Chapter 5 Drilling Survey

5-1 Outline of the Drilling Survey

For the purpose of confirming ore stratigraphy and occurrence, and of providing samples for the ore dressing tests, two holes, MJK-1 and MJK-2 were drilled (Fig.2-5-1, Table 2-5-1). These were drilled by the wire-line drilling method, and have a diameter of 59mm. Laboratory tests were carried out on core samples as shown in Table 1-1-2.

MJK-1 (final depth: 650.5m, vertical) was drilled in the Eastern Orebody in 1995 and the copper mineralization (Ore Horizon 4- I) could be observed between the depths of 598.0m and 605.8m. MJK-2 (final depth: 700.0m, vertical) was drilled in the Central Orebody in 1996. In this drill hole, the copper and lead mineralization (Ore Horizon 4- I) could be observed between the depths of 605.4m and 619.7m, and weak copper mineralization (Ore Horizons 3-VI and 3-II) between the depth of 630.0m and 635.7m and between 688.9m and 692.5m respectively.

The location (X,Y), elevation (m), drillhole inclination, and drilled length(m) are shown in Table 2-5-1. The Drilling program, equipment, consumables including diamond bits, and operational results are listed in Appendices 8, 9, 10, 11 (MJK-1) and in Appendices 19, 20, 21, 22 (MJK-2), respectively.

Table 2-5-1 Outline of the Drill Hole "MJK-1" and "MJK-2" in the Zhaman-Aibat Ore Deposit

Drill Hole	Location		Elevation (m)	Hole Depth (m)	Inclination
	E-W Coordination	N-S Coordination			
MJK-1	96,570	90,210	357.04	650.50	vertical
MJK-2	92,500	90,200	336.90	700.00	vertical

5-2 Survey Method

Drilling Schedule

The drilling schedules of the MJK-1 and the MJK-2 drillholes are summarized below;

MJK-1

August 11, 1995 : Signing of Drilling Contract Agreement
August 12 : Selection and Determination of Drillsite
August 13 : Preparation of Drilling, Commencement of Drilling
August 25 : Drilling Completedtxo to 650.5m
August 26 : Dismantle and Transportation

MJK-2

July 11, 1996 : Signing of Drilling Contract Agreement
July 18 : Selection and Determination of Drillsite
July 19 : Preparation of Drilling, Commencement of Drilling
August 3 : Drilling Completed to 700.0m
August 4 : Dismantle and Transportation

Drilling Works and Drillng Team

The drilling works in the Zhaman-Aibat Area were conducted by the drilling team of the Zhezkazgangeologiya under the supervision of the Japanese survey team. The drill holes were completed by a drilling rig manufactured in the former USSR. The drilling survey team consisted of one drilling operator and one assistant operator working in two shifts of 12 hours over a period of 15 days. The members of drilling team are listed below;

Drilling camp manager :1
Chief engineer :1
Electrical engineer :1
Mechanical engineer :1
Mechanic :1
Geologist :1
Drilling master :1
Operator :2
Assistant Operator :2
Driver (bulldozer,etc.) :4
Electrician :1
Generator technician :1
Cook :1
Assistant cook :1

Transportation of Drilling Equipment and Water

At the drilling site, the drilling rig, pumps, derrick and other equipment were transported by two bulldozers. Water for drilling was transported from wells located 7 - 10km to the north by tankers with $5m^3$ capacity. Electric power was supplied to the drilling sites by a generator in the exploration camp via a transformer.

Drilling Method

MJK-1 and MJK-2 were drilled by conventional and wire-line drilling methods. The casing programs are detailed, below;

MJK-1

surface to 4.0m :112mm ϕ cemented carbide bit with 108mm ϕ casing, by conventional drilling method

4.0 to 38.3m : 93.3mm ϕ cemented carbide bit with 89mm ϕ casing, by conventional drilling method

38.3 to 650.5 m : 59mm ϕ diamond bit without casing, by wire-line method

MJK-2

surface to 3.5m :112mm ϕ cemented carbide bit with 108mm ϕ casing, by conventional drilling method

3.5 to 19.6m : 93.3mm ϕ cemented carbide bit with 89mm ϕ casing, by conventional drilling method

38.3 to 700.0m : 59mm ϕ diamond bit without casing, by wire-line method

General rock faces were represented by the alternation with aleurolite (siltstone) and sandstone. During drilling operations, there were no major interruptions caused by problems, such as sloughing in the drillhole, mechanical breakdown or lack of materials, and the drilling was completed earlier than scheduled. The average daily drilling rate of MJK-1 was 52.0m and the overall core recovery as a percentage of total length of drillhole was calculated to be as high as 98.5%. The average daily drilling rate of the MJK-2 was 50.0m and the overall core recovery as a percentage of total length of drillhole was calculated to be as high as 97.7%. Inclination of these drill holes ranged from 0 degree 30 minutes to 1 degree15 minutes.

5-3 Survey Results of Drill Hole MJK-1

5-3-1 Geology and Mineralization

The geological core logging (scale 1/200) of drill hole MJK-1 is shown in Plate 19. Detailed geological core logging of the mineralized zone and geological sections are shown in Fig.2-5-2, 2-5-3, 2-5-4, 2-5-6. The copper mineralization could be observed between the depths of 598.0m and 605.78m at the bottom of the Zhezkazgan Formation. The outline of general geology is listed below;

Depth	Dominant rock facies	Formation
0.0m to 403.2m	Reddish-brown sandstone and siltstone	Zhiderisai Formation
403.2m to 609.3m	Grey sandstone	Zhezkazgan Formation
69.3m to 650.5m	Alternation beds of grey sandstone and reddish brown siltstone	Taskduk Formation

General Geology

000.0 - 238.0m : Reddish brown siltstone with inter layers of reddish brown sandstone (thickness 0.1 - 5.0m). Numerous layers and veinlets of gypsum with thickness of 5 cm max.

238.0 - 403.2m : Reddish brown, fine - medium grained sandstone, with interlayers of reddish brown siltstone (thickness 0.5 - 15.0m). Carbonate-clay cements occur in entire horizon. Calcite and gypsum veinlets are observed from the surface to this level corresponds to Zhidelisai Formation.

403.2 - 403.5m : Greenish-grey, fine-grained sandstone. slightly calcareous, grey sandstone appeared.

403.5 - 438.8m : Alternation with reddish brown and grey, fine - medium grained sandstone and siltstone. Pyrite disseminated and patched, weak carbonitization and silicification.

438.8 - 592.5m : Grey - greenish grey, fine - medium grained sandstone with interlayer of grey-greenish siltstone. Carbonate cement and film. Pyrite disseminated, weak silicification and chloritization.

592.5 - 597.6m : Grey - greenish grey, fine - medium grained sandstone,thin interlayers of greenish grey siltstone laminating and graded bedding structures observed, weak to strong pyrite dissemination.

597.6 - 608.3m : Dark grey - grey, fine - medium grained sandstone, laminated, mineralized zone (chalcocite>>bornite>galena, chalcopyrite(?), pyrite).

608.3 - 609.3m : Light brown - light greenish grey Raimundo conglomerate consisting of angular pebbles of white or pink-colored muddy sandstone, weak chloritization, pyrite disseminated, no copper mineralization observed. 403.2 - 609.30m correlated to the Zhezkazgan Formation.

609.3 - 605. 5m(bottom) : Alternation with reddish brown fine - coarse sandstone and siltstone, thin interlayers of interformational conglomerate (thickness: 0.1 - 0.9m), matrix; carbonitization, corresponds to the upper most of the Taskuduk Formation.

Mineralization

Main copper mineralization occurs at the depth between 598.0 - 605.78m (length :7.78m) correlating to the grey sandstone formation of the Zhezkazgan Formation. Weak copper mineralizations containing 0.03 - 0.16%Cu are observed both above and below the main copper mineralization, but no visible copper minerals can be observed in this zone. The inclination of the mineralized zone shows moderate angles ranging from flat to approximately 10 degrees. The horizon of the main mineralized zone may correspond to the horizon 4-1 in the Zhezkazgan Formation and is summarized below:

Depth	: 598.00m - 605.78m
Thickness of mineralization	: 7.78m
Metal content	: 3.78%Cu, 1.17%Pb, 0.03%Zn, 22.7g/tAg, 11.2g/tRe
Ore minerals	: chalcocite>>bornite>galena, chalcopyrite
Mineral occurrence	: light grey - greenish grey, fine - medium grained sandstone, interlayers of grey siltstone and interformational conglomerate consisting of pebble - granule, limestone, shale, dacite.
Alteration	: silicification, chloritization.

5-3-2 Laboratory Test

5-3-2-1 Chemical Analysis

76 samples were taken at the depth 435m to 642.3m including dense sampling of the mineralized zone of MJK-1. In order to compare the assay results between the Zhaman-Aibat Deposit and the Zhezkazgan deposit, 4 representative samples from the South Mine of the Zhezkazgan Deposit were assayed.

Sampling intervals and length of split cores are;

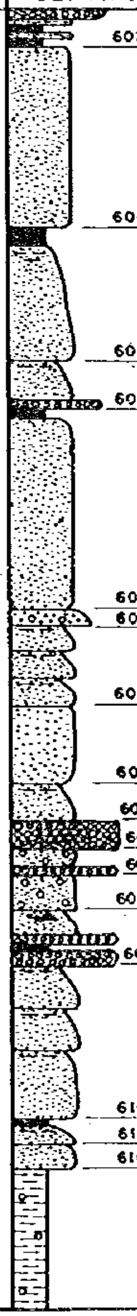
435.0 - 585.0m	: 31 samples, continuous sampling with 0.3m sampling interval.
591.0 - 598.0m	: 14 samples, continuous sampling with 0.5m sampling interval
598.0 - 605.8m	: 21 continuous samples in the mineralization zone, split sampling interval ranges 0.13 - 0.98m due to the lithology and mineralization.
605.8 - 610.0m	: 8 samples, continuous sampling with 0.5m sampling interval.
621.0 - 642.0m	: 2 samples, spot sampling.

Assay elements and detection limit on each element are shown below;

elements	methods	detection limits
Au	: Fire Assay	5 ppb
Ag	: Aqua-Regia digest	0.01 oz/ton
	Fire Assay-Gravimetric	3 g/ton
Cu	: Aqua-Regia digest	0.01 %
Pb	: Aqua-Regia digest	0.01 %
Zn	: Aqua-Regia digest	0.01 %
Re	: Neutron Activation Analysis	1 ppm
FeO	: Titration	0.01 %

SCALE (m)	COLUMN DEPTH (m)	DESCRIPTION
591	590.94	590.94-591.13m: Dark gray, Fine alternating beds of laminated fine sandstone and siltstone, bedded horizontally.
	591.13	
	591.30	591.13-592.16m: Light gray, medium grained sandstone, cross-bedded at the angle 5°-15°, strongly disseminated by pyrite.
592	592.16	592.16-592.70m: Dark gray, massive, medium grained sandstone (arenite), strongly disseminated by pyrite. Contact with underlying layer is at the angle 10°.
	592.72	592.70-593.12m: Alternating beds of medium grained sandstone and siltstone. Weak pyrite dissemination is observed only in the sandstone layers.
593	593.12	
		593.12-593.95m: Light gray, sandstone (arenite) with carbonaceous cement, with graded bedding structure (coarse-medium-fine-muddy) at the angle 0°-5°, weakly disseminated by pyrite.
594	593.95	
	594.35	593.95-594.64m: Gray, massive sandstone, pyrite dissemination and concretion is observed at the interval from 594 50 to 594.64m.
	594.64	
595	595.25	594.35-595.25m: Light gray, sandstone (arenite) with carbonaceous cement, with horizontal graded badding structure (coarse-medium-fine-muddy).
596	595.80	595.25-595.80m: Brown, massive, medium grained sandstone including mudstone patches.
	596.40	595.80-596.40m: Light gray, sandstone (arenite) with horizontal graded bedding structure, very fine grained pyrite disseminated.
597	596.91	596.40-597.60m: Pale greenish gray (596.4-596.9m) - dark gray (596.9-597.9m), laminated sandstone with graded bedding structure. Pyrite dissemination (including a small quantity of chalcopyrite) is observed at the coarse-medium grained sandstone.
	597.33	
	597.60	
598		597.60-598.50m: Greenish dark gray, siltstone with frequent interlayers of fine grained sandstone. Dissemination of chalcocite (range in thickness from 2cm to 3cm) and pyrite are observed in the sandstone layers.
	598.50	
	598.85	598.60-598.85m: Light gray, medium grained sandstone with irregular shaped mud balls. Pyrite dissemination is observed at the bottom of the layer.
599	599.05	
	599.29	
	599.65	
	599.85	598.85-600.12m: Dark gray-gray, Thinly alternating beds of fine grained sandstone(arenite) and siltstone, bedded at the angle 0°-5°. Concentrations of chalcocite (>galena, bornite, chalcopyrite) and weak pyrite dissemination are observed mainly in the sandstone layers.
600	600.12	
601	601.02	600.12-602.75m: Dark gray, massive, medium grained sandstone(arenite) and carbonate-rich granule-pebble conglomerate (including angular gravels of limestone-shale-dacite). These layers are disseminated by chalcocite (>galena, bornite, chalcopyrite>pyrite). Strong concentrations of chalcocite (grain size: 0.2mm-2mm) are observed within the intervals 600 90-601 27m and 602 20-602.75m.
	601.27	
	601.65	
602	602.20	
	602 50	

SAMPLNG LOCATION	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Re (g/t)	S (%)
591.00 ~591.50	0.01	0.05	0.02	0.3	3	0.89
591.50 ~592.00	0.01	0.01	0.01	1.0	< 1	0.78
592.00 ~592.50	0.02	0.04	0.01	0.3	3	1.05
592.50 ~593.00	0.06	0.07	0.02	0.7	< 1	0.74
593.00 ~593.50	0.03	0.03	0.03	0.3	< 1	0.74
593.50 ~594.00	0.07	0.10	0.02	0.3	6	0.48
594.00 ~594.50	0.09	0.01	0.02	0.7	4	0.46
594.50 ~595.00	0.09	0.01	0.04	0.3	< 1	0.59
595.00 ~595.50	0.16	0.02	0.09	0.7	< 1	0.49
595.50 ~596.00	0.11	0.04	0.05	1.0	< 1	0.68
596.00 ~596.50	0.06	0.03	0.03	0.3	< 1	0.64
596.50 ~597.00	0.16	0.01	0.03	0.7	< 1	0.85
597.00 ~597.50	0.04	0.02	0.02	0.3	< 1	0.54
597.50 ~598.00	0.04	0.07	0.05	0.3	3	1.41
598.00 ~598.48	0.53	0.11	0.23	1.4	4	1.71
598.48 ~599.03	0.32	0.02	0.22	1.0	6	0.86
~599.21	2.02	0.03	0.02	6.9	1	1.36
599.21 ~599.82	1.18	0.08	0.02	6.9	1	1.35
~600.02	14.50	1.82	0.02	37.4	9	4.78
600.02 ~600.40	0.51	3.27	0.01	3.4	11	0.69
600.40 ~600.77	1.54	1.04	0.01	7.9	11	0.62
600.77 ~601.75	1.34	6.54	< 0.01	10.6	34	1.38
601.75 ~602.17	12.00	0.08	0.01	85.7	40	3.04
602.17 ~602.68	4.99	0.26	< 0.01	26.4	14	1.34



SCALE (m)	COLUMN DEPTH (m)	DESCRIPTION
	602.75	602.75-604.05m: Brown, massive, medium grained sandstone, disseminated by chalcocite (> chalcopyrite = pyrite). Grain size; Cc.0 5mm-1mm, Cp,Py<0 5mm
603		
604	604.05	604.05-605.32m: Gray, medium grained sandstone, thinly bedded at the angle 3°-7°. Chalcocite dissemination and thin layers (thickness:1-2mm) of chalcocite concentrations are observed. Angle of the chalcocite layers is conformable to the bedding structure of the sandstone. Thin layer of pebble conglomerate (including angular fragments of limestone and mudstone) is observed with in the interval 605 25-605.32m.
605	605.00	
	605.32	605.32-606.87m: Gray, massive, medium grained sandstone. Chalcocite dissemination and thin layers (thickness 2-3mm, horizontal) of chalcocite concentrations are observed within the interval 605 61-605 78m. Weak Dissemination by Chalcocite-Chalcopyrite (grain size;0 5-1mm) is described within the interval 605.75-605 85m. Thin layer of very coarse grained sandstone (including fragments of mudstone) is observed with in the Interva'606.77-606.87m.
606		
	606.77	
	606.87	
607		606.87-608.27m: Dark gray, medium grained sandstone, with horizontal graded bedding structure. Very weak Dissemination by pyrite is occurred within the interval 607.50-607.98m.
	607.45	
	607.98	
608	608.27	
	608.46	
	608.65	608.27-609.30m: Brownish light gray-greenish light gray, intraformational conglomerate (RAUMUNDO Conglomerate), consisting of angular fragments of white or pink-colored limestone and siltstone (sizing from 5 x 5mm to 15 x 30mm) and cement of green colored (caused by weak chloritization) muddy sandstone. At the bottom of the layer, cement is represented by red sandstone. No mineralization observed.
609	608.90	
	609.30	
610		609.30-610.75m: Gray (partially brown), fine-medium grained sandstone(arenite) with siliceous-carbonaceous cement with horizontal graded bedding structure. Contact with underlying layer is wavy. Very weak pyrite dissemination is observed.
	610.37	
	610.58	
	610.75	610.75m- : Reddish brown, siltstone with indistinct bedded structure. Calcite concretions with size 0 3 x 0.6cm are occurred. No mineralization observed
611		

SAMPLING LOCATION	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Re (g/t)	S (%)
~602.68						
602.68 ~603.10	15.30	0.21	< 0.01	119	20	4.07
603.10 ~603.66	1.96	< 0.01	< 0.01	11.3	4	0.62
603.66 ~604.05	2.22	0.74	< 0.01	10.3	5	0.78
~604.15	1.34	< 0.01	0.01	6.9	2	0.40
604.15 ~604.65	2.35	< 0.01	< 0.01	14.4	< 1	0.68
604.65 ~605.00	4.59	< 0.01	0.01	27.8	< 1	1.49
~605.20	3.50	< 0.01	< 0.01	23.7	< 1	1.02
~605.34	10 30	< 0 01	< 0 01	38 7	4	2 61
~605.47	2 62	0 03	< 0 01	16 1	< 1	0.76
~605.61	1 85	< 0 01	0 01	12 0	2	0.57
~605.78	7.51	< 0 01	< 0 01	39.8	< 1	1 96
605.78 ~606.50	0.03	< 0.01	0.01	0.3	< 1	0.13
606.50 ~607.00	0.02	< 0.01	0.01	0.3	< 1	0.09
607.00 ~607.50	0.02	< 0.01	0.01	0.3	3	0.19
607.50 ~608.00	0.01	< 0.01	0.01	0.0	< 1	0.13
608.00 ~608.50	0.02	< 0.01	0.01	0.0	< 1	0.04
608.50 ~609.00	0.03	0.01	0.02	0.0	< 1	0.02
609.00 ~609.50	0.12	0.02	0.01	0.3	2	0.07
609.50 ~610.00	0.04	0.03	0.01	0.3	< 1	0.01

LEGEND

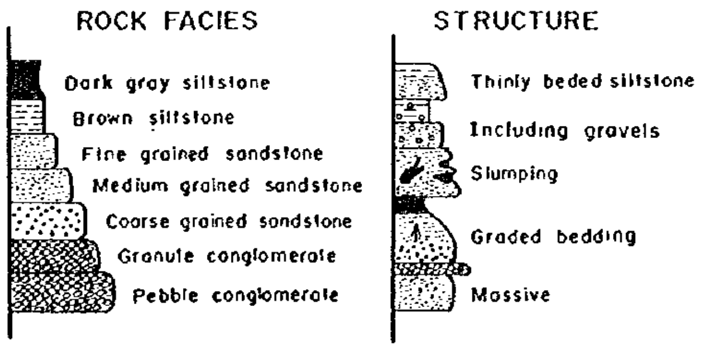
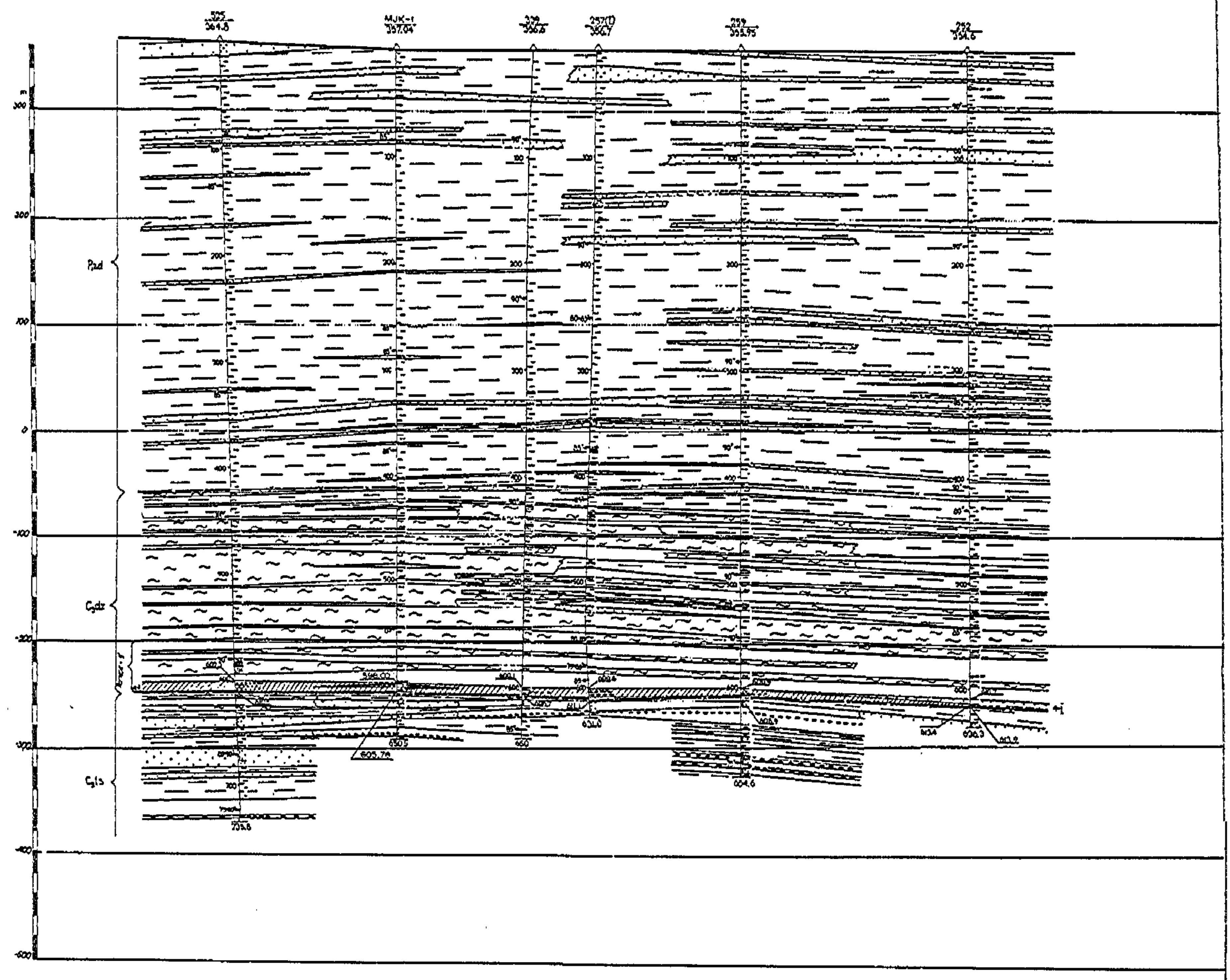
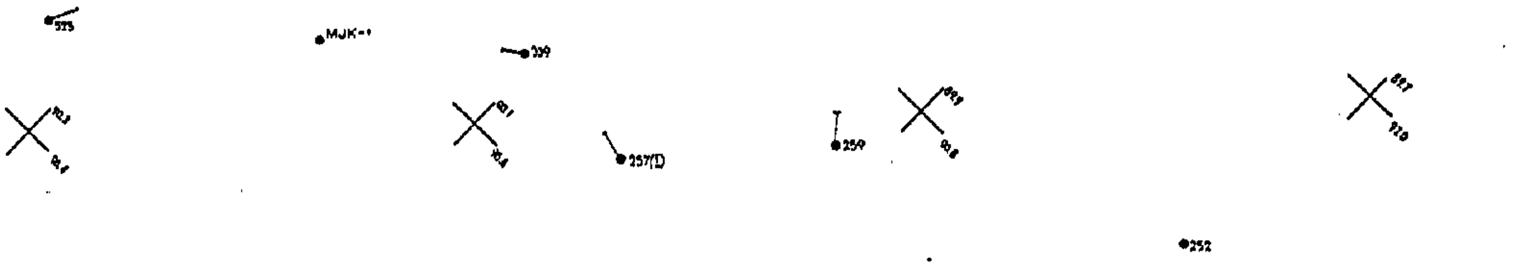


