700
Helper	1,500

Purchased Power Cost:

Computed in accordance with the Section 12 of Electricity Act, Chapter 811.
From Tariff D2. 0.015 US$/kWh

Commodity Price

Fuel oil (diesel oil)	US$ 0.56/ℓ
Fuel oil (gasoline)	US$ 0.75/ℓ
Dynamite	US$ 500/T
AN-F0	US$ 460/T
4 inch cross bit	US$ 200/pc
Tire for 20 T truck	US$ 1,172/pc
Ball for ball mill	US$ 750/T
Caustic soda	US$ 0.33/kg
Sodium silicate (water glass)	US$ 0.22/kg
Lila flot	US$ 2.33/kg

SUPPLEMENT

SUPPLEMENT

1. FINANCIAL EVALUATION

1.1 Evaluation Method

For the convenience of comparison of the project with projects in other fields, a single rate, which is known as the internal rate of return, is used in the present evaluation.

In general, the price of non-metallic mineral is low and especially in inland areas, the delivery costs often exceed production costs.

The mining project should, as a prerequisite, demand that a phosphatic manure plant be constructed. But selection of a site for a fertilizer plant is beyond the scope of the present study. Therefore, a delivery cost of apatite concentrates is excluded from the calculation. The evaluation of the mining project is based on the mine-site realization derived from a sale of concentrates.

1.2 Assumed Parameters

(1) Capital

Initial expenses have been estimated to be in the sum of $12.8 million, exluding interest yielded during a construction period. The amount of the capital is equivalent to $3.5 million, some 27% of the total funds employed.

(2) Loan

The balance of the funds is financed by the borrowing of a long-term loan from a bank. The debt is deferred at compound interest to the end of the construction period and returned uniformly for the next fifteen years. The rate of interest is set at 4% per annum on the exchange value of the U.S. dollar.

(3) Depreciation

After the completion of the construction period, an accelerated depreciation method is applied as stipulated in the Income Tax Act.

(4) Taxation

A tax is exempted during the construction period. A rate of 45% of tax is imposed on a taxable income.

(5) Production and Sale

During the construction period, some 26,000 tonnes of ore are fed in to the dressing plant for a test run, with an expected effectiveness of 75% of the normal recovery rate. From the third year, a sum of 104,000 tonnes of ore is treated annually to produce 35,000 tonnes of apatite concentrates for fourteen years. On the 17th year, 69,000 tonnes of ore are treated and 23,000 tonnes of concentrates are recovered.

(6) Price

A price of apatite concentrates at the mine-site is calculated with the basis of cost-and-fee.

The total cost consists of a capital cost and an operation cost. The investment in fixed assets, working capital and commodity inventories, is provided by the capital fund and the bank loan. The present value of the initial investment at the end of the second year is calculated at 4% of the interest rate. The amounts of additional investment scheduled in the future are similarly converted to the present value at the end of the second year and added to the present value of the initial investment. The total funds utilized add up to $14,920,000 at the end of the second year. The amount is then multiplied by the factor of amortization at 4% for 15 years (0.08994).

The amount needed in the amortization of the capital funds is $1,342,000 per annum. Therefore, a cost at the mine-site would be;

	in US$1,000
Capital cost	1,342
Operation cost	1,148
Total	$2,490

During the first six months in 1983, the parastatal mining sector gave an average rate of profit at 6.47% against the turnover before tax.

Taking this into account, the 8% is applied as the rate of fee against the total cost. If the rate is raised to a high level, the resulting price makes the products to be insufficiently competitive with foreign materials.

In accordance with this assumption, a price of concentrates is calculated as follows:

Cost	($)2,490,000 ÷ 35,000 (tonnes)	=	$71.14
Fee	($) 71.4 × 0.08	=	$5.69
Total			$76.83

Consequently, the mine-site realization is set at $77 per tonne of apatite concentrates.

(7) Exclusion

No provisions have been provided for exploration, lease purchase or mineral tax.

Escalation clauses are not introduced with the assumption that as an operation cost accounts for 46% of the total cost, or 47% if the additional investments are added, an increase of cost may be absorbed by an increase of the selling price.

Salvage values are deemed to be nil.

1.3 Internal Rate of Return

In accordance with these premises, a statement of profit and loss account and the calculation of the internal rate of return is shown in Table 1-3. The results are as follows.