Fig. 2-5-2 Detailed Geological Logging for the Mineralized Zone of the Drill Hole MJK-1



LEGEND

- Aleurolite aleurosandstone, fine-grained sandstone
  - 1. red
  - 2. gray
- Fine-coarse-grained sandstone
  - 1. red
  - 2. grayish-red
  - 3. reddish-gray
  - 4. gray
- Conglomerate, gritstone
  - 1. Intraformational
  - 2. Interformational ("Raimundo")
- Ore
  - 1. copper ore (balanced)
  - 2. complex ore (balanced)
  - 3. copper ore (off-balanced)
- 1. Boundary of horizon
- 2. Boundary of formations



Well no	mineralization horizon		thickness m	core recovery		grade						Ore type	Deposit	Commercial characteristics
	from	to		m	%	Cu %	Pb %	Zn %	Re g/t	Ag g/t	S %			
525	602.3	610.5	8.2	8.2	100	1.59	0.23	0.14	1.63	4.33	1.39	copper	4-1	balance
339	600.1	603.7	8.6	8.6	100	2.76	-	-	0.70	8.10	0.80	copper	4-1	balance
257(I)	600.8 611.1	611.1 613.1	10.3 2.0	10.3 2.0	100 100	2.11 0.36	- -	- -	- -	5.93	- -	copper copper	4-1 4-1	balanc off-bal. in soil
259	600.9	608.9	8.0	8.0	100	1.61	-	-	0.58	3.88	0.71	copper	4-1	balance
252	610.7	615.4	4.7	4.4	94	0.39	-	-	0.06	1.58	0.09	copper	4-1	off-balance in the roof
	615.4	615.9	0.5	0.5	100	3.04	-	-	0.33	17.0	0.75	copper	4-1	balance
MJK-1	598.00	605.78	7.78	7.55	97	3.78	1.17	0.03	11.2	22.7	1.47	copper	4-1	balance

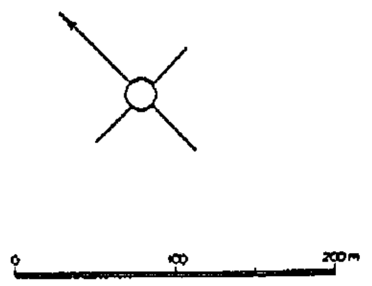
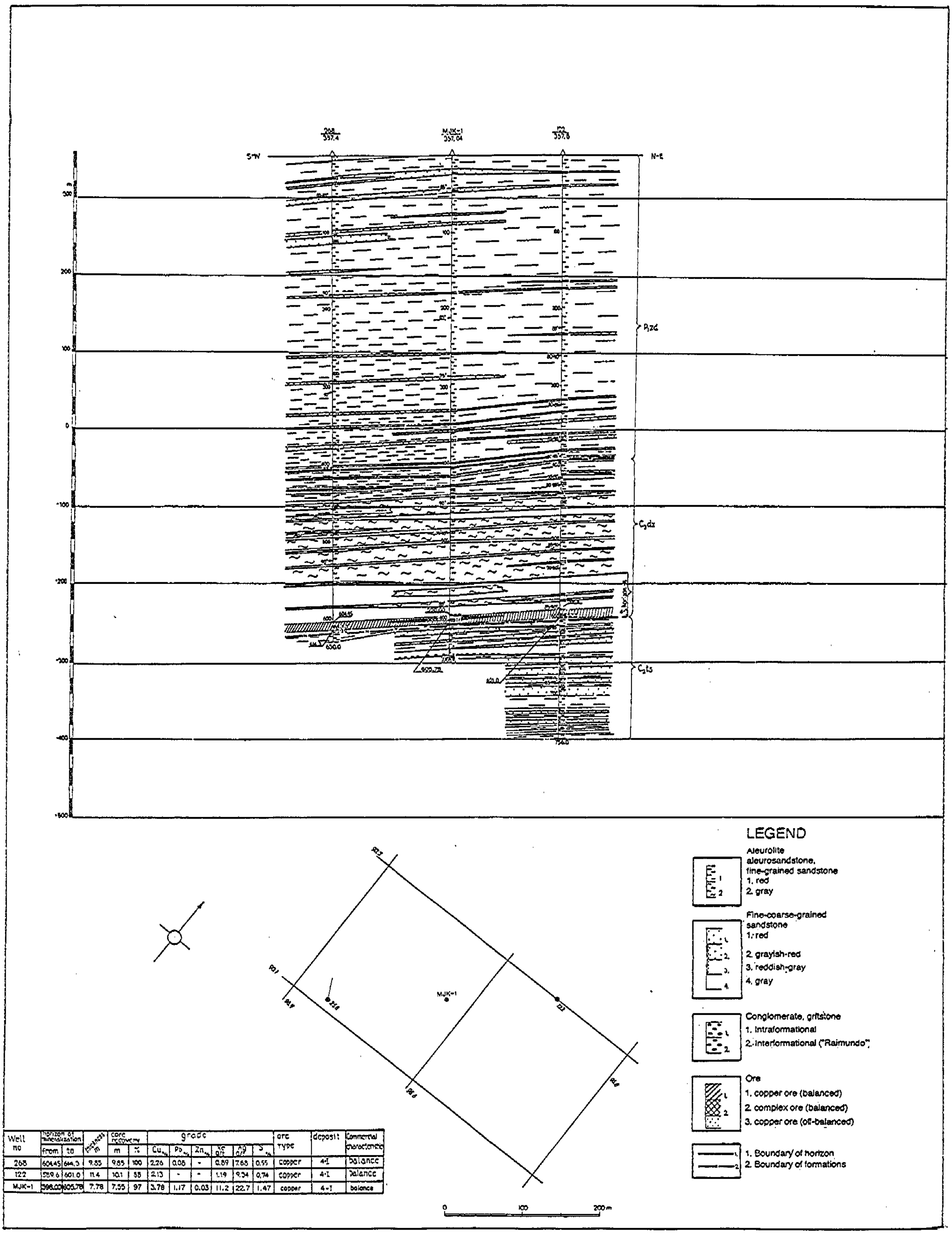


Fig.2-5-3 Geological Section of the Eastern Orebody Along the Line DH525-DH252, Zhaman-Aibat Ore Deposit



Well no	horizon of mineralization		thickness m	core recovery		grade						ore type	deposit	Commercial characteristic
	from	to		m	%	Cu %	Pb %	Zn %	Re g/t	Ag g/t	S %			
268	604.45	614.3	9.85	9.85	100	2.26	0.05	-	0.89	7.68	0.55	copper	4-I	balance
122	589.6	601.0	11.4	10.1	88	2.13	-	-	1.19	9.34	0.74	copper	4-I	balance
MJK-1	598.00	605.78	7.78	7.55	97	3.78	1.17	0.03	11.2	22.7	1.47	copper	4-I	balance

Fig.2-5-4 Geological Section of the Eastern Orebody Along the Line DH268-DH122, Zhaman-Aibat Ore Deposit

elements	methods	detection limits
Fe (total)	: Titration	0.01 %
S (sulphide)	: Gravimetric	0.01 %
S (sulphate)	: Gravimetric	0.01 %
S (element)	: Gravimetric	0.01 %

The assay results of 80 samples are shown in Appendix 13, the outline of the main mineralization zone (4-I horizon, depth:598.00m-605.78m) is summarized below;

	maximum	minimum	average
Cu (%)	15.30	0.32	3.78
Pb (%)	6.54	<0.01	1.17
Zn (%)	0.22	<0.01	0.03
Ag (g/t)	119.0	1.0	22.7
Re (g/t)	40.0	<1.0	11.2

5-3-2-1 Whole Rock Analysis

5 representative samples taken at the depth from 440.0m to 606.5m, were assayed for whole rock analyses. All samples are correlated to grey sandstone. Assay results are shown in Appendix 14. Two samples, namely N22J3 and N26J3, were taken in the mineralization zone. The former sample was collected at the point of 0.98m split sample and due to chemical assay results of this split interval are 1.34%Cu , 6.50%Pb, 1.34%Fe(total), 1.33%S(total), total content (%) is (lower than) 92.19%.

No significant difference in the chemical content between grey to greenish grey mineralized sandstone and dark grey un-mineralized sandstone could be observed in 12 assay elements.

5-3-2-1 Petrological and Mineralogical Examinations

Microscopic Observations of Thin Sections

5 rock specimens taken at the depth from 203.5m to 644.25m were microscopically observed. The summary of examination results and the related explanatory descriptions are shown in Appendices 16.

95-TS-01 and 95-TS-02 are representative specimens from Red Sandstone Formation corresponding to the Zhidelisai Formation. The matrix of all specimens except 95-TS-06, is cemented by carbonate minerals, mainly calcite. The identified minerals are : quartz, plagioclase, K-feldspar, carbonate minerals(mainly calcite), clay minerals, goethite, opaque minerals. Volcanic rock fragments and quartz in plutonic rocks are commonly observed in the coarse fraction of sandstone.

Specimen 95-TS-06 was taken at the depth of 601.5m in the middle part of the mineralization zone. 95-TS-06 is a typical specimen of copper ore, representing that matrix of angular or subangular grains of coarse grained sandstone are fully cemented by opaque minerals, namely chalcocite.

Microscopic Observations of Polished Sections

9 specimens from the mineralization zone of MJK-1 and one specimen from the South Mine of the Zhezkazgan Deposit, were microscopically observed in the polished sections. The summary of the results and related explanatory descriptions are shown in Appendix 15.

The identified minerals are listed below ;

primary minerals : chalcocite series minerals; such as chalcocite, digenite, djurleite, bornite, galena, chalcopyrite, sphalerite, Ag rich electrum, gersdorffite-cobaltite series minerals, pyrite, zhezkazganite(?)

secondary minerals : covellite, goethite

Of these ore minerals, chalcocite series minerals, such as chalcocite abundant, digenite and djurleite rare are observed, accompanied by small amounts of bornite occurring as an interstice-filling product among sedimentary grains and as veinlets in clay rich parts. Ag rich electrum (max 0.1 mm in size) is observed in veinlets of less than 2 mm in width, which consist of chalcocite like minerals (chiefly chalcocite) and less bornite in dark bluish green clay. In this year's campaign much effort was paid to identifying zhezkazganite in the ores obtained by MJK-1 drill. It is concluded that zhezkazganite-like minerals occur rarely in the ore from the Zhezkazgan deposit. It is observed in chalcocite-like minerals, but it is difficult to identify because of its tiny mineral grain size. In the polished sections of MJK-1, zhezkazganite could not be identified. Further observations are needed in next year's campaign.

Qualitative Analysis of Ore Minerals by Electron Probe Microanalyser

Qualitative analysis of ore minerals on 3 specimens of MJK-1 core were carried out by electron probe microanalyser (EPMA). The analytical results produced by X-ray image color mapping techniques are shown in Appendix 18.

It is concluded that the main mineralization zone of MJK-1 drillhole consists of copper and lead minerals, such as chalcocite, bornite, galena, stromeyerite. Silver is found not only in silver minerals, namely stromeyerite, but also in Ag rich electrum which occurs in chalcocite and bornite. The result of the super high grade ore (Cu:30%) from the South Mine in the Zhezkazgan deposit showed that Ag content is much higher in chalcocite series minerals than in bornite.

5-4 Survey Results of Drill Hole MJK-2

5-4-1 Geology and Mineralization

The geological core logging (scale 1/200) of drill hole MJK-2 is shown in Plate 20. Detailed geological core logging of the mineralized zone and geological sections are shown in Fig.2-5-6, 2-5-7, 2-5-8 and 2-5-9. Copper and lead mineralization could be observed in the lower part of the Zhezkazgan Formation (ore horizon 4-I), and in the middle - upper part of the Taskduk Formation (ore horizon 3-II, and 3-VI).

The outline of the general geology is summarized below;

Depth	Dominant rock facies	Formation
0.0m to 425.9m	Reddish-brown siltstone and sandstone	Zhiderisai Formation
425.9m to 619.9m	Grey sandstone	Zhezkazgan Formation
619.9m to 700.0m (bottom)	Alternation beds of grey sandstone and reddish brown siltstone	Taskduk Formation

General Geology

000.0 - 321.6m : Purplish brown - reddish brown siltstone with inter layers (0.5m to 3m in thickness) of reddish brown sandstone, and fine alternation beds of siltstone and sandstone. Numerous layers and veinlets of gypsum and anhydrite with maximum thickness of 20cm. Carbonate cements and concretions occur in entire horizon.

321.6 - 353.2m : Purplish brown - reddish brown siltstone with inter layers (0.5m to 1.5m in thickness) of reddish brown sandstone. Carbonate cements and concretions occur in entire horizon. In some places brecciated zones are observed.

353.2 - 377.9m : Brown, fine alternation beds of siltstone and fine-grained sandstone with inter layers (1m to 2m in thickness) of fine grained sandstone. Numerous brecciated zones with width of 20 - 100cm. The inclination ranges from 45 to 75 degrees.

377.9 - 425.9m : Reddish brown siltstone with inter layers (0.5m to 3.5m in thickness) of light brown fine grained sandstone. Numerous veinlets of gypsum and chlorite films are observed. The interval from the surface to the depth of 425.9m correlates with the Zhiderisai Formation.

425.9 - 543.3m : Alternation beds of reddish brown siltstone and greenish grey sandstone. Abundant dissemination and patches of pyrite and veinlets of marcasite and calcite are observed in 461 - 543m. Bituminous (oil saturated) fine grained black sandstone in the interval 493.4 - 494.0m, and 533.2 - 541.0m.

543.3 - 619.9m : Dark grey fine grained sandstone and dark brown siltstone. Partially including intraformational granule - pebble conglomerate layers (Raimundo Conglomerate). Pyrite dissemination is observed in the interval 543.3 -

599.6m. Copper and lead mineralization occurs in the interval 605.4 - 619.7m correlated with Ore Horizon 4-I. The interval 425.9 - 619.9m correlates with the Zhezkazgan Formation.

619.9 - 700.0m : Greenish grey fine grained sandstone and dark brown siltstone with thin layers of (granule - pebble) conglomerate including patches of siltstone. Pyrite dissemination is observed in the interval 673.2 - 679.5m. Copper mineralization occurs in the interval 630.0 - 635.7m correlated with Ore Horizon 3-VI, and copper and lead mineralization occurs in the interval 688.9 - 692.5m correlated with Ore Horizon 3-II. The interval 619.9 - 700.0m (bottom of the drill hole) correlates with the Taskduk Formation.

Mineralization

Main mineralization zones are observed at between 605.40 - 619.65m depth (length :14.25m, Ore Horizon 4-I), between 630.00 - 635.70m (length :5.70m, Ore Horizon 3-VI) and between 688.85 - 692.45m depth (length :3.60m, Ore Horizon 3-II). The inclination of the mineralized zone shows moderate angles ranging from Horizontal to approximately 10 degrees. The main mineralized zones are summarized below:

Upper part of Ore Horizon 4-I

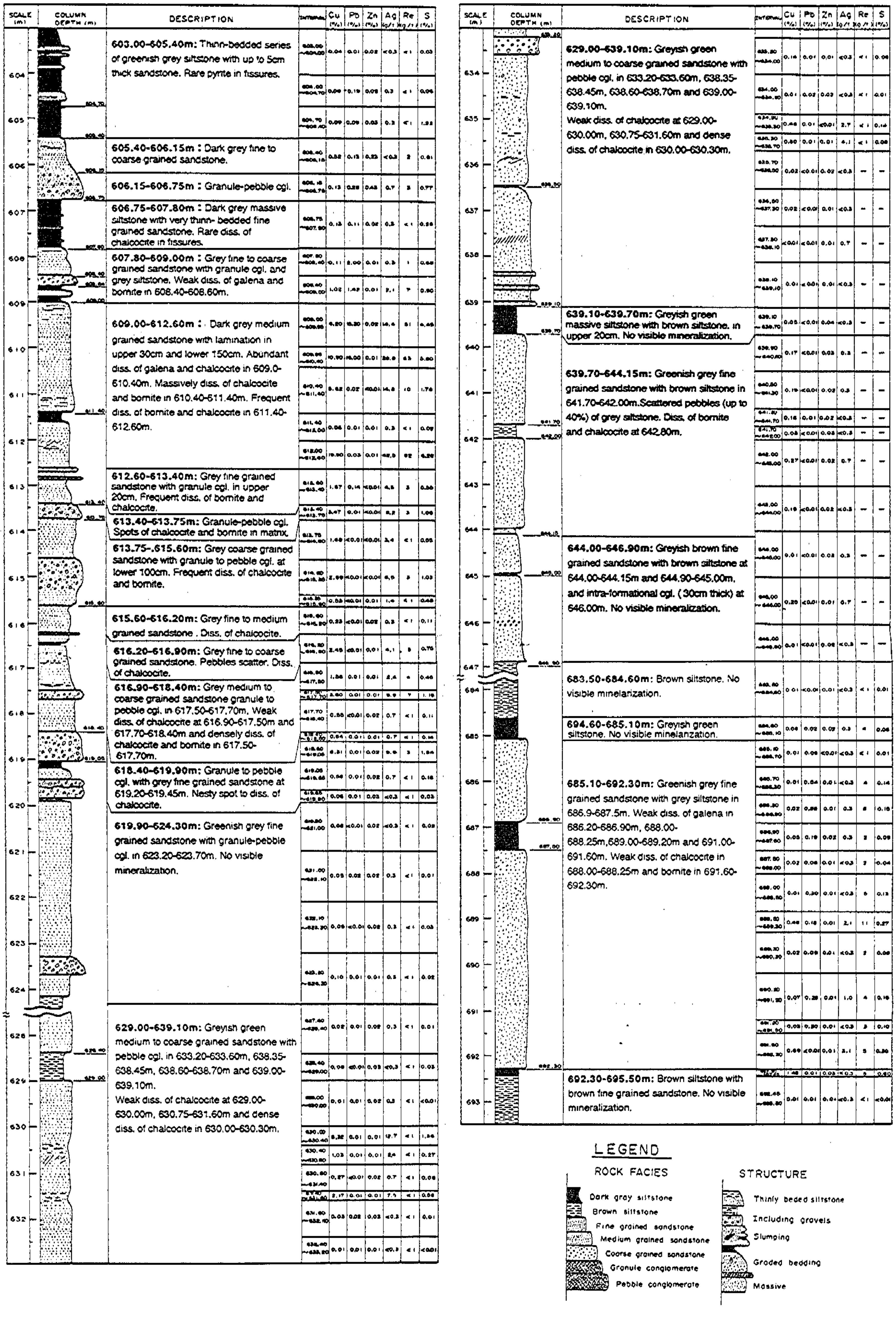
Depth : 605.40m - 613.40m
Thickness of mineralization : 8.00m
Metal content : 3.88%Cu, 3.04%Pb, 0.06%Zn, 9.09g/tAg, 18.3g/tRe
Ore minerals : chalcocite>> bornite, galena>> chalcopyrite, pyrite, covellite, electrum, stromeyerite
Ore structure : disseminated, patches, laminated
Country rock : grey - dark grey, fine - coarse grained sandstone (with carbonate cement) and siltstone, with thin interlayers of granule conglomerate (Raimundo Conglomerate)

Lower part of Ore Horizon 4-I

Depth : 613.40m - 619.65m
Thickness of mineralization : 6.25m
Metal content : 1.89%Cu, 0.00%Pb, 0.01%Zn, 3.68g/tAg, 1.7g/tRe
Ore minerals : chalcocite>> bornite
Ore structure : disseminated, patches
Country rock : light grey, medium - coarse grained sandstone with carbonate cement and minor siltstone, with interlayers (20 - 100cm in thickness) of granule - pebble conglomerate (Raimundo Conglomerate)

Ore Horizon 3-VI

Depth : 630.0m - 635.70m
Thickness of mineralization : 5.70m
Metal content : 0.67%Cu, 0.01%Pb, 0.01%Zn, 1.87g/tAg, 0.0g/tRe
Ore minerals : chalcocite>> bornite> pyrite
Ore structure : disseminated, lamonated
Country rock : Greenish grey, midium - coarse grained sandstone with carbonate cement



603.00-605.40m: Thinn-bedded series of greenish grey siltstone with up to 5cm thick sandstone. Rare pyrite in fissures.

605.40-606.15m : Dark grey fine to coarse grained sandstone.

606.15-606.75m : Granule-pebble cgl.

606.75-607.80m : Dark grey massive siltstone with very thinn- bedded fine grained sandstone. Rare diss. of chalcocite in fissures.

607.80-609.00m : Grey fine to coarse grained sandstone with granule cgl. and grey siltstone. Weak diss. of galena and bornite in 608.40-608.60m.

609.00-612.60m : Dark grey medium grained sandstone with lamination in upper 30cm and lower 150cm. Abundant diss. of galena and chalcocite in 609.0-610.40m. Massively diss. of chalcocite and bornite in 610.40-611.40m. Frequent diss. of bornite and chalcocite in 611.40-612.60m.

612.60-613.40m: Grey fine grained sandstone with granule cgl. in upper 20cm. Frequent diss. of bornite and chalcocite.

613.40-613.75m: Granule-pebble cgl. Spots of chalcocite and bornite in matrix.

613.75-.615.60m: Grey coarse grained sandstone with granule to pebble cgl. at lower 100cm. Frequent diss. of chalcocite and bornite.

615.60-616.20m: Grey fine to medium grained sandstone . Diss. of chalcocite.

616.20-616.90m: Grey fine to coarse grained sandstone. Pebbles scatter. Diss. of chalcocite.

616.90-618.40m: Grey medium to coarse grained sandstone granule to pebble cgl. in 617.50-617.70m. Weak diss. of chalcocite at 616.90-617.50m and 617.70-618.40m and densely diss. of chalcocite and bornite in 617.50-617.70m.

618.40-619.90m: Granule to pebble cgl. with grey fine grained sandstone at 619.20-619.45m. Nesty spot to diss. of chalcocite.

619.90-624.30m: Greenish grey fine grained sandstone with granule-pebble cgl. in 623.20-623.70m. No visible mineralization.

629.00-639.10m: Greyish green medium to coarse grained sandstone with pebble cgl. in 633.20-633.60m, 638.35-638.45m, 638.60-638.70m and 639.00-639.10m.
Weak diss. of chalcocite at 629.00-630.00m, 630.75-631.60m and dense diss. of chalcocite in 630.00-630.30m.

629.00-639.10m: Greyish green medium to coarse grained sandstone with pebble cgl. in 633.20-633.60m, 638.35-638.45m, 638.60-638.70m and 639.00-639.10m.
Weak diss. of chalcocite at 629.00-630.00m, 630.75-631.60m and dense diss. of chalcocite in 630.00-630.30m.

639.10-639.70m: Greyish green massive siltstone with brown siltstone. in upper 20cm. No visible mineralization.

639.70-644.15m: Greenish grey fine grained sandstone with brown siltstone in 641.70-642.00m.Scattered pebbles (up to 40%) of grey siltstone. Diss. of bornite and chalcocite at 642.80m.

644.00-646.90m: Greyish brown fine grained sandstone with brown siltstone at 644.00-644.15m and 644.90-645.00m, and intra-formational cgl. (30cm thick) at 646.00m. No visible mineralization.

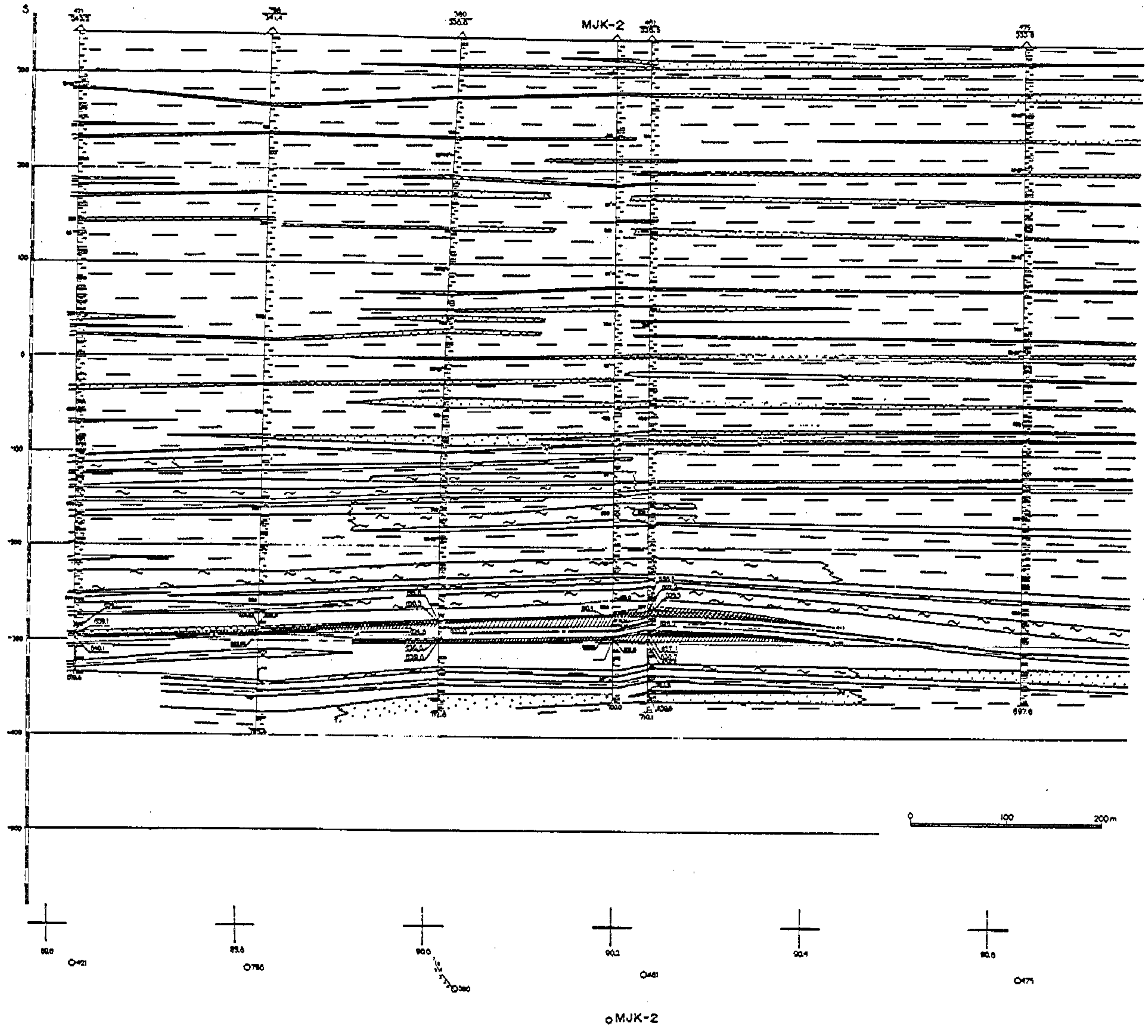
683.50-684.60m: Brown siltstone. No visible minelarization.

694.60-685.10m: Greyish green siltstone. No visible minelarization.

685.10-692.30m: Greenish grey fine grained sandstone with grey siltstone in 686.9-687.5m. Weak diss. of galena in 686.20-686.90m, 688.00-688.25m,689.00-689.20m and 691.00-691.60m. Weak diss. of chalcocite in 688.00-688.25m and bornite in 691.60-692.30m.

692.30-695.50m: Brown siltstone with brown fine grained sandstone. No visible mineralization.

Fig. 2-5-6 Detailed Geological Logging for the Mineralized Zone of the Drill Hole MJK-2



Well no	horizon of mineralization		thickness m	core recovery		grade						deposit	commercial characteristic	ore type
	from	to		%	m	Cu %	Pb %	Zn %	Ag g/t	Re g/t	S %			
421	635.1	639.1	4.0	100		0.10	0.35	1.80	1.08	2.12	1.35	4-I	balance	Pb-Zn
	639.1	640.1	1.0	100		1.31			16.85	0.54	0.55	4-I+3-VI	balance	copper
798	625.7	626.65	0.95	100		0.06	0.36	0.15	0.52	<0.05	0.47		mineraliz	lead
	626.65	628.6	1.95	100		0.15	2.51	2.04	4.90	0.37	2.27	4-I	balance	Pb-Zn
	628.6	632.05	3.45	100		4.02	2.66	0.90	8.56	12.0	2.38	4-I	balance	complex
	632.05	632.5	0.45	100		0.28	-	0.04	0.15	0.8	0.14			copper
380	593.6	594.6	1.0	100		0.20							mineraliz	copper
	600.8	601.8	1.0	100		0.22							mineraliz	copper
	616.8	620.3	3.5	94		7.46	3.25	0.32	21.07			4-I	balance	complex
	620.3	624.2	3.9	96		0.94			20.97			4-I	balance	copper
	624.2	626.3	2.1	95		0.26			0.24			4-I	min.in bottom	copper
	635.8	636.6	0.8			0.33			1.7			3-VI	off-balance	copper
	636.6	639.8	3.2	97		1.04			3.86			3-VI	balance	copper
MJK-2	607.8	610.4	2.6	100		4.25	7.93					4-I	balance	complex
	610.4	619.65	9.25	100		3.3						4-I	balance	copper
	630.0	631.6	1.6	100		1.92						3-VI	balance	copper
461	576.9	577.9	1.0	100			0.10	0.46					mineraliz	zink
	593.9	594.4	0.5	100		0.28	0.15	0.30	<0.5				mineraliz	lead
	598.6	602.9	4.3	98		2.17		0.1	13.83	2.75	1.06	4-I	balance	copper
	602.9	609.5	6.6	100		1.25	5.13	0.1	4.73	4.26	1.63	4-I	balance	complex
	624.3	625.3	1.0	100				0.64	<0.5	0.62	1.13	3-VI	off-bal.in top	zink
	625.3	627.1	1.8	100		0.21	0.57	2.05	1.23	3.49	2.26	3-VI	balance	Pb-Zn
	627.1	630.7	3.6	100		4.35	3.60	1.39	10.94	8.58	4.59	3-VI	balance	complex
	630.7	636.6	5.9	100		2.14	0.29	0.1	5.75	3.63	1.76	3-VI	balance	copper
	641.9	642.9	1.0	100		0.32			<0.5				mineraliz.	copper
	662.2	664.2	2.0	100		0.36			<0.5	0.17	<0.1		mineraliz	copper
	687.4	687.9	0.5	100		0.20			<0.5	0.29	<0.1		mineraliz.	copper
	687.9	690.8	2.9	100		2.07			6.21			3-II	balance	copper
475	535.25	635.95	0.7	100		0.26			1.0	0.05	0.32	4-I	mineraliz.	copper

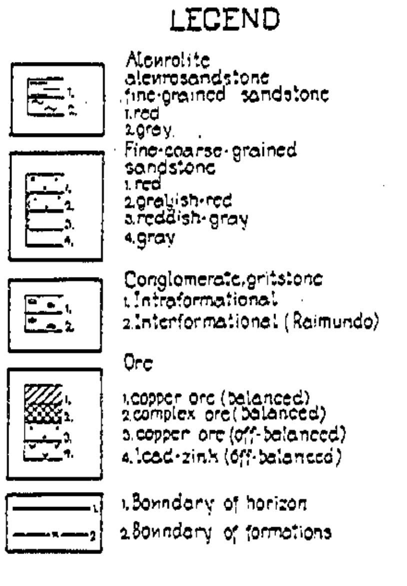


Fig. 2-5-8 Geological Section of the Central Orebody
Along the Line DH421-DH475, Zhaman-Aibat Ore Deposit

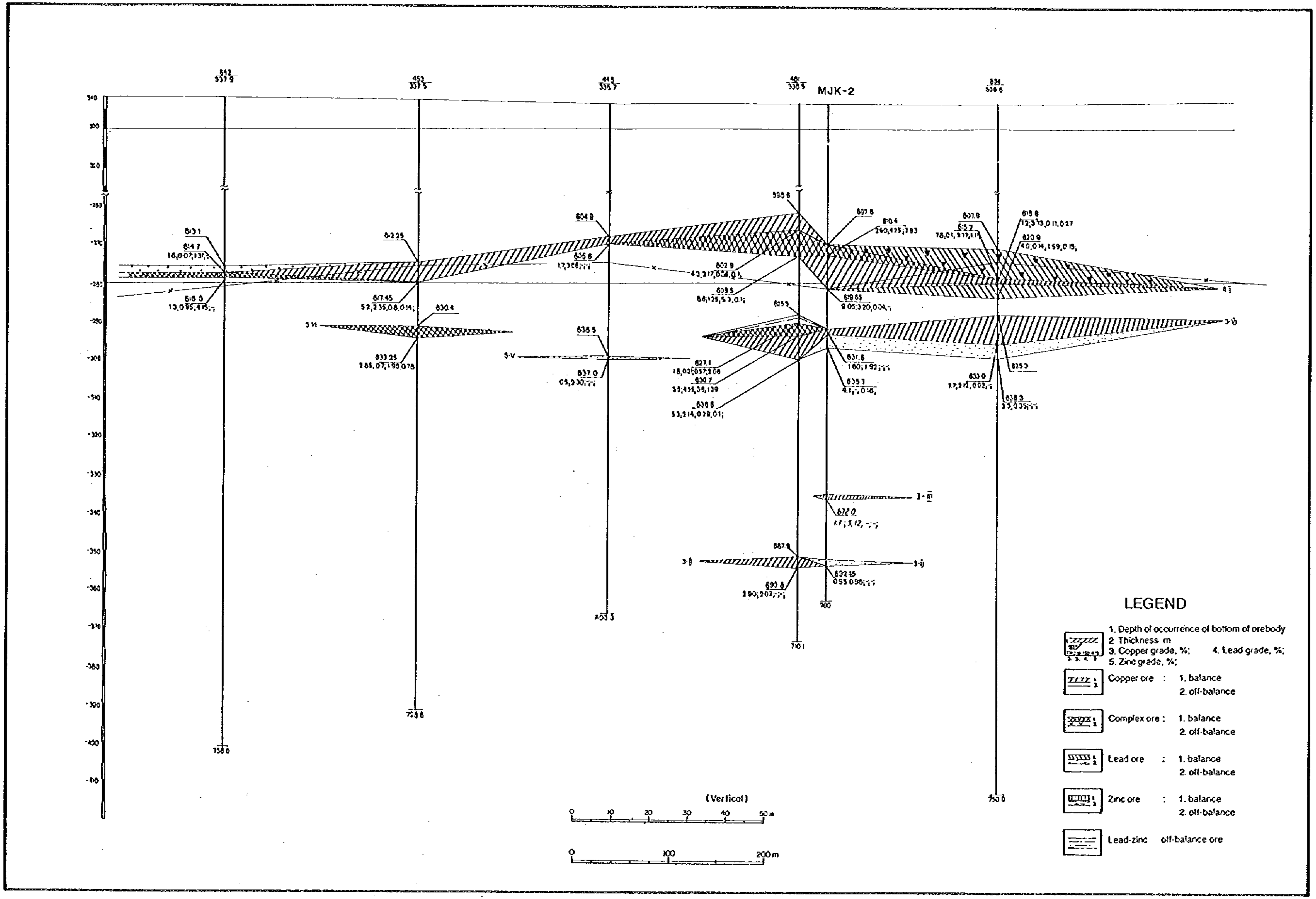


Fig. 2-5-9 Detailed Section of the Central Orebody
Along the Line DH842-DH836, Zhaman-Aibat Ore Deposit

Ore Horizon 3-II

Depth : 688.85m - 692.45m
Thickness of mineralization : 3.60m
Metal content : 0.30%Cu, 0.14%Pb, 0.01%Zn, 1.45g/tAg, 4.6g/tRe
Ore minerals : chalcocite>> bornite> galena, chalcopyrite, pyrite
Ore structure : disseminated
Country rock : Greenish grey, fine grained sandstone with carbonate cement and siltstone

5-4-2 Laboratory Tests

5-4-2-1 Chemical Analysis

133 samples (core length :25 - 100cm) were taken at depths between 599.6m and 700.0m.

Assay elements and the detection limit of each element are shown below;

elements	method	detection limits	No. of samples
Au	: Fire Assay	5ppb	103
Ag	: Aqua-Regia digestion	0.01oz/ton	133
	Fire Assay-Gravimetric	3g/ton	
Cu	: Aqua-Regia digestion	0.01%	133
Pb	: Aqua-Regia digestion	0.01%	133
Zn	: Aqua-Regia digestion	0.01%	133
Re	: Neutron Activation	1ppm	103
FeO	: Titration	0.01%	6
Fe (total)	: Titration	0.01%	103
S (sulphide)	: Gravimetric	0.01%	103
S (sulphate)	: Gravimetric	0.01%	6

The assay results are shown in Appendix 24, the outline of the main mineralization zones are summarized below;

		maximum	minimum	average
Upper part of	Cu (%)	19.90	0.06	3.88
Ore Horizon 4-I	Pb (%)	16.00	0.01	3.04
Depth: 605.40 - 613.40m	Zn (%)	0.43	<0.01	0.06
11 samples	Ag (g/t)	42.5	<0.3	9.09
	Re (g/t)	82.0	<1.0	18.31
Lower part of	Cu (%)	6.31	0.23	1.89
Ore Horizon 4-I	Pb (%)	0.01	<0.01	-
Depth: 613.40 - 619.65m	Zn (%)	0.02	<0.01	-
12 samples	Ag (g/t)	9.9	0.3	3.68
	Re (g/t)	7.0	<1.0	1.69
Ore Horizon 3-VI	Cu (%)	5.32	0.01	0.67
Depth: 630.00 - 635.70m	Pb (%)	0.02	<0.01	-
10 samples	Zn (%)	0.03	<0.01	-
	Ag (g/t)	12.7	<0.3	1.87
	Re (g/t)	<1.0	<1.0	-
Ore Horizon 3-II	Cu (%)	1.48	0.02	0.30
Depth: 688.85m - 692.45m	Pb (%)	0.28	<0.01	0.14
6 samples	Zn (%)	0.03	0.01	0.01
	Ag (g/t)	3.1	<0.3	1.45
	Re (g/t)	11.0	2.0	4.56

5-4-2-2 Whole Rock Analysis

Five samples, namely: a massive, medium grained sandstone (sample No. 96-W1) from the Zhiderisai Formation, a fine grained sandstone (96-W2) and siltstone (96-W3) from the Zhezkazgan Formation, a fine conglomerate bearing fine grained sandstone (96-W5) and siltstone (96-W6) from the Taskduk Formation, were collected and provided for whole rock analysis. No mineralization could be seen in these samples. The assay results are shown in Appendix 25.

SiO_2 content of the samples of the Taskduk Formation (63.69% - 66.74%) were higher than that of other samples (SiO_2 : 56.55% - 59.88%). No significant difference between the SiO_2 content of grey sandstone and brown siltstone could be observed.

The CaO content of fine grained sandstone (96-W2) sampled from the Zhezkazgan Formation showed 13.74% which was much higher than that of other samples whose CaO content were 2.42% ~ 4.22%. The CaO contents represent the amount of $CaCO_3$ which cemented the sand particles.

5-4-2-3 Petrological and Mineralogical Examinations

Microscopic Observations of Thin Sections

10 rock specimens sampled at depths between 608.1m and 688.1m were microscopically observed. 96-TS-01 - 96-TS-08 are representative specimens from the Ore Horizon 4-I, 96-TS-09 is from Ore Horizon 3-VI, and 96-TS-10 is from Ore Horizon 3-II. The summary of examination results and the related explanatory descriptions are shown in Appendices 29.

These rocks will be classified as graywacke cemented in a matrix of sand particles with calcite and / or sulfide minerals.

The sand grains are composed of single crystals of quartz, quartz aggregates derived from chert origin, and weakly altered plagioclases (sericites observed along cleavages). Grains of sedimentary rocks and volcanic rocks and welded-tuffs can be observed. Most of the sand grains are angular.

The matrix is cemented by carbonate minerals, mainly calcite, and sulfide minerals. Chlorite, biotite, and sericite are frequently observed in the thin sections 95-TS-03, 04, 05.

Microscopic Observation of Ppolished Sections

15 specimens from the mineralization zone of MJK-2 were microscopically observed in the polished sections. The summary of the results and related explanatory descriptions are shown in Appendix 26.

The identified minerals are listed below ;

primary minerals : chalcocite series minerals; such as chalcocite, digenite, djurleite, bornite, galena, chalcopyrite, pyrite, sphalerite, stromeyerite, Ag rich electrum

secondary minerals : covellite, goethite

Of these ore minerals, chalcocite, digenite, bornite and galena are abundant and pyrite, chalcopyrite and goethite are common, the other minerals, such as Ag-minerals are rare. The occurrence of the typical complex ore is summarized below;

Aggregates of anhedral grains, 0.2mm - 0.5mm in size, of chalcocite series minerals - galena - bornite (chalcocite is dominant) and a small amount of subhedral - euhedral grains of pyrite occur interstitially among the sedimentary particles and rarely occur as veinlets. Bornite and chalcocite series minerals sometimes show micrographic texture. Occasionally, Ag-rich electrum and stromeyerite, 0.1mm - 0.2mm in size, contacting with chalcocite and bornite are observed. Subhedral - euhedral grains of galena, pyrite and goethite sometimes occur in the sand particles, showing dissemination structure.

By the results of microscopic observations of polished ore sections, it is concluded that the mineralizations of complex ore in the Central Orebody took place in two stages. The first stage was a prodrome mineralization of fine-grained pyrite and galena, and the second was the main mineralization stage of the cementation of the ore sulfide minerals in a matrix of sand grains.

Qualitative Analysis of Ore Minerals by Electron Probe Microanalyser

Qualitative analysis of ore minerals in 13 points were carried out by an electron probe microanalyser (EPMA). The analytical results and X-ray color image mapping are shown in Appendix 30 and 31.

It is concluded that the main mineralization zone of MJK-2 drillhole consists of copper and lead minerals, such as chalcocite, digenite, bornite and galena. It is difficult to distinguish digenite and chalcocite by microscopic observation. The qualitative analysis by the electron probe microanalyser revealed that chalcocite and digenite are common Cu minerals. Cu-S series minerals which show a lower Cu/S ratio than digenite were observed but digenite and chalcocite commonly represented micro-graphic structure. For the determination of details, additional studies are needed.

X-Ray Diffraction Test

5 rock specimens sampled at depths between 608.1m and 630.5m were subjected to X-ray diffraction tests. 4 samples (96-X-01 - 96-X-04) are representative specimens of fine - coarse grained sandstone from Ore Horizon 4-I, and 96-X-05 is a fine grained sandstone from Ore Horizon 3-VI.

The summary of examination results and the related explanatory descriptions are shown in Appendix 32. No significant difference between the mineral assemblages in Ore Horizon 4-I and Ore Horizon 3-VI was seen. The identified minerals are listed below;

Silica mineral : quartz
Clay mineral : sericite
Carbonate mineral : calcite
Sulfide minerals : chalcocite, digenite, bornite, galena, pyrite, chalcopyrite
Others : plagioclase, goethite

Chapter 6 Ore Dressing Test

6-1 Outline of the Ore Dressing Tests

For the purpose of confirming ore stratigraphy and occurrence, and of providing samples for the ore dressing tests, two holes, MJK-1 and MJK-2 were drilled. MJK-1 (final depth: 650.5m, vertical) was drilled in the Eastern Orebody and the Copper Ore could be observed between the depths of 598.0m and 605.8m (Ore Horizon 4- I). MJK-2 (final depth: 700.0m, vertical) was drilled in the Central Orebody. In this drill hole, the Complex Ore could be observed between the depths of 605.4m and 613.4m (upper part of the Ore Horizon 4- I).

The ore dressing tests of the Copper Ore and of the Complex Ore were carried out using the core samples of MJK-1 and MJK-2. The outline of the ore dressing tests for the Copper Ore and for the Complex Ore is shown in Table 1-1-3 and Table 1-1-4, respectively.

6-2 Test of the Copper Ore

6-2-1 Test Samples

Composite Samples and Chemical Analyses

The amount of the composite samples used for the metallurgical tests was about 23kg. It consisted of MJK-1 core (7.78m length, 3.78%Cu) mixed with some rock from the hanging wall and foot wall (0.01～0.16%Cu). The grade of the metallurgical test samples after adjustment was 1.69%Cu, 0.51%Pb, 0.03%Zn, 1.80%Fe, 1.01%S, <0.1 g/t Au and 12 g/t Ag (Table 2-6-1).