The internal rate of return before tax 7.1%

The internal rate of return after tax 5.9%

Table 1-3. Profit and Loss Account and the Internal Rate of Return

in US$1,000

	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	T
Tonnes, milled (x 10^3)		26	104	104	104	104	104	104	104	104	104	104	104	104	104	104	69	1,551
Tonnes of Conc. (x 10^3)		6.56	35	35	35	35	35	35	35	35	35	35	35	35	35	35	23	519.56
Capital Funds & Bank Loan	9,941	2,354																12,295
Sale of Products at $77/t		505	2,695	2,695	2,695	2,695	2,695	2,695	2,695	2,695	2,695	2,695	2,695	2,695	2,695	2,695	1,771	40,006
Total Available	9,941	2,859	2,695	2,695	2,695	2,695	2,695	2,695	2,695	2,695	2,695	2,695	2,695	2,695	2,695	2,695	1,771	52,301
Operation Expenses		125	1,148	1,148	1,148	1,148	1,148	1,148	1,148	1,148	1,148	1,148	1,148	1,148	1,148	1,120	638	16,807
Interest of Financing			370	346	321	296	272	247	222	197	173	148	123	99	74	49	24	2,961
Depreciation			1,177	1,201	1,226	1,251	1,275	1,300	1,325	1,350	1,374	1,399	858	–	15	–	–	13,751
Taxable Income													566	1,448	1,458	1,526	1,109	6,107
Tax at 45% Net													255	652	656	687	499	2,749
Net Profit													311	796	802	839	610	3,358
Taxable Income + Depreciation			1,177	1,201	1,226	1,251	1,275	1,300	1,325	1,350	1,374	1,399	1,424	1,448	1,473	1,526	1,109	19,858
Investment to be written-off	9,581	2,140	14		15		346	177	172	347	177	767		15				13,751
Working Capital & Inventories	360	594															–954	0
Total Investment	9,941	2,743	14		15		346	177	172	347	177	767		15			–954	13,751
Tax at a Payable Year														255	652	656	1,186	2,749
Total Required	9,941	2,859	1,532	1,494	1,484	1,444	1,766	1,572	1,542	1,692	1,498	2,063	1,271	1,517	1,874	1,825	894	36,268
Net Inflow	–9,941	–2,354	1,163	1,201	1,211	1,251	929	1,123	1,153	1,003	1,197	632	1,424	1,178	821	870	877	3,738
Add Interest on Financing			1,533	1,547	1,532	1,547	1,201	1,370	1,375	1,200	1,370	780	1,547	1,277	895	919	901	18,994
Discount Rate at 5.864%																		
Present Value	–9,390	–2,100	1,293	1,233	1,153	1,100	806	869	824	679	732	394	738	576	381	370	342	0
Capital Funds	3,500																	
Bank Loan, 1st half term	779																	
2nd half term	5,662																	
3rd half term		578																
4th half term		1,776																
Total Carried			9,262	8,644	8,026	7,408	6,790	6,172	5,554	4,936	4,318	3,700	3,082	2,464	1,846	1,228	610	0
Retirement			618	618	618	618	618	618	618	618	618	618	618	618	618	618	610	9,262

If the owned capital stands at $3.5 million, the implementation of the project will yield an interest at 9.3% on the capital.

1.4 Sensibility of Internal Rate

When the unit price of the concentrates is changed, the internal rate of return varies as follows.

$65	2.6%
$70	4.0%
$77	5.9%
$85	7.9%
$100	11.7%

Among the initial investment, if the cost of fixed assets together with the price of commodity inventories are changed either up or down to 20%, the financial internal rate varies as follows.

+20%	4.0%
Model	5.9%
–20%	8.5%

If the fuel cost increases by 20%, the internal rate becomes 5.5% and if the fuel cost decreases by 20%, the rate will stand at 6.2%.

1.5 Discussion

In regard to the Chilembwe project, the operation cost remains within the range of an acceptable standard in open pit mining. Due to the scale of ore deposits and geographical condition of location, the cost of construction, or depreciation cost per tonne of ore, is somewhat higher. Yet, the annual cost for amortization of the capital funds remains within the ratio of 117% against the operation cost. Enlargement of the rate of production is confined by the volumes of both ore reserves and available water.

If the mine-site realization is assumed to be $77 per tonne of the apatite concentrates as a cost with a fee at the rate of 8%, the financial internal rate of return stands at 5.9%. The rate depends mainly on the value of the products at the mine-site. An effect on the internal rate by fluctuation in fuel cost is rather moderate.

In this evaluation, the delivery cost of the products has been excluded from the calculation. Depending on the locality of the fertilizer plant, the products are requested to be competitive in price with foreign materials.

But import of apatite from abroad conflicts with the national policy to save foreign currencies and much effort should be made to compromise on the terms of pricing the domestic apatite concentrates, enabling both the mine and the fertilizer plant to come into existence.

In case that an additional ore deposit is found nearby, such a deposit can be exploited without raising funds for a capital investment, although heavy duty vehicles will be renewed.