The samples were crushed to -6 mesh by a jaw crusher and a crushing roll, and after adjusting the grade, they were divided into 500g subsamples, placed in small plastic bags and kept in a refrigerator.

X-Ray Diffractometer Analyses

As the results of X-ray diffraction, quartz, albite and a small amount of muscovite and kaolinite were detected as the constituent minerals of the country rock (Appendix 33).

Microscopic Observation of Polished Section

The main ore mineral observed under a reflection microscope was chalcocite with small amounts of bornite, galena and pyrite. Minor amounts of chalcopyrite, sphalerite and rutile were also present together with extremely rare covelite. The size of mineral particles was generally fine, 1～500 μ m. The particle size of chalcocite generally ranged between 1 and 500 μ m, most of

Table 2-6-1 Chemical Analysis of Test Samples,
Copper Ore from the Eastern Orebody

Element	Cu	Pb	S	Fe	Zn	SiO_2	Al_2O_3	MgO	CaO	K_2O
%	1.69	0.51	1.01	1.80	0.03	64.90	11.40	1.11	5.31	2.01
Element	**Na_2O**	**Sb**	**As**	**Bi**	**Cd**	**Re**	**Hg**	**Au**	**Ag**	**Total**
%	3.20	<0.01	<0.01	<0.01	<0.01	<0.1	<0.1	<0.1	12	92.97

Hg: ppm, Au and Ag: g/t, others : %

Table 2-6-2 Texture of Main Ore Minerals,
Copper Ore from the Eastern Orebody

Ore Mineral	size	Structures
Chalcocite	30 ~20 μ m	Most of them are scattered in country rock. Includes gangue minerals of less than 20 μ m
Bornite	30 ~100 μ m	Most of them are scattered in country rock. Includes gangue minerals of less than 20 μ m
Chalcopyrite	20 ~50 μ m	Most of them are scattered in country rock. Partly associated with fine grained pyrite.
Galena	20 ~100 μ m	Most of them are scattered in country rock. Partly associated with fine grained chalcocite, chalcopyrite and pyrite

Remarks of photo. : Cc : Calcocite Bor : Bornite Cp : Chalcopyrite Py : Pyrite Gal : Galena
Sp : Sphalerite Rut : Rutile Cov : Covelite G : Gangue minerals

Table 2-6-3 EPMA Analysis of Ore Minerals,
Copper Ore from the Eastern Orebody (%)

The points of	Cu	S	Fe	Pb	Ti	O	Minerals
a	80.6	19.4					Chalcocite
b					58.6	41.4	Rutile
c		13.4		86.6			Galena
d		13.6		86.4			Galena
e	80.7	19.3					Chalcocite
f	64.1	24.7	11.2				Bornite
g		13.6		86.4			Galena
h	79.9	20.1					Chalcocite
i	80.3	19.7					Chalcocite
j	80.0	20.0					Chalcocite
k	80.5	19.5					Chalcocite

which varied between 30 and 200 μ m. The chalcocite contained fine gangue minerals of 20 to 100 μ m. The particle size of bornite was 3 to 300 μ m, most of which were 20 to 100 μ m and also contained gangue minerals of less than 10 μ m particles size. Galena was observed with particle size of 1 to 300 μ m, most of which varied from 20 to 100 μ m. Galena was mostly scattered in the country rocks, but was occassionally associated with the chalcocite, chalcopyrite and pyrite. The texture of the main minerals are shown in Table 2-6-2.

6-2-2 Preliminary Ore Dressing Tests

(1) Grindability Test

Because of small amount of the sample to be necessary to measure the grindability shown in Japan Industrial Standard (JIS), simplified method was adopted to measure the grindability of ore by comparison with ore of known work index. This method is as follows;

①The value of 80% passing size of the ground ore of known work index is obtained by seiving after a given grinding time.

②The sample ore of which the work index is unknown is ground several times under the same conditions as ① but with varying grinding times. The grinding time which produces 80% passing size same as ①, is estimated.

③It is thought that the energy used for girinding is proportional to the grinding time. So, the unknown work index is estimated by multiplying the value of the known work index by the ratio of (grinding time ②)/(grinding time ①).

The known work index, standard ore (Morenci mines's ore in Arizona, USA. The work index= 12.1KWH/t) was ground for 15 minutes by a test rod mill and 80% passing size of the ground product was 80 μ m. The test sample was ground under the same condition for 15 and 20 minutes, and the grinding time for which 80% passing size was 80 μ m was estimated to be 19.1 minutes. Therefore, the estimeted work index was 15.4 KWH/t. Generally speaking, this sample ore was rather hard.

(2) Grinding Test

After 500g of the sample ore was ground by both a test rod mill and a ball mill each for 5 minutes,7.5 minutes and 10 minutes, screen analyses were done and each fractions were analysed for copper, lead and sulphur (Appendix 34). From the results, copper grade had a peak at the fraction size of 20 μ m and the lead grade increased as the size became fine.

(3) Rougher Flotation Test (Bulk Flotation)

Some bulk flotation tests were carried out by varying the grinding time in three stages and the relation between the flotation size and recoveries was obtained. The flowsheet of the bulk flotation test is shown in Fig.2-6-1. The results of the bulk rougher flotation are shown in Table 2-6-4. The relation between the flotation size and recovery of the froth is shown in Fig.2-6-2. These results demonstrate that the recoveries become higher as the flotation particle size decreases. But the recoveries of copper and lead were 92% at the size of 64% under 200 mesh (74 μ m) and the recovery did not increase for finer particle sizes. So, it was considered that the best size of rougher flotation was about 65% under 200 mesh.

(4) Flotation Speed Test

In order to get sufficient rougher flotation time, froth(1), froth(2), froth(3) and froth(4) were obtained for flotation times of 3, 3, 4 and 7 minutes respectively. The flowsheet and the test results are shown in Fig.2-6-3 and Table 2-6-6.

Both copper and lead floated with recovery 88～90% in the first 3 minutes indicating that this sample ore floated easily with rapid floatability. The recoveries of copper and lead were 92～97% in 10 minutes and extending the flotation time to 17 minutes, the amount of increased recovery was small. Therefore approximately 10 minutes of rougher flotation time was sufficient to optimise recovery.

(5) Straight Differential Flotation

In order to compare to bulk defferential flotation, lead-copper straight differential flotation tests were carried out. The flowsheet of the straight differential flotation test is shown in Fig.2-6-4. Some reagents such as sodium cyanide (NaCN), potassium ferrocyanide ($K_4[Fe(CN)_6]$), sodium hydrosulphide (NaHS) were tested as depressant reagents. The test results are shown in Table 2-6-7.

Sodium Cyanide did not depress copper, because the ore contained much chalcocite. Large amount of potassium ferrocyanide depressed copper well and increased lead recovery. However satisfactory separation of copper and lead were not obtained. Sodium hydrosulphide depressed both copper and lead, and therefore did not achieve satisfactory separation. Of these reagents, sodium ferrocyanide was the best depressant.

(6) Size Analyses of Flotation Tailing

Size analyses and chemical analyses of size fractions were carried out on the tailings of test KS-1,2,3. The results are shown in Appendix 35. The copper and lead grade of fractions +149 μ m and +105 μ m decreased as the size became finer. Therfore coarse particles must be ground, but over-grinding must be avoided.

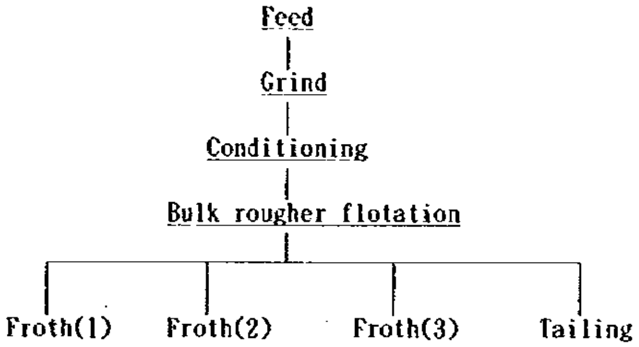


Fig. 2-6-1 Flowsheet of Bulk Rougher Flotation, Copper Ore from the Eastern Orebody

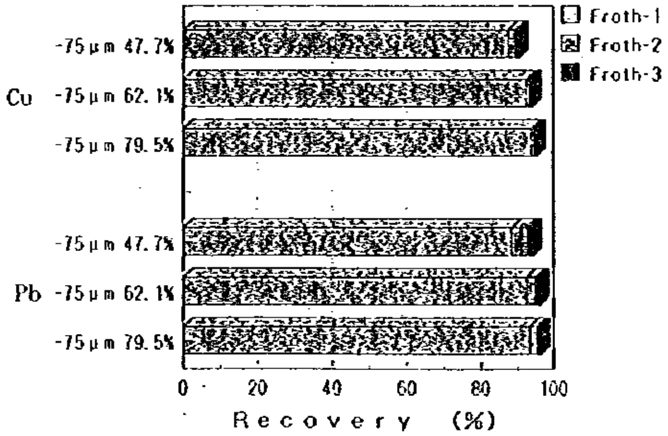


Fig. 2-6-2 Relationship between Flotation Size and Recovery of Froth, Copper Ore from the Eastern Orebody

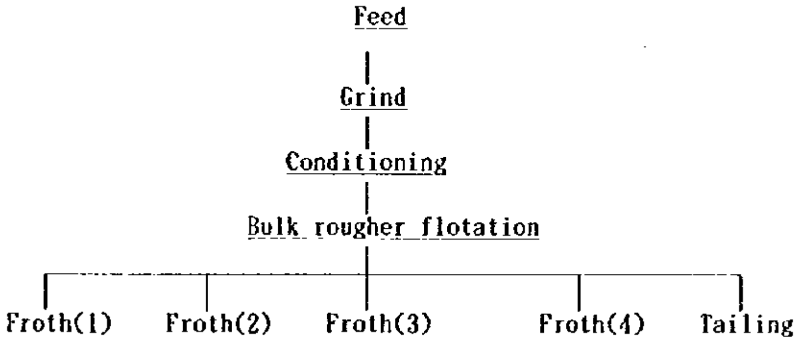


Fig. 2-6-3 Flowsheet of Flotation Speed Test, Copper Ore from the Eastern Orebody

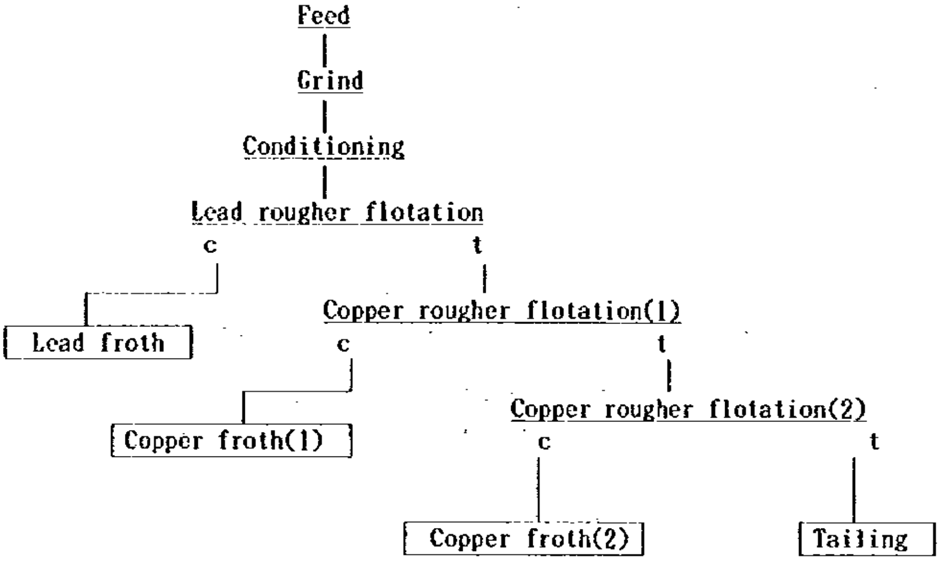


Fig. 2-6-4 Flowsheet of Straight Differential Flotation Test, Copper Ore from the Eastern Orebody

Table 2-6-4 Results of the Bulk Rougher Flotation, Copper Ore from the Eastern Orebody

Test No.	Product	Wt.%	Grade %		Recovery %		Cum. Recovery %	
			Cu	Pb	Cu	Pb	Cu	Pb
	Feed	100.00	1.69	0.48	100.00	100.00		
KS－1	Froth(1)	8.42	17.57	5.07	87.55	88.60	87.55	88.60
(-200mesh	Froth(2)	2.31	1.50	0.89	2.05	4.26	89.60	92.86
46%)	Froth(3)	2.06	0.91	0.40	1.11	1.71	90.71	94.57
	Tailing	87.21	0.18	0.03	9.29	5.43		
	Feed	100.00	1.66	0.48	100.00	100.00		
KS－2	Froth(1)	9.78	15.77	4.54	92.70	92.80	92.70	92.80
(-200mesh	Froth(2)	2.70	0.57	0.44	0.93	2.49	93.63	95.29
62.6%)	Froth(3)	2.17	0.56	0.25	0.73	1.14	94.36	96.43
	Tailing	85.35	0.11	0.02	5.64	3.57		
	Feed	100.00	1.69	0.49	100.00	100.00		
KS－3	Froth(1)	10.78	14.78	4.27	94.14	93.55	94.14	93.55
(-200mesh	Froth(2)	2.50	0.56	0.40	0.83	2.04	94.97	95.59
78%)	Froth(3)	1.90	0.46	0.25	0.52	0.96	95.49	96.55
	Tailing	84.82	0.09	0.02	4.51	3.45		

Table 2-6-5 Weight of Structural Minerals of Bulk Concentrate, Copper Ore from the Eastern Orebody

Minerals	Chalcocite	Bornite	Chalcopyrite	Pyrite	Galena	Sphalerite	Gangue
wt%	13.8	2.5	0.4	5.4	4.4	0.4	73.1

Table 2-6-6 Results of Flotation Speed Test, Copper Ore from the Eastern Orebody

Test No.	Product	Wt.%	Grade %		Recovery %		Cum. Recovery %	
			Cu	Pb	Cu	Pb	Cu	Pb
	Feed	100.00	1.70	0.49	100.00	100.00		
KS－4	Froth(1)	6.14	24.50	7.09	88.29	88.74	88.29	88.74
(-200mesh	Froth(2)	1.90	2.03	0.96	2.27	3.73	90.56	92.47
64%)	Froth(3)	2.23	1.10	0.50	1.44	2.27	92.00	94.74
	Froth(4)	2.82	0.83	0.30	1.37	1.72	93.37	96.46
	Tailing	86.91	0.13	0.02	6.63	3.54		
	Feed	100.00	1.70	0.49	100.00	100.00		
KS－5	Froth(1)	5.87	26.23	7.53	90.63	90.63	90.63	90.63
(-200mesh	Froth(2)	2.30	1.61	0.93	2.18	4.38	92.81	95.01
78%)	Froth(3)	2.34	0.90	0.42	1.24	2.02	94.05	97.03
	Froth(4)	2.22	0.63	0.26	0.82	1.18	94.87	98.21
	Tailing	87.27	0.10	0.01	5.13	1.79		

Table 2-6-7 Results of Straight Differential Flotation Test, Copper Ore from the Eastern Orebody

Test No.	Product	Wt.%	Grade %		Recovery %		Cum. Recovery %	
			Cu	Pb	Cu	Pb	Cu	Pb
	Feed	100.00	1.71	0.51	100.00	100.00		
KS－6	Lead froth	6.21	25.93	7.17	94.26	87.41	94.26	87.41
(NaCN+	Cu froth(1)	4.17	0.52	0.68	1.27	5.56	95.53	92.97
ZnSO 4)	Cu froth(2)	4.70	0.36	0.22	0.99	2.03	96.52	95.00
	Tailing	84.92	0.07	0.03	3.48	5.00		
	Feed	100.00	1.73	0.51	100.00	100.00		
KS－8	Lead froth	3.37	20.52	13.71	39.98	90.48	39.98	90.48
(K 4 [Fe	Cu froth(1)	5.13	17.63	0.26	52.25	2.61	92.23	93.09
(CN) 6)	Cu froth(2)	5.23	0.92	0.18	2.78	1.84	95.01	94.93
	Tailing	86.27	0.10	0.03	4.99	5.07		
	Feed	100.00	1.77	0.49	100.00	100.00		
KS－9	Lead froth	2.45	2.65	0.50	3.65	13.47	3.65	13.47
(NaHS)	Cu froth(1)	8.28	19.60	2.78	91.45	79.06	95.10	92.53
	Cu froth(2)	4.01	0.68	4.82	1.54	0.71	96.64	93.24
	Tailing	85.26	0.07	0.09	3.36	6.76		

6-2-3 Differential Flotation Test

(1) Bulk Differential Flotation Test (Investigation of Particle Size in Copper-Lead Separation Flotation)

In order to separate copper and lead from the bulk concentrate, the effect of the paticle size of regrinding was investigeted. The flowsheet of the bulk differential flotation test is shown in Fig.2-6-5.

The test results of lead-copper separation differential flotation which were carried out with various regrinding sizes in three stages is shown in Table 2-6-7.

With greater re-grinding, the lead recovery increased but excessive re-grinding decreasedthe lead recovery because of the fine particle size of galena. The copper recovery increa-sed as the re-grinding increased. The depression of copper in the lead concentrate was not sufficient, in spite of decreasing the copper grade in the lead concentrate by finer re-grinding. In order to get the high grade of lead concentrate, it is necessary to investigate the combination of collectors and frothers, cleaning times, pH and so on.

(2) Bulk Differential Flotation Test (Investigation of Collectors in Copper-Lead Separation Flotation)

In order to investigate collectors, four different kinds; Sodium isopropylxanthate (NaIPX), NaIPX + Aeropromoter 242 (AP242), Aeropromoter 3418a (AP3418a) and Sodium ethylxanthate (NaEX), were tested. The re-grinding size was 80% passing size 27 μ m. The test results are shown in Table 2-6-9.

The difference between the collctors was not significant, and the copper grade in the lead concentrate was lowest when NaIPX was used. From that, NaIPX was confirmed as the best collector for this ore.

The reason for the high copper grade higher than 40%Cu in Pb cleaning Tailing-1 and -2 was that chalcocite floating at the lead rougher flotation was depressed during the lead cleaning flotation. Combining the lead cleaning tailing to the copper concentrate, the grade and recovery of the copper concentrate are shown in Table 2-6-9 and Table 2-6-10.

(3) Confirmation Test (Investigation of the Best Conditions)

Based on the above observations, the best conditions to get the most desirable copper and lead concentrates were investigeted by bulk differential flotation and straight differential flotation tests.

① Bulk differential flotation test

After grinding under 200 mesh 80%, bulk flotation was carried out using NaIPX as a collector, MIBC as a frother, sodium sulfide, sodium carbonate and sodium silicate as conditioners. Flotation time was 17 minutes. The bulk concentrate was obtained by cleaning the bulk rougher froth three times.

After re-grinding of the bulk concentrate, lead cleaning was carried out 4~5 times by using potassium ferrocyanide as a depressant of copper. At the upper stage of lead cleaning, some good results were obtained by using sodium cyanide as a depressant of chalcopyrite which was not depressed by potassium ferrocyanide. The test results are shown in Table 2-6-11.

Combining the lead cleaning tailing and the copper concentrate, the grade and recovery of the copper concentrate is shown in Table 2-6-12.

In these tests, the copper concentrate of 39%Cu and 1%Pb with copper recovery 86% and the lead concentrate of 48%Pb and 11%Cu with lead recovery 67% were obtained. But high grades of lead concentrate (higher than 50%) could not be obtained.

② Straight differential flotation Test

After grinding under 200 mesh 80%, lead rougher flotation was carried out using NaIPX as collector, MIBC as a frother, potassium ferrocyanide as a depressant of copper and sodium sulfide, sodium carbonate and sodium silicate as conditioners. Flotation time was 17 minutes. After re-ginding of the lead rougher froth the lead concentrate was gained by cleaning 5~6 times. After the lead rougher flotation, copper rougher flotation was carried out and the sink became the tailing. The copper rougher froth was combined with the lead cleaning sink and then re-ground, cleaned 3 ~5 times and the copper concentrate was obtained. The flowsheet of these tests is shown in Fig.2-6-6 and an example of test results is shown in Table 2-6-13.

In these tests, a copper concentrate of 30%Cu and 3%Pb with copper recovery 85% and a lead concentrate of 59%Pb and 10% Cu with lead recovery 64% were obtained.

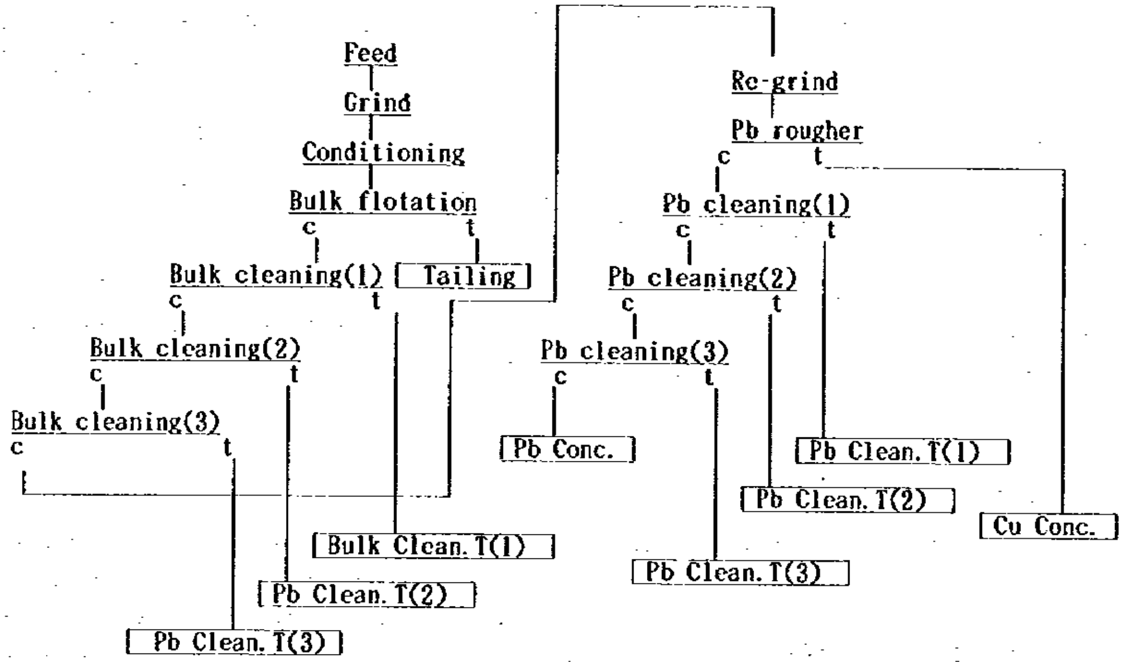


Fig. 2-6-5 Flowsheet of the Bulk Differential Flotation Test, Copper Ore from the Eastern Orebody

Table 2-6-8 Results of Bulk Differential Flotation Test (Effect of Particle Size), Copper Ore from the Eastern Orebody

Test No.	Product	Wt.%	Grade %		Recovery %		Cum. Recovery %	
			Cu	Pb	Cu	Pb	Cu	Pb
KS－10 (-200mesh 86.3%)	Feed	100.00	1.74	0.50	100.00	100.00		
	Pb Conc.	2.05	31.10	17.10	36.48	69.45	36.48	69.45
	Pb clean. T-2	0.56	49.80	1.56	15.93	1.73	52.41	71.18
	Pb clean. T-1	0.74	36.60	2.15	15.61	3.18	68.02	74.36
	Cu Conc.	1.92	22.60	3.10	24.90	11.83	92.92	86.19
	Bulk clean T-3	0.33	0.71	0.69	0.13	0.45	93.05	86.64
	Bulk clean T-2	1.46	0.52	0.52	0.44	1.51	93.49	88.15
	Bulk clean T-1	6.31	0.29	0.26	1.05	3.25	94.54	91.40
	Tailing	86.63	0.11	0.05	5.46	8.60		
KS－11 (-200mesh 93.7%)	Feed	100.00	1.70	0.49	100.00	100.00		
	Pb Conc.	1.64	28.90	20.50	27.62	68.34	27.62	27.62
	Pb clean. T-2	0.49	53.90	2.31	15.26	2.28	42.88	70.62
	Pb clean. T-1	1.05	47.90	2.38	29.40	5.09	72.28	75.71
	Cu Conc.	2.09	16.90	2.40	20.54	10.17	92.82	85.88
	Bulk clean T-3	0.33	0.71	0.69	0.13	0.45	92.95	86.33
	Bulk clean T-2	1.46	0.52	0.52	0.44	1.54	93.39	87.87
	Bulk clean T-1	6.31	0.29	0.26	1.06	3.33	94.45	91.20
	Tailing	86.63	0.11	0.05	5.55	8.80		
KS－12 (-200mesh 95.1%)	Feed	100.00	1.70	0.49	100.00	100.00		
	Pb Conc.	1.21	27.30	22.00	19.41	54.70	19.41	54.70
	Pb clean. T-2	0.50	52.40	4.87	15.52	5.04	34.93	59.74
	Pb clean. T-1	1.07	45.70	4.32	28.70	9.49	63.63	69.23
	Cu Conc.	2.49	19.80	3.20	29.10	16.44	92.73	85.67
	Bulk clean T-3	0.33	0.71	0.69	0.13	0.46	92.86	86.13
	Bulk clean T-2	1.46	0.52	0.52	0.45	1.57	93.31	87.70
	Bulk clean T-1	6.31	0.29	0.26	1.08	3.38	94.39	91.08
	Tailing	86.63	0.11	0.05	5.61	8.92		

Table 2-6-9 Results of Bulk Differential Flotation Test (Effect of Collectors), Copper Ore from the Eastern Orebody

Test No.	Product	Wt %	Grade %		Recovery %		Cum. Recovery %	
			Cu	Pb	Cu	Pb	Cu	Pb
	Feed	100.00	1.67	0.50	100.00	100.00		
KS－13	Pb Conc.	0.82	13.10	44.10	6.43	73.04	6.43	73.04
(NaIPX+	Pb clean. T-2	0.46	50.90	3.41	14.02	3.17	20.45	76.21
AP242)	Pb clean. T-1	1.12	52.40	1.63	34.13	3.69	55.58	79.90
	Cu Conc.	2.63	24.10	0.90	37.80	4.76	93.38	84.66
	Bulk clean T-3	0.55	0.83	0.85	0.27	0.93	93.65	85.59
	Bulk clean T-2	1.64	0.50	0.55	0.49	1.82	94.14	87.41
	Bulk clean T-1	7.29	0.29	0.27	1.26	3.97	95.40	91.38
	Tailing	85.49	0.09	0.05	4.60	8.62		
	Feed	100.00	1.68	0.47	100.00	100.00		
KS－14	Pb Conc.	0.78	9.17	43.30	4.29	71.83	4.29	71.83
(NaIPX)	Pb clean. T-2	0.42	43.10	4.18	10.87	3.73	15.16	75.56
	Pb clean. T-1	1.15	44.30	1.71	30.32	4.15	45.48	79.71
	Cu Conc.	2.68	30.00	0.74	47.91	4.19	93.39	83.90
	Bulk clean T-3	0.55	0.83	0.85	0.27	0.98	93.66	84.88
	Bulk clean T-2	1.64	0.50	0.55	0.49	1.92	94.15	86.80
	Bulk clean T-1	7.29	0.29	0.27	1.26	4.16	95.41	90.96
	Tailing	85.49	0.09	0.05	4.59	9.04		
	Feed	100.00	1.70	0.48	100.00	100.00		
KS－15	Pb Conc.	0.94	13.70	38.00	7.63	73.83	7.63	73.83
(AP3418a)	Pb clean. T-2	0.49	49.60	2.79	14.41	2.83	22.04	76.66
	Pb clean. T-1	1.38	45.00	1.46	37.06	4.19	59.10	80.85
	Cu Conc.	2.22	26.00	0.73	34.31	3.36	93.41	84.21
	Bulk clean T-3	0.55	0.83	0.85	0.27	0.97	93.68	85.18
	Bulk clean T-2	1.64	0.50	0.55	0.49	1.88	94.17	87.06
	Bulk clean T-1	7.29	0.29	0.27	1.26	4.08	95.43	91.14
	Tailing	85.49	0.09	0.05	4.57	8.86		
	Feed	100.00	1.69	0.49	100.00	100.00		
KS－16	Pb Conc.	0.75	11.10	43.00	4.95	68.14	4.95	68.14
(NaEX)	Pb clean. T-2	0.35	40.80	5.77	8.59	4.32	13.54	72.46
86.3%)	Pb clean. T-1	1.13	42.50	2.77	28.43	6.59	41.97	79.05
	Cu Conc.	2.80	31.00	0.83	51.45	4.90	93.42	83.95
	Bulk clean T-3	0.55	0.83	0.85	0.27	0.97	93.69	84.92
	Bulk clean T-2	1.64	0.50	0.55	0.49	1.91	94.18	86.83
	Bulk clean T-1	7.29	0.29	0.27	1.25	4.15	95.43	90.98
	Tailing	85.49	0.09	0.05	4.57	9.02		

Table 2-6-10 Combined Copper Concentrate, Copper Ore from the Eastern Orebody

Test No.	Wt %	Grade %		Recovery %	
		Cu	Pb	Cu	Pb
KS－13	4.21	36.69	1.37	86.95	11.62
KS－14	4.25	35.16	1.34	89.10	12.07
KS－15	4.09	35.24	1.22	85.78	10.38
KS－16	4.28	36.01	1.75	88.47	15.81

Table 2-6-11 Results of the Bulk Differential Flotation Test, Copper Ore from the Eastern Orebody

Test No.	Product	Wt.%	Grade %		Recovery %		Cum. Recovery %	
			Cu	Pb	Cu	Pb	Cu	Pb
	Feed	100.00	1.70	0.50	100.00	100.00		
KS-17	Pb concentrate	0.71	9.95	46.80	4.17	66.37	4.17	66.37
	Pb clean. T-4	0.20	7.82	11.90	0.04	4.82	4.21	71.19
	Pb clean. T-3	0.39	47.90	4.95	10.89	3.81	15.10	75.00
	Pb clean. T-2	0.71	59.50	1.39	24.92	1.97	40.02	76.97
	Pb clean. T-1	1.10	44.10	1.88	28.49	4.11	68.51	81.08
	Cu concentrate	1.92	22.10	0.76	24.99	2.91	93.50	83.99
	Bulk clean.T-3	0.80	1.26	1.02	0.60	1.63	94.10	85.62
	Bulk clean.T-2	2.05	0.52	0.50	0.63	2.05	94.73	87.67
	Bulk clean.T-1	7.85	0.28	0.25	1.30	3.92	96.03	91.59
	Tailing	84.27	0.08	0.05	3.97	8.41		
	Feed	100.00	1.69	0.49	100.00	100.00		
KS-18	Pb concentrate	0.68	11.10	48.70	4.46	67.03	4.46	67.03
	Pb clean. T-4	0.10	6.14	18.10	0.02	3.72	4.48	70.75
	Pb clean. T-3	0.16	4.50	10.10	0.43	3.32	4.91	74.07
	Pb clean. T-2	0.29	3.05	3.02	0.53	1.80	5.44	75.87
	Pb clean. T-1	0.31	41.20	2.21	7.65	1.41	13.09	77.28
	Cu concentrate	1.79	52.10	1.25	55.25	4.55	68.34	81.83
	Bulk clean.T-3	1.62	24.60	0.70	23.58	2.30	91.92	84.13
	Bulk clean.T-2	1.24	1.26	0.94	0.92	2.36	92.84	86.49
	Bulk clean.T-1	2.57	0.39	0.36	0.59	1.88	93.43	88.37
	Bulk clean.T	9.77	0.22	0.17	1.27	3.37	94.70	91.74
	Tailing	81.47	0.11	0.05	5.30	8.26		

Table 2-6-12 Combined Copper Concentrate, Copper Ore from the Eastern Orebody

Test No.	Wt %	Grade %		Recovery %	
		Cu	Pb	Cu	Pb
KS－17	3.73	35.71	1.21	78.40	8.99
KS－18	3.72	39.23	1.09	86.48	8.26

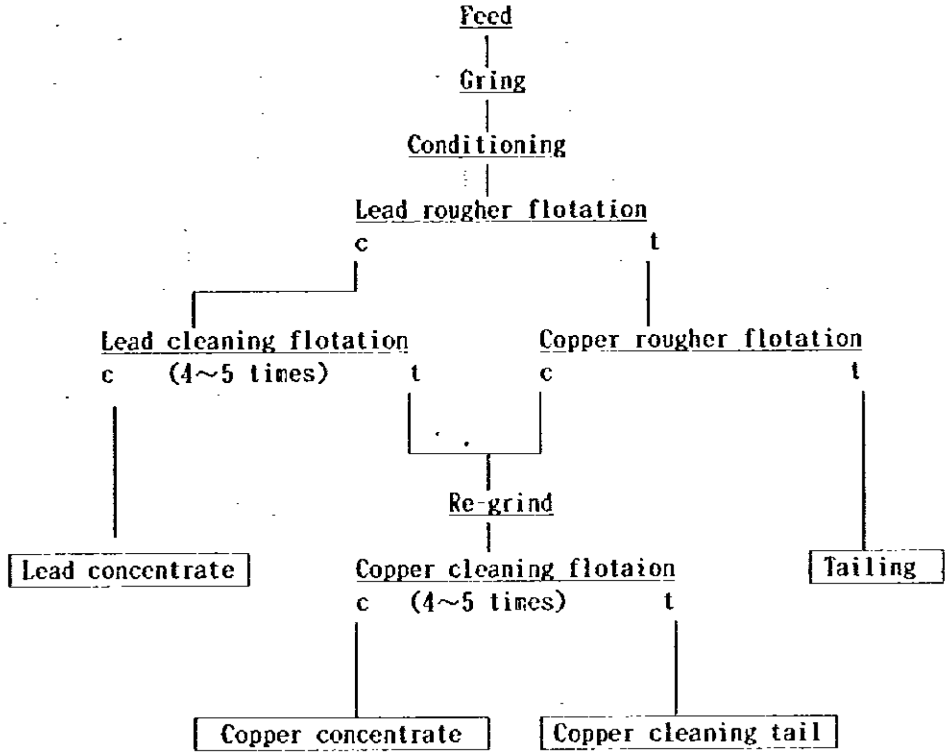


Fig. 2-6-6 Flowsheet of Straight Differential Flotation, Copper Ore from the Eastern Orebody

Table 2-6-13 Results of Straight Differential Flotation Test, Copper Ore from the Eastern Orebody

Test No.	Product	Wt.%	Grade %		Recovery %		Cum. Recovery %	
			Cu	Pb	Cu	Pb	Cu	Pb
	Feed	100.00	1.63	0.48	100.00	100.00		
KS－22	Pb concentrate	0.52	9.65	59.90	3.05	64.20		
	Cu concentrate	4.63	30.20	2.96	85.63	28.48	85.63	28.48
	Cu clean. T-5	0.41	2.79	0.61	0.71	0.53	86.34	29.01
	Cu clean. T-4	0.78	2.23	0.37	1.07	0.60	87.41	29.61
	Cu clean. T-3	1.75	1.39	0.19	1.49	0.69	88.90	30.30
	Cu clean. T-2	4.98	0.98	0.12	3.00	1.25	91.90	31.55
	Cu clean. T-1	7.61	0.25	0.06	1.17	0.95	93.07	32.50
	Tailing	79.32	0.08	0.02	3.88	3.30		

6-2-4 Determination of the Flotation Flow Process

A high grade of lead concentrate (higher than 50%Pb) was not obtained by the bulk differential flotation, however a high grade of copper concentrate (higher than 35%Cu) was gained with high copper recovery. By the straight differential flotation method, a high grade of lead concentrate (higher than 50%) could be obtained but the copper grade and copper recovery of the copper concentrate was lower than the bulk differential flotation. Therefore, considering the lead concentrate, straight differential flotation is better, but considering the copper concentrate, bulk differential flotation is better. The copper grade of ore is three times greater than that of lead and the metal price of copper is far high than that of lead. So, the bulk differential flotation process is economically superior.

From the above result, the most suitable flotation flowsheet was decided as Fig.2-6-7.

6-2-5 Chemical Analyses of Copper and Lead Concentrates

The chemical analyses of the copper and lead concentrates are as follows;

Table 2-6-14 Chemical Analysis of Copper and Lead Concentrate,
Copper Ore from the Eastern Orebody

Element	Cu	Pb	S	Zn	Fe	Sb	As	Bi
Cu conc.	34.0	1.79	16.5	0.18	7.73	<0.05	<0.05	<0.05
Pb conc.	9.65	59.9	14.6	2.13	1.40	<0.05	<0.05	<0.05

Element	Cd	Hg	Au	Ag	Cl	F	Re	SiO_2
Cu conc.	0.01	0.3	<0.1	131	0.02	0.02	<0.05	26.1
Pb conc.	0.23	1.2	0.3	557	0.01	<0.01	<0.05	3.42

Element	Al_2O_3	MgO	CaO	K_2O	Na_2O
Cu conc.	4.90	0.45	1.89	1.12	1.48
Pb conc.	0.74	0.06	0.33	0.15	0.20

Au, Ag; g/t Hg; ppm Others; %

6-2-6 Comment and Recommendation

In these flotation tests, some good results were obtained by bulk and straight differential flotation. But copper depression of this ore was rather difficult, so it is necessary to investigate the floating of copper from the bulk froth and depressing of lead.

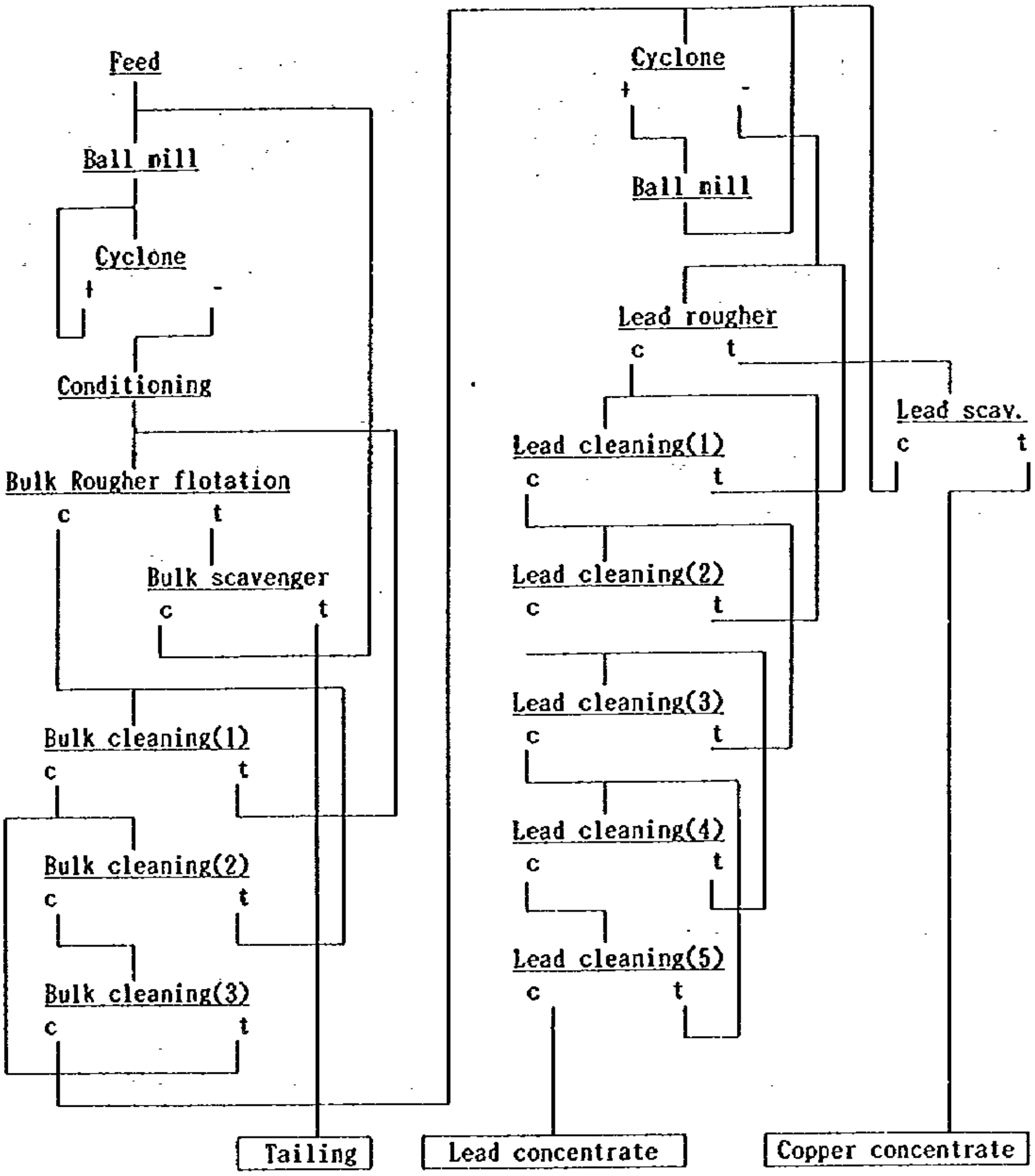


Fig. 2-6-7 Optimum Flowsheet, Copper Ore from the Eastern Orebody

6-3 Test of the Complex Ore

6-3-1 Composite Samples and Chemical Analyses of ore

About 80kg of ore was used to prepare composite samples for metallurgical tests. This material was composed of MJK-2 core (8.00m length, ore horizon 4-1) and ore samples which had been obtained by the Kazakhstan team. The composite samples after preparation were 1.70%Cu, 1.11%Pb, 0.03%Zn, 2.21%Fe, 1.00%S, <0.1g/tAu, and 5g/tAg. Full analytical results are shown in Table 2-6-15.

Ore samples were crushed to -6mesh (3.35mm) by a jaw crusher and crushing roller, and after preparing the designed grade, they were divided into 500g subsamples, placed in small plastic bags and kept in a refrigerator.

Table 2-6-15 **Chemical Analysis of Test Sample, Complex Ore**

Element	Cu	Pb	Zn	S	Fe	SiO2	Al2O3	MgO	CaO	K2O
%	1.70	1.11	0.03	1.00	2.21	62.4	11.6	1.29	5.09	1.51
Element	Na2O	Cl	F	Mo	Te	Sb	As	Bi	Cd	
%	2.88	0.18	0.03	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	
Element	Hg	Re	Au	Ag						
	0.1ppm	3g/t	<0.1g/t	5g/t						

6-3-2 Physical Properties of Ore Samples

(1) Result of X-ray Diffractometer Analyses of Ore

As the results of X-ray diffractometer analyses, quartz, albite, anorthite, kaolinite, muscovite and lithidionite were detected as the constituent minerals of the host rock (Appendix 36).

(2) Microscopic Observation of polished Ore samples

The main ore mineral observed under a reflection microscope was chalcocite with small amounts of bornite, galena. Minor amounts of chalcopyrite, pyrite were also present. The size of mineral particles was generally fine, 20~200 μ m. The particle size of chalcocite generally ranged between 1 and 500 μ m, most of which varied between 30 and 200 μ m. The chalcocite contained fine gangue minerals of 3 to 20 μ m. The particle size of bornite was 3 to 300 μ m, most of which were 20 to 100 μ m and also contained gangue minerals of less than 10 μ m particles size. Galena was observed with particle size of 1 to 300 μ m, most of which varied from 20 to 100 μ m. Galena was mostly scattered in the country rocks, but was occassionally associated with the chalcocite, chalcopyrite and pyrite. The texture of the main minerals are shown in Table 2-6-16.

Table 2-6-16 Texture of Test Sample, Complex Ore

Ore Mineral	Size	Structures
Chalcocite	30-200 μm	Most of them are scattered in country rock. Includes gangue minerals of less than 20 μm
Bornite	20-100 μm	Most of them are scattered in country rock. Includes gangue minerals of less than 20 μm
Chalcopyrite	20-50 μm	Most of them are scattered in country rock.
Galena	20-100 μm	Most of them are scattered in country rock. Partially associated with chalcocite and bornite, and closely associated with sphalerite

(3) Grindability Work Index Measurement

Grindability testing of complex ore was carried out by JIS (Japanese Industrial Standard). After testing, the results were obtained through the procedure for the calculation of the work index of grindability as follow.

F : 80% passing particle size of the feed ore sample (μ m) =2284 μ m
P1 : screen aperture used for grindability test (μ m) =149 μ m
P : 80% passing particle size of material passing size P1 (μ m)=108 μ m
Gbp : Mass of material passing screen P1 per 1 revolution of testing ball mill (g) =.714g
Wi : work index (kwh/t)

Calculation formula

$$\mathrm{Wi} = \frac{44.5}{\mathrm{P1}^{0.23} \times \mathrm{Gbp}^{0.82} \times \left(\frac{10}{\sqrt{\mathrm{P}}} - \frac{10}{\sqrt{\mathrm{F}}}\right)} \times 1.10$$

$$= \frac{44.5 \times 1.1}{3.16 \times 1.55\ (0.96 - 0.209)} = \underline{\underline{13.2\,\mathrm{kwh/t}}}$$

This work index is useful for rhe design and control of grinding operations. It is considered that 13.2kwh/t is the work index of ordinary copper ore (Appendix 37-1).

6-3-3 Preliminary Ore Dressing Test

(1) Grinding Test

500g of the sample ore was ground by both a rod mill and ball mill each for 8, 14, 19 and 23 minutes, and then, screen analysis was carried out through standard test sieves. Size distributions for each grinding time are shown in Fig.2-6-8.

Each size fraction was chemically analysed for copper, lead, zinc, iron and sulphur. From the results, copper, lead and zinc grade had a peak at the size fraction of around 20-45 μ m and the lead grade tended to increase with finer size (Appendix 37-2).

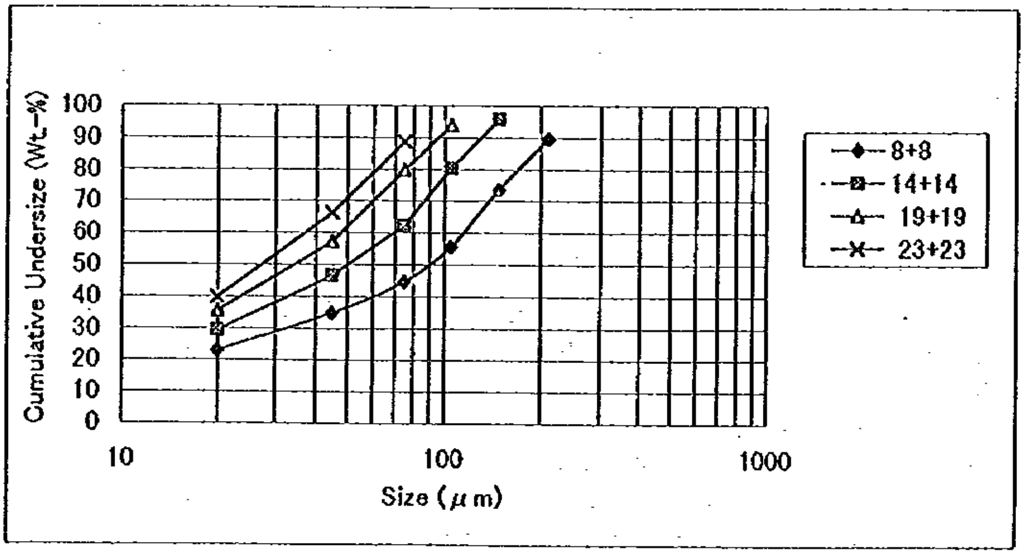


Fig.2-6-8 Relationship between Grinding Time and Size Distribution, Complex Ore

(2) Bulk Rougher Flotation Test

A rougher flotation test was carried out with grinding times for both rod mill and ball mill being 19 minutes respectively. Flotation time was 17 minutes. As a result of this test, total recoveries of copper, lead and zinc are expected to be about 98%, 95% and 75% respectively by rougher flotation. The test flowsheet is shown in Fig.2-6-9 and flotation results are shown in Table 2-6-17. The mineral constituents of this concentrate were obtained by microscopic analyses and are shown in Table 2-6-18. Chalcocite and bornite are the main copper minerals and galena is the main lead mineral.

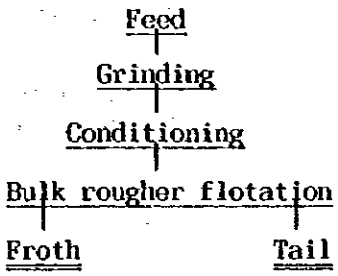


Fig.2-6-9 Flowsheet of the Bulk Rougher Flotation Complex Ore

Table 2-6-17 Results of the Bulk Rougher Flotation Complex Ore

Product	Wt%	Grade%				Distribution %			
		Cu	Pb	Zn	S	Cu	Pb	Zn	S
Feed	100.00	1.70	1.15	0.03	0.93	100.00	100.00	100.00	100.00
Froth	17.37	9.59	6.32	0.15	4.44	98.05	95.68	75.93	83.09
Tail	82.63	0.04	0.06	0.01	0.19	1.95	4.32	24.07	16.91

Table 2-6-18 Minerals in the Bulk Concentrate, Complex Ore

Mineral	Cc	Bor	Cp	Py	Gal	Sp	G
Weight %	7.8	5.3	0.7	1.2	7.2	0.2	77.6
Cu-Mineral	56.5	38.4	5.1				

Cc:Chalcocite Bor:Bornite Cp:Chalcopyrite
Py:Pyrite Gal:Galena Sp:Sphalerite
G: Gangue Mineral

(3) Bulk Rougher Flotation in Different Grinding Sizes

Bulk rougher flotation tests were carried out by varying the grinding sizes(time) in four levels in order to obtain the relationship between the grinding size and recovery of copper, lead and zinc. The flowsheet of the bulk flotation tests is shown in Fig.2-6-10. The results of the bulk rougher flotation with varying grinding sizes are shown in Fig.2-6-11. Grinding sizes were -75 μ m 44.67%, 62.18%, 79.83% and 88.71% respectively.

Also, the relationship between grinding size, rougher concentrate grades and recoveries of copper, lead and zinc are shown in Fig.2-6-12, 2-6-13, and 2-6-14.

From these results, it was shown that at finer grinding sizes, higher recoveries are obtained, though the differences became small at more than 65% -75 μ m. At more than 80% -75 μ m, recovery of lead decreased.

In the case of grinding sizes coarser than 45% -75 μ m, copper and lead recoveries decreased inversely with the higher concentrate grade. With regard to zinc, the effects of grinding size were not clearly evident due to low ore sample grades. From these tests, grinding size is preferably in the order of 65%-75 μ m.

The relationship between grinding size, copper recovery and flotation time is shown in Fig.2-6-15 and the relationship between grinding size, tailing grade and flotation time is shown in Fig.2-6-16. Except for the case of 45%-75 μ m, more than 95% recovery was achieved in 3 minutes of flotation time, and extended time has little effect on the recovery. As a result, the flotation speed is rather high, so 15 minutes of bulk rougher flotation time should be sufficient to optimise recovery.

The copper, lead and zinc recoveries in each size fraction of the flotation feed are shown in Fig.2-6-17, 2-6-18. From this result, it was indicated that 90% of such minerals are recovered if they are ground finer than -75 μ m (Appendix 37-4).

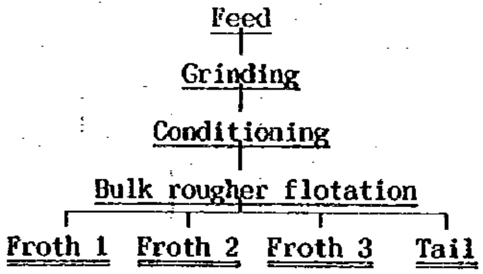


Fig.2-6-10 Flowsheet of the Bulk Rougher Flotation Complex Ore(Effect of Particle Size)

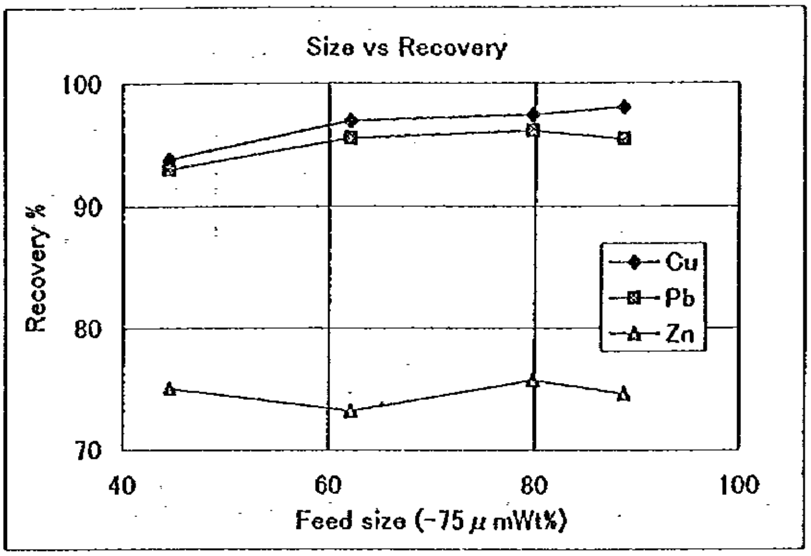


Fig.2-6-11 Relationship between Grinding Size and Recovery(Cu,Pb and Zn), Complex Ore

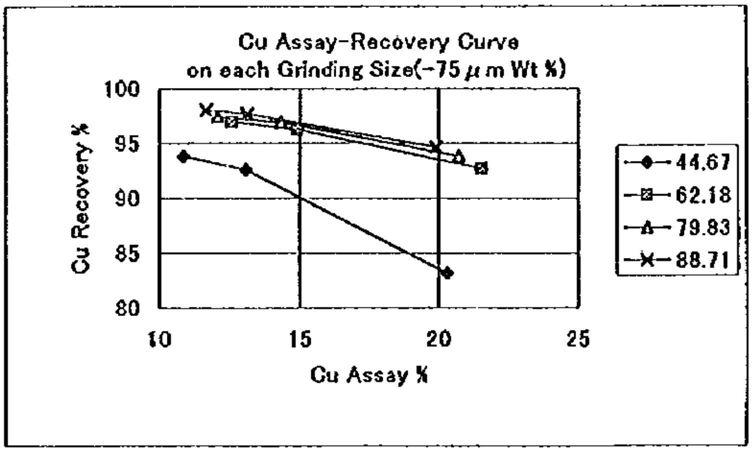


Fig.2-6-12 Relationship between Grinding Size, Cu-Recovery and Grade of Froth, Complex Ore

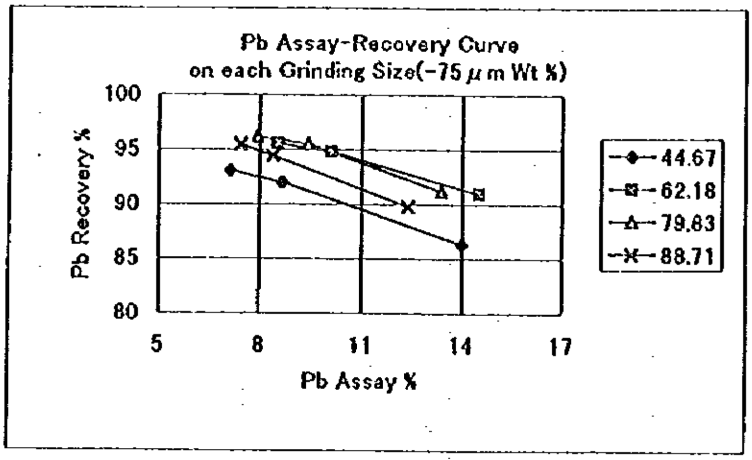


Fig.2-6-13 Relationship between Grinding Size,
Pb-Recovery and Grade of Froth, Complex Ore

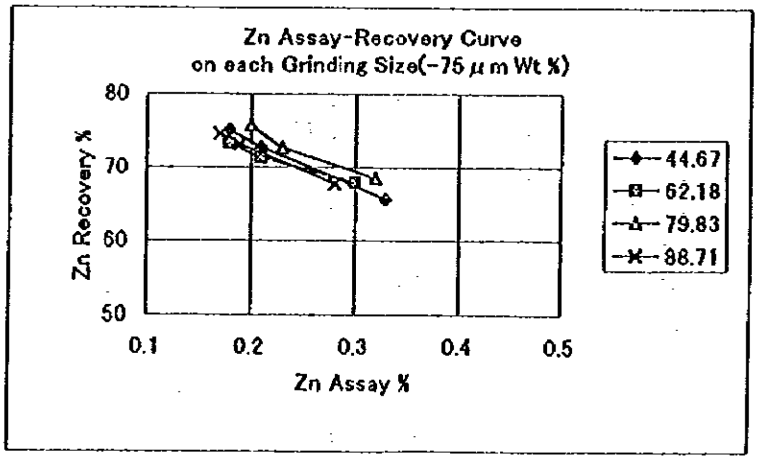


Fig.2-6-14 Relationship between Grinding Size,
Zn-Recovery and Grade of Froth, Complex Ore

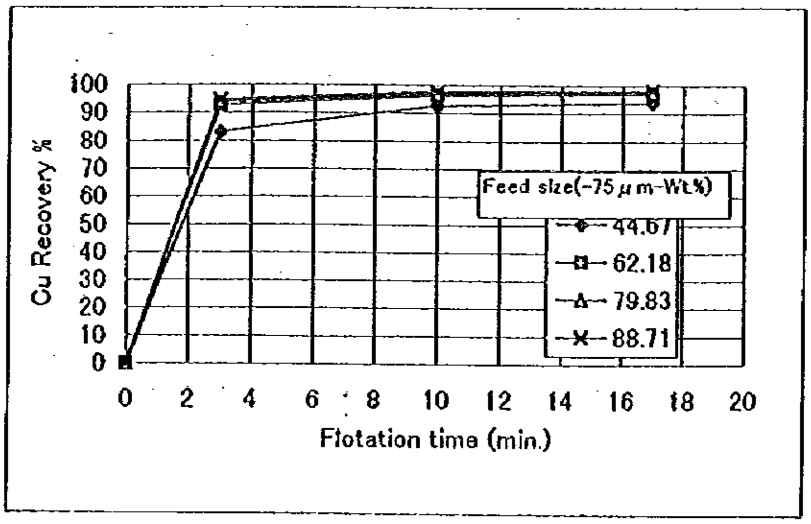


Fig.2-6-15 Relationship between Grinding Size, Cu-Recovery and Flotation Time, Complex Ore

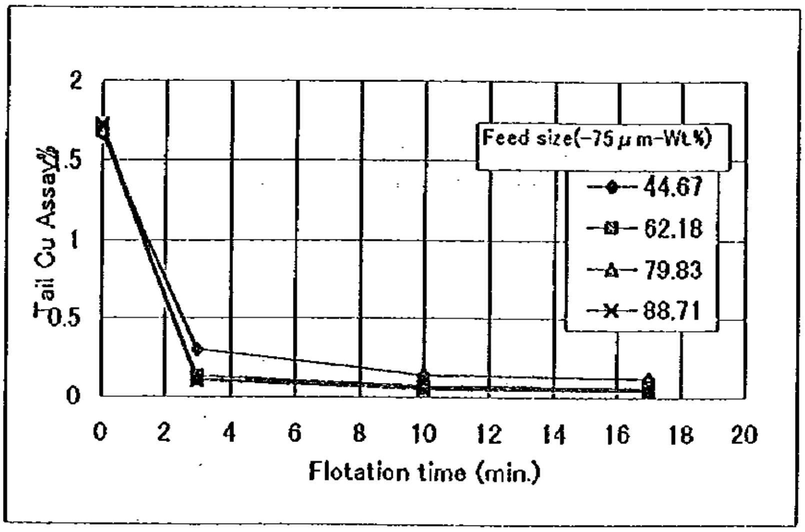


Fig.2-6-16 Relationship between Grinding Size, Cu-Grade in Tail and Flotation time, Complex Ore

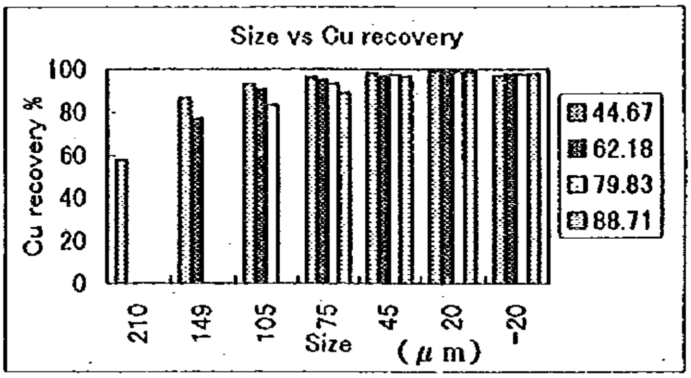


Fig.2-6-17 Relationship of the Copper Recovery in each Size Fraction and Size Distribution

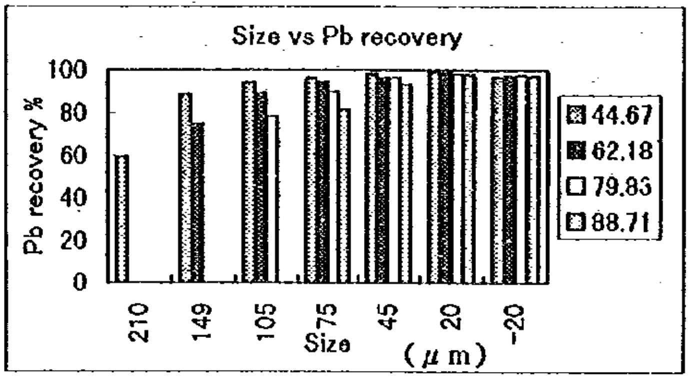


Fig.2-6-18 Relationship of the Lead Recovery in each Size Fraction and Size Distribution

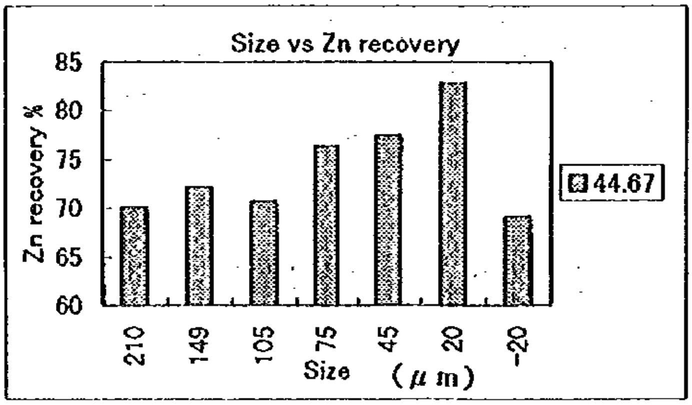


Fig.2-6-19 Relationship of the Zinc Recovery in each Size Fraction and Size Distribution

(4) Selection of Collector

Sodium isopropyl xanthate (NaIPX), M1661, a mixture of AP3418 and AP5415 and AP242 were tested as collectors in bulk rougher flotation.

The test flowsheet is shown in Fig.2-6-20 and its results are shown in Fig.2-6-21 and 2-6-22. It was found that for similar grade of concentrate, NaIPX showed the highest recovery (Appendix 37-5).

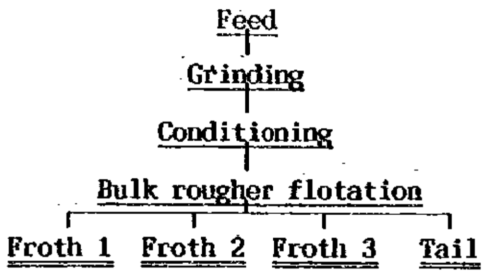


Fig.2-6-20 Flowsheet of the Bulk Rougher Flotation, Complex Ore(Effect of Collector)

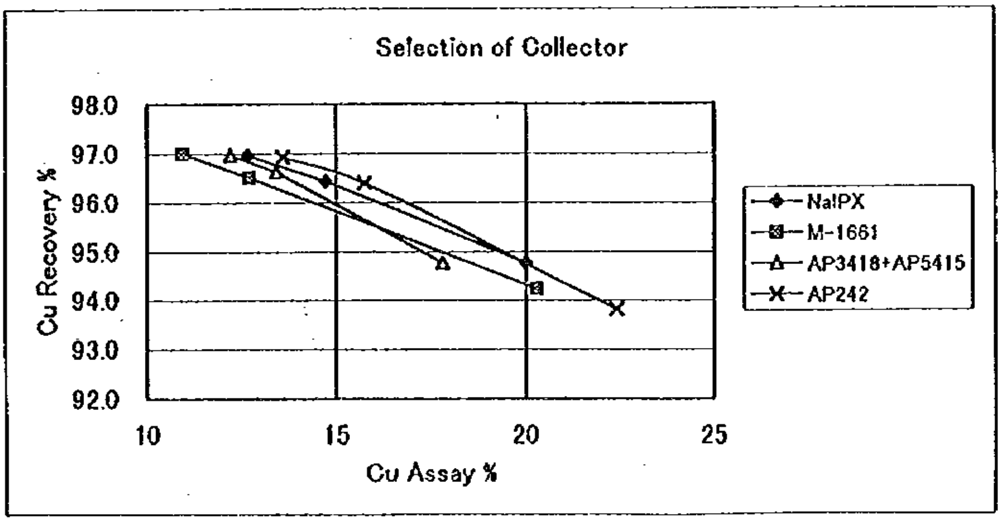


Fig.2-6-21 Relationship Between Collectors, Cu-Recovery and Grade of Froth, Complex Ore

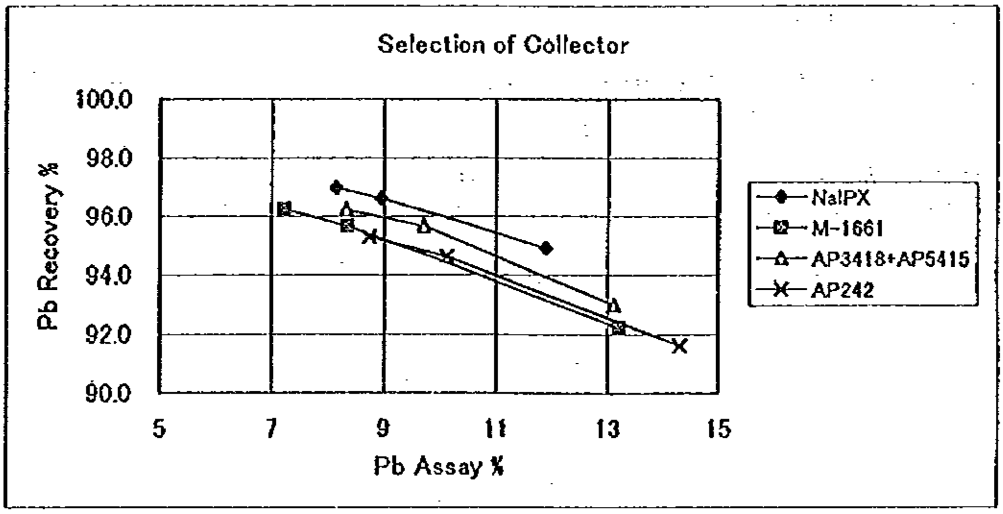


Fig.2-6-22 Relationship between Collectors,Pb-Recovery and Grade of Froth,Complex Ore

(5) Straight-Differential Flotation

When the copper and lead are separated from the complex ore, generally straight-differential flotation and the bulk-differential flotation are applicable. Here straight-differential flotation test results are presented. The test flowsheet is shown in Fig.2-6-23. Its flotation result is shown in Table 2-6-19.

Lead concentrate of 58.0% lead and copper concentrate of 38.1% copper were obtained at the recoveries of 67.4% and 78.1% respectively. Zinc was found in both concentrates with 41% being distributed in the lead concentrate (Appendix 37-6).

Table 2-6-19 Result of the Straight Differential Flotation

Product	Weight	Assay(%)			Distribution(%)		
	(%)	Cu	Pb	Zn	Cu	Pb	Zn
Feed	100.00	1.48	1.06	0.03	100.00	100.00	100.00
Pb-Conc	1.24	11.8	58.00	1.08	9.87	67.42	41.20
Cu-Conc	5.16	23.13	4.22	0.14	80.73	21.42	22.54
Tail	93.6	0.15	0.13	0.01	9.40	11.16	36.26

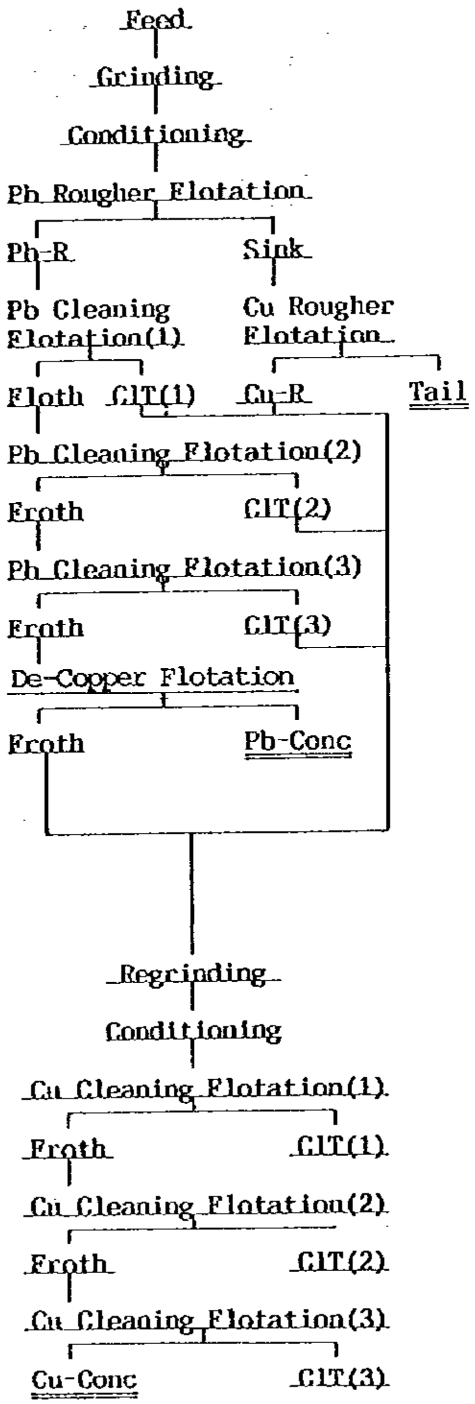


Fig.2-6-23 Flowsheet of the Straight-Differential Flotation, Complex Ore

(6) Semi-Bulk Flotation

After bulk flotation, the bulk concentrates were cleaned 3 times with and without regrinding for 3 minutes. The flowsheet is shown in Fig.2-6-24 and Fig.2-6-25, size distribution is shown in Fig.2-6-26 and flotation results are shown in Fig.6-27, 28 and 29. In the case of no grinding, the cleaning concentrate had 23.8% copper grade with 95.49% recovery and 15.9% lead grade with 94.94% recovery. After regrinding, the concentrate had 30.2% copper grade with 95.49% recovery and 19.0% lead grade with 94.17% recovery. It was found that the effect of regrinding improved concentrate grades (Appendix 37-7).

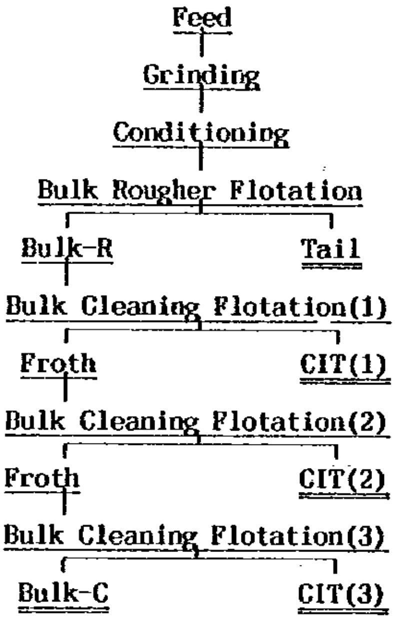


Fig.2-6-24
Flowsheet of the Semi-Bulk Flotation without Regrinding, Complex Ore

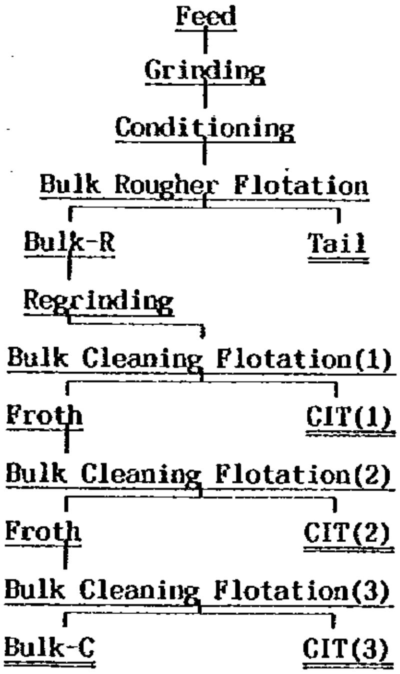


Fig.2-6-25
Flowsheet of the Semi-Bulk Flotation with Regrinding, Complex Ore

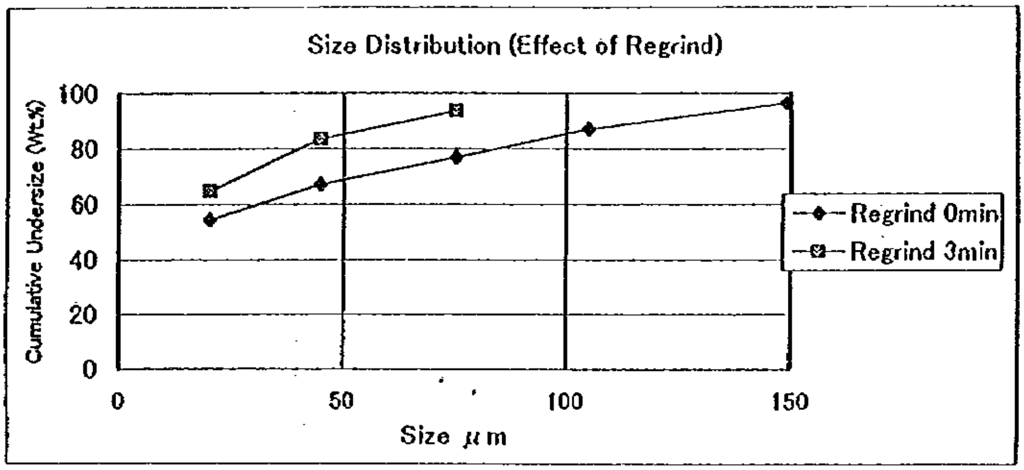


Fig.2-6-26 Size Distribution before and after Regrinding of the Rougher Froth, Complex Ore

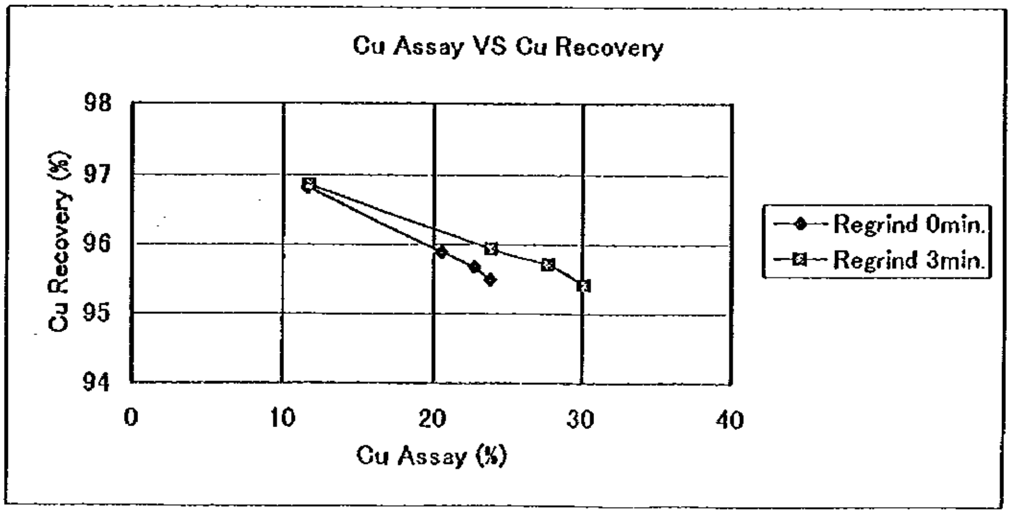


Fig.2-6-27 Relationship between Cu Recovery, Grade and Regrinding, Complex Ore

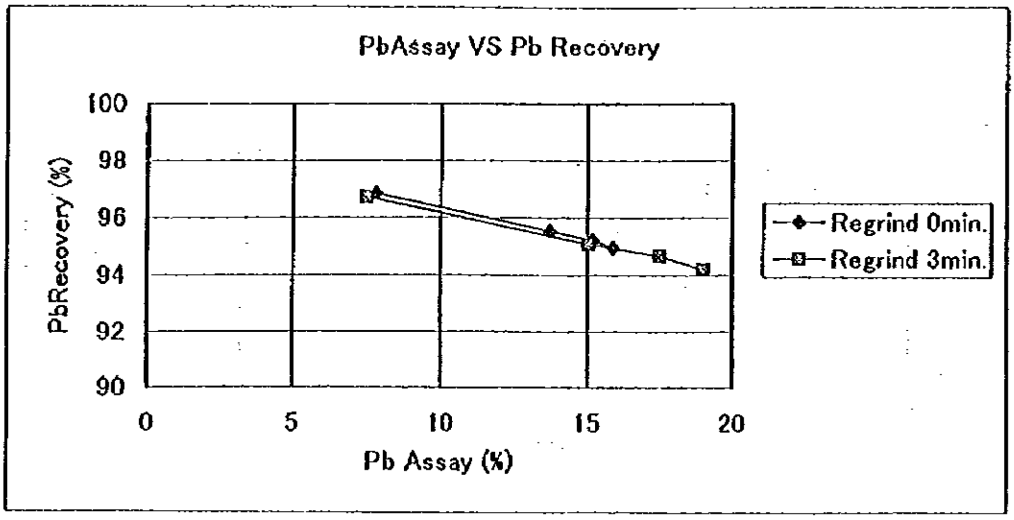


Fig.2-6-28 Relationship between Pb Recovery,Grade and Regrinding, Complex Ore

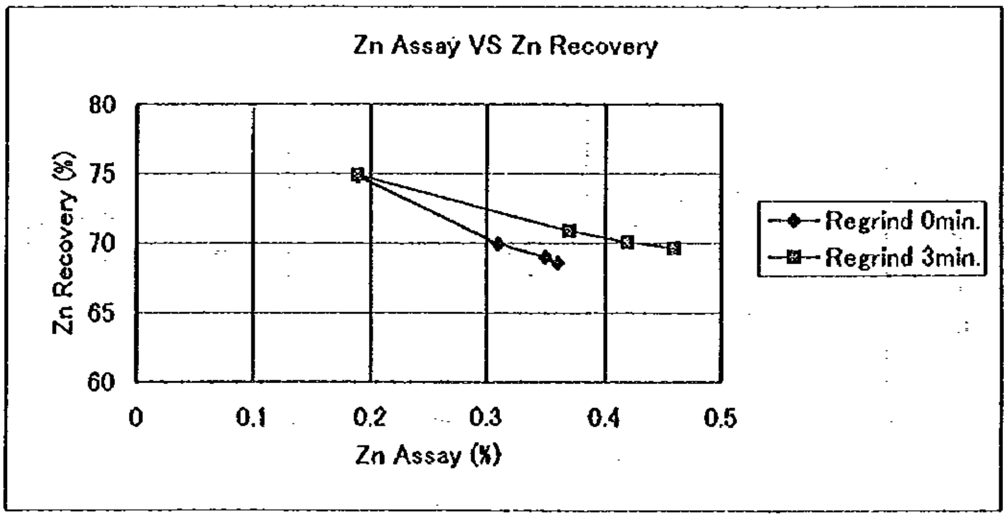


Fig.2-6-29 Relationship between Zn Recovery,Grade and Regrinding, Complex Ore

(7) Size Analyses of Bulk Flotation Tailings

Size analyses and chemical analyses of size fractions were carried out on bulk flotation test tailings. Test results are shown in Table 2-6-20.

In the case of flotation feed size of -75 μ m 79.83%, after screen analysis the maximum grades, 0.09% copper and 0.06% lead were found in the largest size fraction, +105 μ m, and about 40-50% of total copper and lead in the tailings remained in the size fraction of -20 μ m.

In the case of flotation size of -75 μ m 44.67%, the screening and analysis of the tailings showed that the copper grade of size fraction +149 μ m was 0.30% copperand lead was 0.15% and distribution of copper and lead were 61% and 53% respectively. This indicate that the grinding time was insufficient.

Table 2-6-20 Size Distribution of the Bulk Flotation Tailing Complex Ore

Size Fraction	Weight(%)	Grade(%)			Distribution(%)		
		Cu	Pb	Zn	Cu	Pb	Zn
a) -75 μm 42.98%(Flotation feed size 44.57%)							
Feed(μm)	100.00	0.13	0.07	0.01	100.00	100.00	100.00
1 +210	10.89	0.49	0.25	0.01	41.34	36.91	9.01
2 -210 +149	15.48	0.17	0.08	0.01	20.39	16.80	12.82
3 -149 +105	18.49	0.10	0.05	0.01	14.33	12.54	15.30
4 -105 +75	12.16	0.07	0.04	0.01	6.59	6.59	10.06
5 -75 +45	10.32	0.05	0.03	0.01	4.00	4.20	8.54
6 -45 +20	11.87	0.04	0.02	0.01	3.68	3.22	9.87
7 -20	20.79	0.06	0.07	0.02	9.67	19.74	34.44
1+2	26.37	0.30	0.15	0.01	61.73	53.71	21.83
1+2+3	44.86	0.22	0.11	0.01	76.06	66.25	37.13
1+2+3+4	57.02	0.19	0.09	0.01	82.65	72.64	47.19
1+2+3+4+5	67.34	0.17	0.08	0.01	86.65	77.04	55.73
1+2+3+4+5+6	79.21	0.15	0.07	0.01	90.33	80.26	65.56
6+7	32.66	0.05	0.05	0.02	13.35	22.96	44.27
5+6+7	42.98	0.05	0.05	0.01	17.35	27.16	52.81
4+5+6+7	55.14	0.06	0.05	0.01	23.94	33.75	62.87
3+4+5+6+7	73.63	0.07	0.05	0.01	38.27	46.29	78.17
2+3+4+5+6+7	89.11	0.08	0.05	0.01	58.66	63.09	90.99
b) -75 μm 78.29%(Flotation feed size 79.83%)							
Feed(μm)	100.00	0.05	0.04		100.00	100.00	
1 +105	6.43	0.09	0.06	<0.01	11.19	9.70	
2 -105 +75	15.28	0.06	0.04	0.01	17.72	15.36	
3 -75 +45	23.01	0.05	0.03	<0.01	22.24	17.35	
4 -45 +20	23.62	0.04	0.03	0.01	18.26	17.81	
5 -20	31.66	0.05	0.05	0.02	30.59	39.78	
1+2	21.71	0.07	0.05		28.92	25.06	
1+2+3	44.72	0.06	0.04		51.15	42.41	
1+2+3+4	68.34	0.05	0.04		69.41	60.22	
4+5	55.28	0.05	0.04	0.02	48.85	57.59	
3+4+5	78.29	0.05	0.04		71.09	74.94	
2+3+4+5	93.57	0.05	0.04		88.81	90.30	

6-3-4 Substantial Flotation Test

(1) Regrinding of Bulk Concentrate

After regrinding and cleaning of the bulk concentrate 3 times, the cleaned concentrate was reground for 1.5 and 3.0 minutes and then these concentrates were separated to the copper and lead concentrates by differential flotation. This time, potassium ferrocyanide was applied as a depressor for the copper. The test flowsheet is shown in Fig.2-6-30, and the flotation results are shown in Table 2-6-21.

Compared with no regrinding, the case of 1.5 minutes grinding increased lead recovery by about 3%.But, 3.0 minutes regrinding showed no improvement in the recovery of both the copper and the lead. Overgrinding might have occurred (Appendix 38-1).

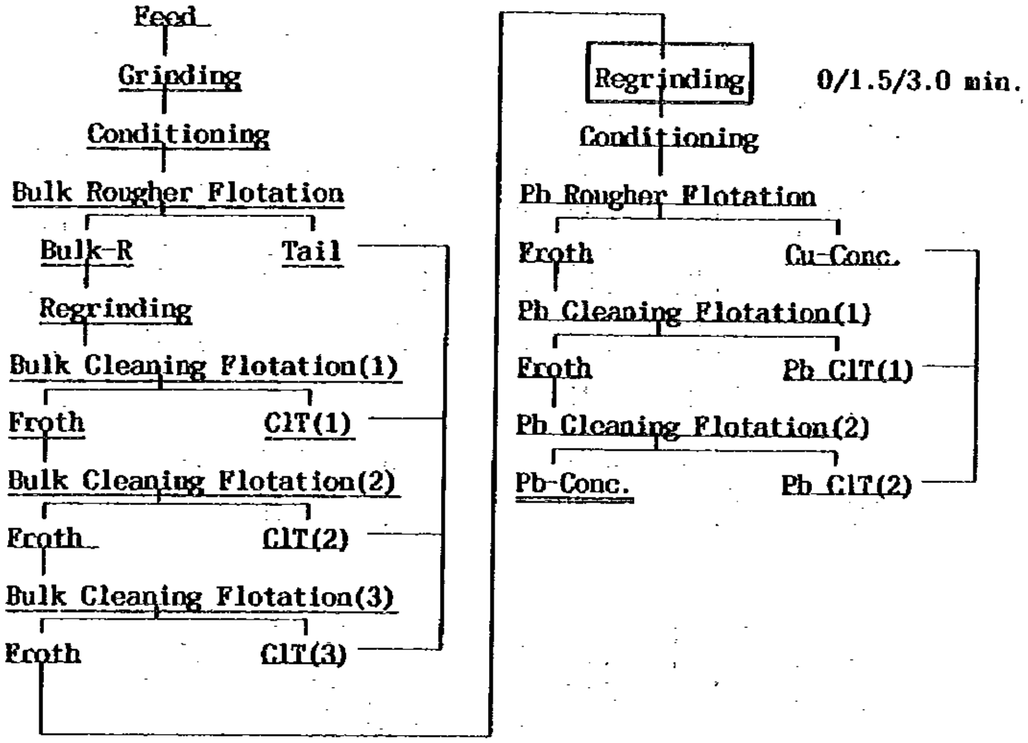


Fig.2-6-30 Flowsheet of the Differential Flotation with/without Regrinding of the Rougher Concentrate,Complex Ore

Table 2-6-21

Result of Flotation Test of the Differential Flotation with/without Regrinding of the Rougher Concentrate, Complex Ore

Products	Weight	Assay(%)			Distribution(%)		
Regrinding 0 min.							
	(%)	Cu	Pb	Zn	Cu	Pb	Zn
Feed	100.00	1.62	1.14	0.03	100.00	100.00	100.00
Pb-Conc.	1.34	15.1	58.1	1.62	12.10	68.18	62.03
Cu-Conc	4.02	34.59	8.38	0.08	83.27	26.31	7.80
Tail	94.64	0.08	0.07	0.01	4.63	5.51	30.17
Regrinding 1.5 min.							
	(%)	Cu	Pb	Zn	Cu	Pb	Zn
Feed	100.00	1.69	1.11	0.03	100.00	100.00	100.00
Pb-Conc.	1.34	15.0	59.0	1.62	11.89	71.27	63.67
Cu-Conc	4.02	35.16	7.11	0.05	83.53	23.05	5.37
Tail	94.64	0.08	0.07	0.01	4.58	5.68	30.96
Regrinding 3.0 min.							
	(%)	Cu	Pb	Zn	Cu	Pb	Zn
Feed	100.00	1.69	1.09	0.03	100.00	100.00	100.00
Pb-Conc.	1.47	21.2	54.0	1.38	18.42	72.71	62.67
Cu-Conc	3.89	33.46	6.00	0.03	77.01	21.54	4.83
Tail	94.64	0.08	0.07	0.01	4.57	5.75	32.50

(2) Lead-Copper/Zinc Bulk Differential Flotation and Cleaning Flotation

As a separation process for the lead, copper and zinc complex ore, bulk-differential flotation tests were conducted, and the cleaning flotation of the lead and copper rougher concentrates was also carried out.The test flowsheet is shown in Fig.2-6-31 and the result is shown in Table 2-6-22. Preliminary test results helped to define the test methodology. The bulk concentrate was reground and cleaned 3 times, and its concentrate was reground again. The copper minerals were depressed by potassium ferrocyanide, the lead rougher concentrate was floated through lead rougher flotation and this was cleaned 4 times. The cleaned concentrate was recovered as the final lead concentrate. The copper concentrate was recovered as the sink of the lead rougher. The other flotation products of the middlings were distributed in the lead and copper concentrate.

As a result of this test, the lead concentrate was 64.80% in grade and 79.90% in recovery. The copper concentrate was 40.14% in grade, 85.95% in recovery. More than 6 0% of total zinc content was distributed in the lead concentrate, 1.5% zinc in grade.

After the lead of the reground rougher concentrate was depressed by sulphur dioxide, starch and calcium hydroxide, the copper rougher concentrate was recovered and cleaned to make the copper concentrate. However, the separation of the lead and copper was poor because of difficulties in the depressing of the lead.

Here, the comparison of the straight differential flotation and the bulk differential flotation is shown in Fig.2-6-33(1) and the effects of sulphur dioxide and potassium ferrocyanide as a depressant of copper minerals is shown in Fig. Fig.2-6-33(2).

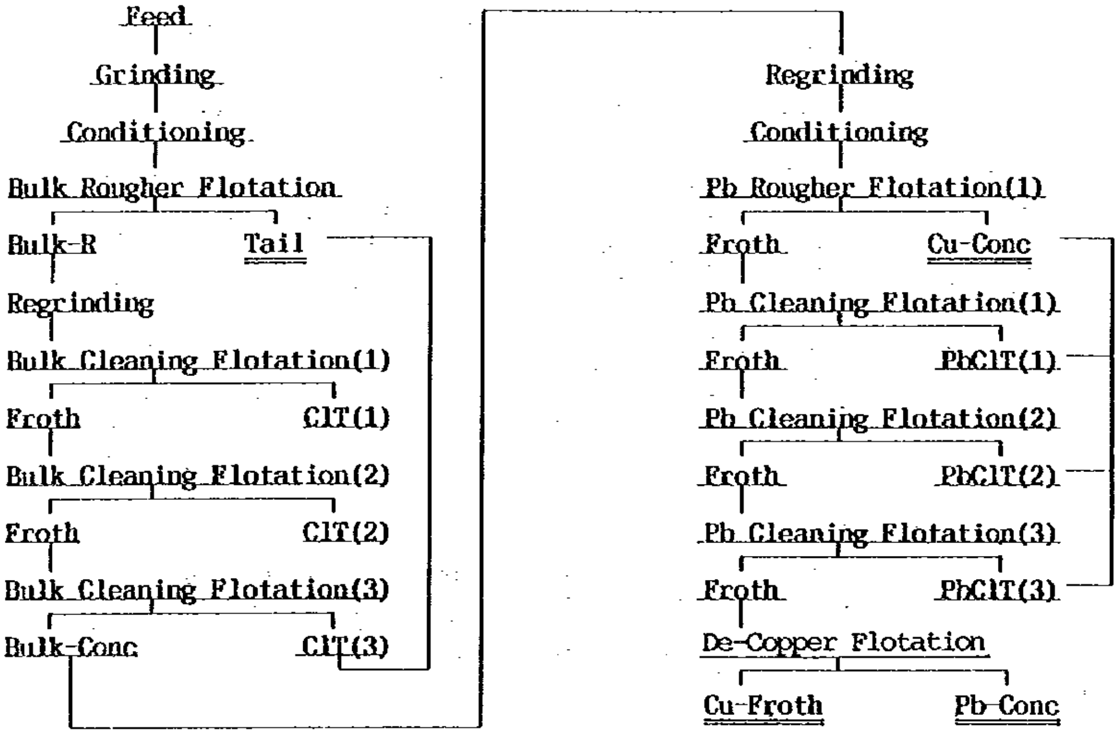


Fig.2-6-31 Flowsheet of the Bulk-Differential Flotation Test(Cu-Pb), Complex Ore

Table 2-6-22 Result of the Bulk-Differential Flotation Test(Cu-Pb), Complex Ore

Product	Weight	Assay(%)			Distribution(%)		
	(%)	Cu	Pb	Zn	Cu	Pb	Zn
Feed	100.00	1.72	1.07	0.03	100.00	100.00	100.00
Pb-Conc	1.31	11.30	64.80	1.51	8.62	79.90	61.65
Cu-Conc	3.68	40.15	3.94	0.05	85.94	13.61	5.45
Tail	95.01	0.10	0.06	0.01	5.44	6.49	32.90

(3) Separation of Copper, Lead and Zinc

In the case of separation for copper, lead and zinc from complex ore, the behavior of zinc in flotation is important. This complex ore contained so little zinc only 0.03% that the behavior of zinc was not clear. Tests carried out to date indicated that the zinc tended to move with the lead. Then, zinc separation tests were carried out on the lead concentrate which contained zinc.

The test flowsheet is shown in Fig.2-6-32, and the flotation results are shown in Table 2-6-23.

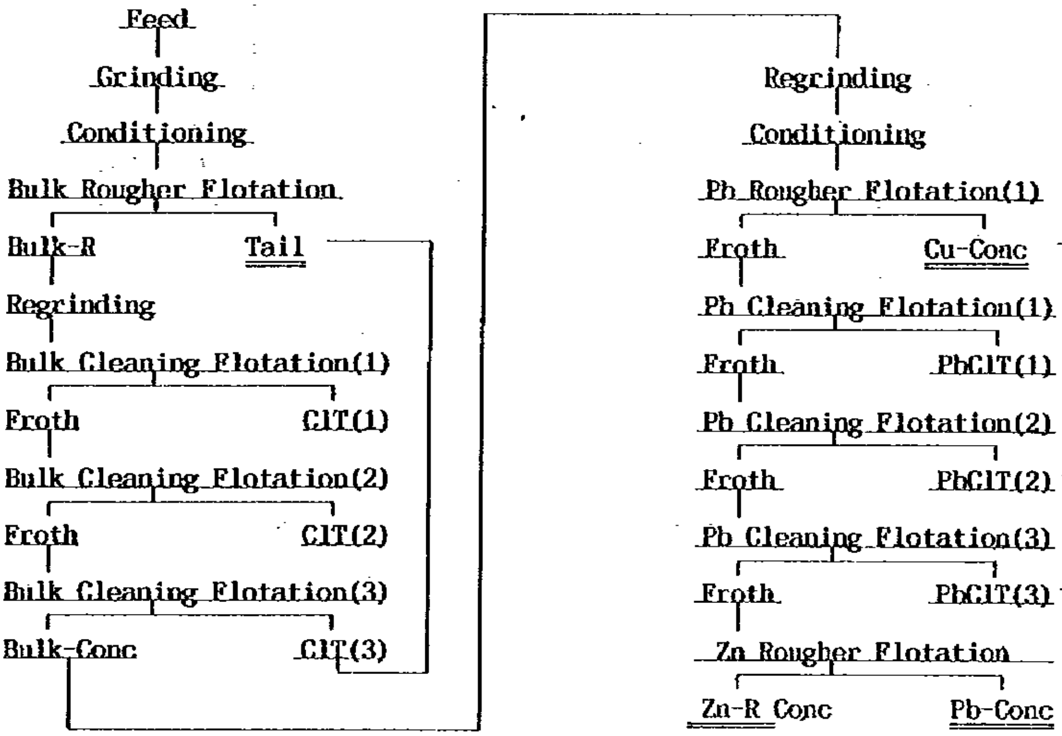


Fig.2-6-32 Flowsheet of the Bulk-Differential Flotation Test(Cu-Pb-Zn), Complex Ore

Table 2-6-23 Result of the Bulk-Differential Flotation (Cu-Pb-Zn)Test, Complex Ore

Product	Weight	Assay(%)			Distribution(%)		
	(%)	Cu	Pb	Zn	Cu	Pb	Zn
Feed	100.00	1.72	1.07	0.03	100.00	100.00	100.00
Pb-Conc	1.09	11.80	64.0	1.43	7.49	65.6	48.25
Zn-R-Conc	0.22	8.75	68.5	1.95	1.13	14.30	13.40
Cu-Conc	3.68	40.15	3.94	0.05	85.94	13.61	5.54
Tail	95.01	0.10	0.06	0.01	5.44	6.49	32.90

In this test, only 2.2g of zinc rougher concentrate was obtained due to low grade in the feed ore. The zinc concentrate grade was 1.95% zinc, 68.5% lead. It was difficult to recover the zinc concentrate. (Appendix 38-2).

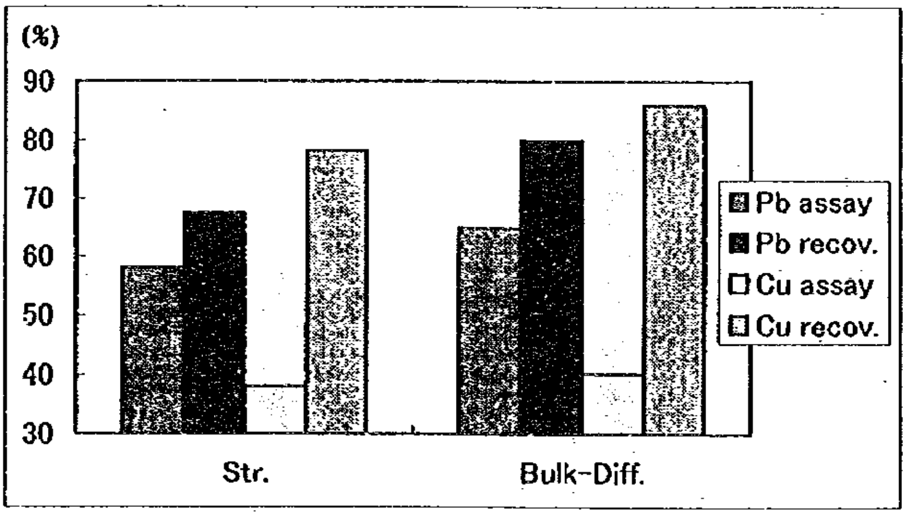


Fig.2-6-33(1) Comparative Results of the Straight and Bulk Differential Flotation

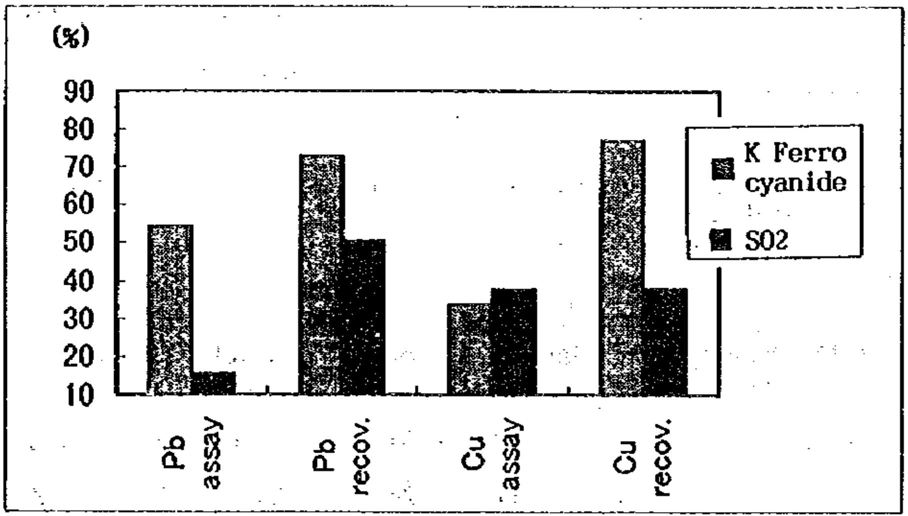


Fig.2-6-33(2) Effect of Potassium Ferrocyanide and Sulphur Dioxide

(4) Cleaning of Rougher Copper and Lead Concentrates

While maintaining the recoveries of the copper and lead, tests of increasing the grade of each concentrate were carried out. The test flowsheet is shown in Fig.2-6-34, and results are shown in Table 2-6-24. After the lead rougher concentrate was cleaned 4 times, a final lead concentrate grade of 73.4% lead was achieved, although recovery was only 50.82%. After cleaning 3 times, the grade of concentrate was 61.9% lead with recovery of 79.86%, the copper concentrate was 37.2% copper and its recovery was 84.99% (Appendix 38-3).

Table 2-6-24 Result of the Bulk-Differential Flotation Test,

Product		Weight	Assay(%)			Distribution(%)		
		(%)	Cu	Pb	Zn	Cu	Pb	Zn
0	Feed	100.00	1.68	1.08	0.04	100.00	100.00	100.00
1	Pb-Conc	0.75	2.79	73.4	2.40	1.25	50.82	50.99
2	PbClT(4)	0.65	23.1	48.5	0.71	8.92	29.04	13.05
3	PbClT(3)	0.35	61.8	5.84	0.05	13.05	1.91	0.50
4	PbClT(2)	0.62	61.0	4.72	0.04	22.45	2.69	0.70
5	PbClT(1)	1.28	50.0	4.67	0.05	38.03	5.51	1.81
6	Cu-Conc	1.59	12.1	2.22	0.06	11.46	3.26	2.71
7	ClT(3)	0.51	1.13	1.11	0.03	0.34	0.52	0.43
8	ClT(2)	1.25	0.46	0.50	0.03	0.34	0.58	1.07
9	ClT(1)	8.24	0.23	0.23	0.02	1.13	1.75	4.68
10	Tail	84.76	0.06	0.05	0.01	3.03	3.92	24.06
	1+2	1.40	12.2	61.9	1.62	10.17	79.86	64.02
	3+4+5+6	3.84	37.2	3.77	0.05	84.99	13.37	5.72
	7+8+9+10	94.76	0.09	0.08	0.01	4.84	6.77	30.24

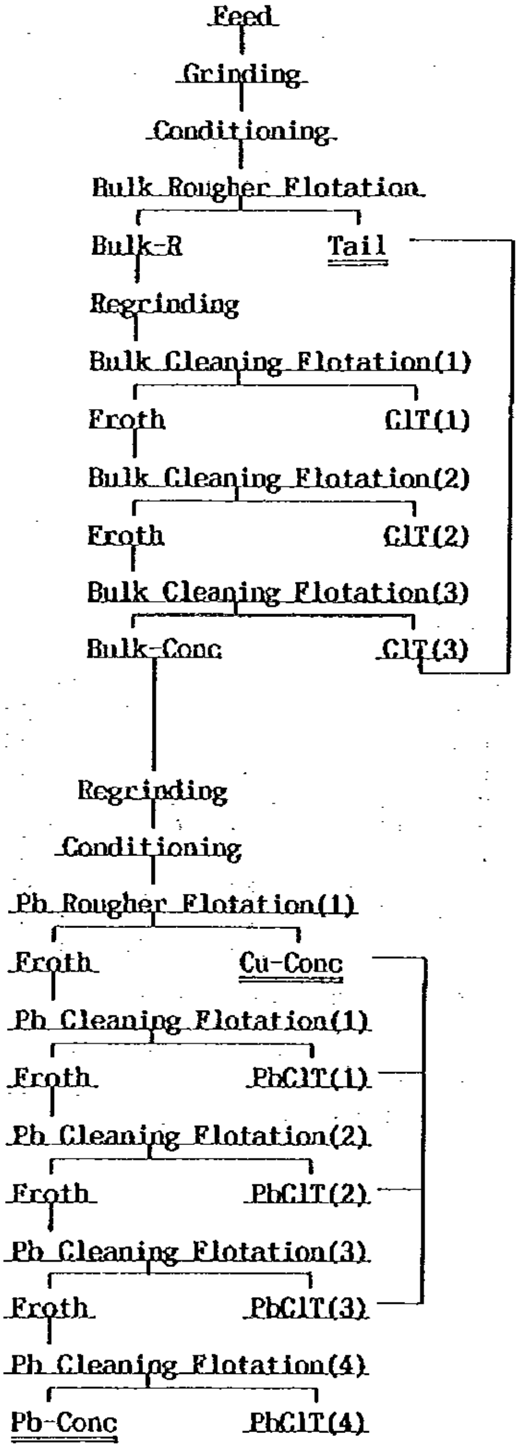


Fig.2-6-34 Flowsheet of the Bulk-Differential Flotation Test, Complex Ore

(5) Confirmation Test

From the test results described above, optimum conditions for bulk and differential flotation such as grinding size, kinds and dosage of flotation reagenta, flotation time etc were determined and a comprehensive test was carried out to confirm the results.

In bulk flotation, grinding size was -75 μ m 62.18%, frother was MIBC, collector were NaIPX and AP242 and rougher concentrate was reground and cleaned 3 times. In differential flotation, potassium ferrocyanide and sodium sulfide were used for the copper depressant in order to separate the copper and lead concentrate. As a result, copper concentrate was 32.3% copper in grade and 92.3% in recovery and lead concentrate was 66.1% lead in grade and 77. 6 % in recovery. The total zinc distribution in the lead concentrate was 52% and its grade was 1.88%.

The test flowsheet, flotation conditions and flotation results are shown in Fig 2-6-35, Table 2-6-25 and Table 2-6-26 respectively.

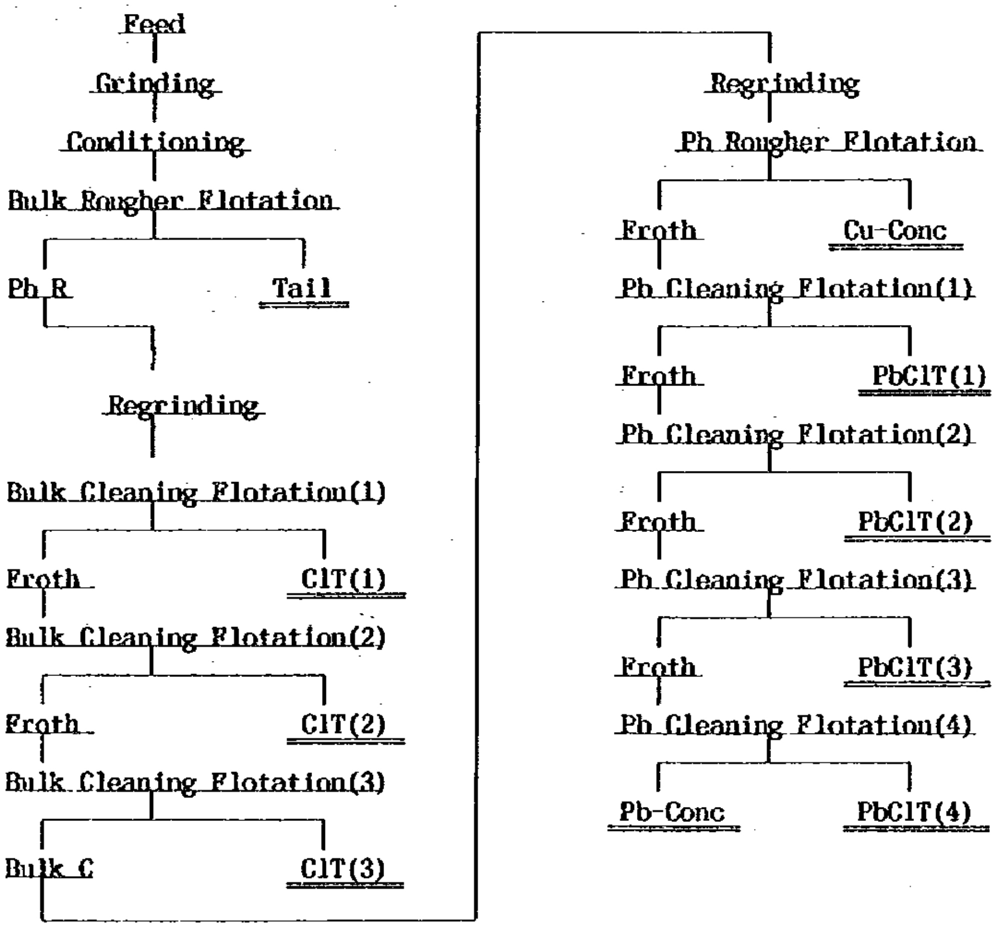


Fig.2-6-35 Flowsheet of the Confirmation Test

Table 2-6-25 Flotation Conditions of Confirmation Test

Table-
Condition
Test No. K-21

Circuit Name	Grind	Condition	Bulk Rou. Flotation	Regrind	Bulk Cleaning Flotation			Regrind	Condition	Pb Rougher Flotation	Pb Cleaner Flotation				Total
					1	2	3				1	2	3	4	
Grinding time (min.)	(14+14)*6			6*3				9							
Condition time (min.)		3							10		10	10	10	10	
Flotation time (min.)			17*3		10*3	12	10			5	5	4	3	3	
Pulp temperature (°C)	16		19	18	18-18	19-19	19-19	14		14-16	17-20	17-20	18-20	18-20	
Pulp pH	8.2	8.4-8.4	8.4-8.3	8.4	8.4-8.3	8.3-8.3	8.3-8.3	10	10.0-10.3	10.3-9.8	9.7-10.5-10	9.7-10.2-9.	9.8-10.2-9.8	8.9-9.5-9.2	
Pulp ORP (mV)	120	136-134	134-134	117	117-117	118-145	124-131	30	30-10	10-60	50-50-70	60-80-90	80-90-90	120-100-120	
Feed size (-75 μ mWt%)	62.18														
Reagent (g/t)															
MIBC		48	33								1	1			83
AP242		60	15		2	1.3	2				1	1			82.3
NaIPX			20		3	2	2				1	1			29
Potassium Ferrocyanide								1000			500	500	500	200	2700
Na2S											5	5			10
	SUS Steel			SUS8 Mild2				SUS4 Mild1							
Test Mill	Rod Ball			Rod				Rod							
Test Machine	No.1	AG2000g	AG2000g	No.1	AG2000g	AG2000g	AG2000g	No.1	AG500g	AG500g	AG500g	AG500g	AG500g	AG500g	

Table 2-6-26 Flotation Results of confirmation Test

Test No. K-21

Products	Weight (g)	Weight (%)	Assay (%) Cu	Pb	Zn	Distribution (%) Cu	Pb	Zn
Feed	5910.8	100.00	1.72	1.14	0.05	100.00	100.00	100.00
0 Pb-Conc.	61.4	1.04	2.3	75.0	2.32	1.40	68.43	49.97
1 PbClT(4)	8.1	0.14	10.8	44.5	0.37	0.86	5.36	1.06
2 PbClT(3)	9.5	0.16	28.7	26.8	0.34	2.68	3.78	1.13
3 PbClT(2)	21.4	0.36	58.3	5.36	0.46	12.28	1.70	3.45
4 PbClT(1)	121.8	2.06	54.0	5.06	0.41	64.75	9.16	17.51
5 Cu-Conc.	131.4	2.22	9.71	2.28	0.08	12.56	4.45	3.69
6 CL-T(3)	67.5	1.14	0.56	0.68	0.04	0.37	0.68	0.95
7 CL-T(2)	131.5	2.22	0.32	0.40	0.03	0.41	0.78	1.38
8 CL-T(1)	595.2	10.07	0.24	0.24	0.02	1.41	2.12	4.17
9 Tail	4763	80.59	0.07	0.05	0.01	3.28	3.54	16.70
10								
1	61.4	1.04	2.3	75.0	2.32	1.40	68.43	49.97
1+2	69.5	1.18	3.3	71.4	2.09	2.26	73.79	51.02
1+2+3	79	1.34	6.3	66.1	1.88	4.94	77.57	52.15
1+2+3+4	100.4	1.70	17.4	53.14	1.58	17.22	79.27	55.60
1+..+5	222.2	3.76	37.47	26.78	0.94	81.97	88.43	73.11
1+..+6	353.6	5.98	27.16	17.68	0.62	94.53	92.88	76.80
6	131.4	2.22	9.71	2.28	0.08	12.56	4.45	3.69
5+6	253.2	4.28	31.02	3.62	0.24	77.31	13.61	21.20
4+5+6	274.6	4.64	33.14	3.75	0.26	89.59	15.31	24.65
3+4+5+6	284.1	4.80	32.99	4.52	0.26	92.27	19.09	25.78
2+..+6	292.2	4.94	32.38	5.63	0.26	93.13	24.45	26.83
9+10	5358.2	90.66	0.09	0.07	0.01	4.69	5.66	20.87

(6) Chemical and Microscopic Analyses of Concentrates

The copper and lead concentrate which were obtained by the confirmation test were chemically analysed and microscopically examined. The results are shown in Table 2-6-27 and 2-6-28 respectively.

Table 2-6-27 Chemical Analysis of the Copper ,Lead Concentrate and Zinc Rougher Concentrate

Element	Analytical Result(%)		
	Cu-Conc.	Pb-Conc.	ZnR-Conc. *
Cu	39.4	2.29	31.5
Pb	6.01	71.5	26.4
Zn	0.02	2.1	0.64
S	12.5	13.9	14.6
Fe	4.16	1.15	2.76
SiO2	24.1	4.08	13.5
Al2O3	4.43	0.80	3.29
MgO	0.57	0.08	0.43
CaO	1.97	0.37	1.37
K2O	0.72	0.10	0.38
Na2O	1.1	0.11	0.51
Cl	0.02	0.01	0.02
F	0.01	<0.01	<0.01
Mo	<0.05	<0.05	<0.05
Te	<0.05	<0.05	<0.05
Sb	<0.05	<0.05	<0.05
As	<0.05	0.16	0.05
Bi	<0.05	<0.05	<0.05
Cd	<0.01	0.09	0.03
C	N.D	2.2	N.D
Hg (ppm)	0.3	2.8	1.1
Re (g/t)	28	13	23
Au (g/t)	0.6	0.3	N.D
Ag (g/t)	62	89	N.D
Total	95.01	96.74	95.48

*Zinc Rougher Concentrate : Zn content(%) is very low due to lower Zn grade(%) in test samples.

Table 2-6-28 Microscopic Observation of the Copper ,Lead Concentrate and Zinc Rougher Concentrate

Flotation Product	Mineral							
	Gal	Cc	Bor	Cp	Py	Sp	C	G
Bulk concentrate	7.2	7.8	5.3	0.7	1.2	0.2		77.6
Lead concentrate	82.3	0.5	0.7	3.5	0.2	3.2	2.4	7.2
Copper concentrate	7.0	29.1	25.9	0.2	0.5	0.1		37.3
Zinc Rougher concentrate	30.7	24.4	20.2	0.1	0.1	0.3	0.2	24.1

Cc :Chalcocite Bor:Bornite Cp:Chalcopyrite Py:Pyrite
Gal:Galena Sp :Sphalerite C:Graphite G :Gangue Mineral

6-3-5 Selection of the Flowsheet

From the series of test results of complex ore, it is considered that the bulk differential flotation is adaptable to this complex ore. However, the zinc concentrate was not recovered due to low grade of sample ore and zinc is contained in the lead concentrate in this flowsheet. Flowsheet of the bulk differential flotation is shown in Fig.2-6-36.

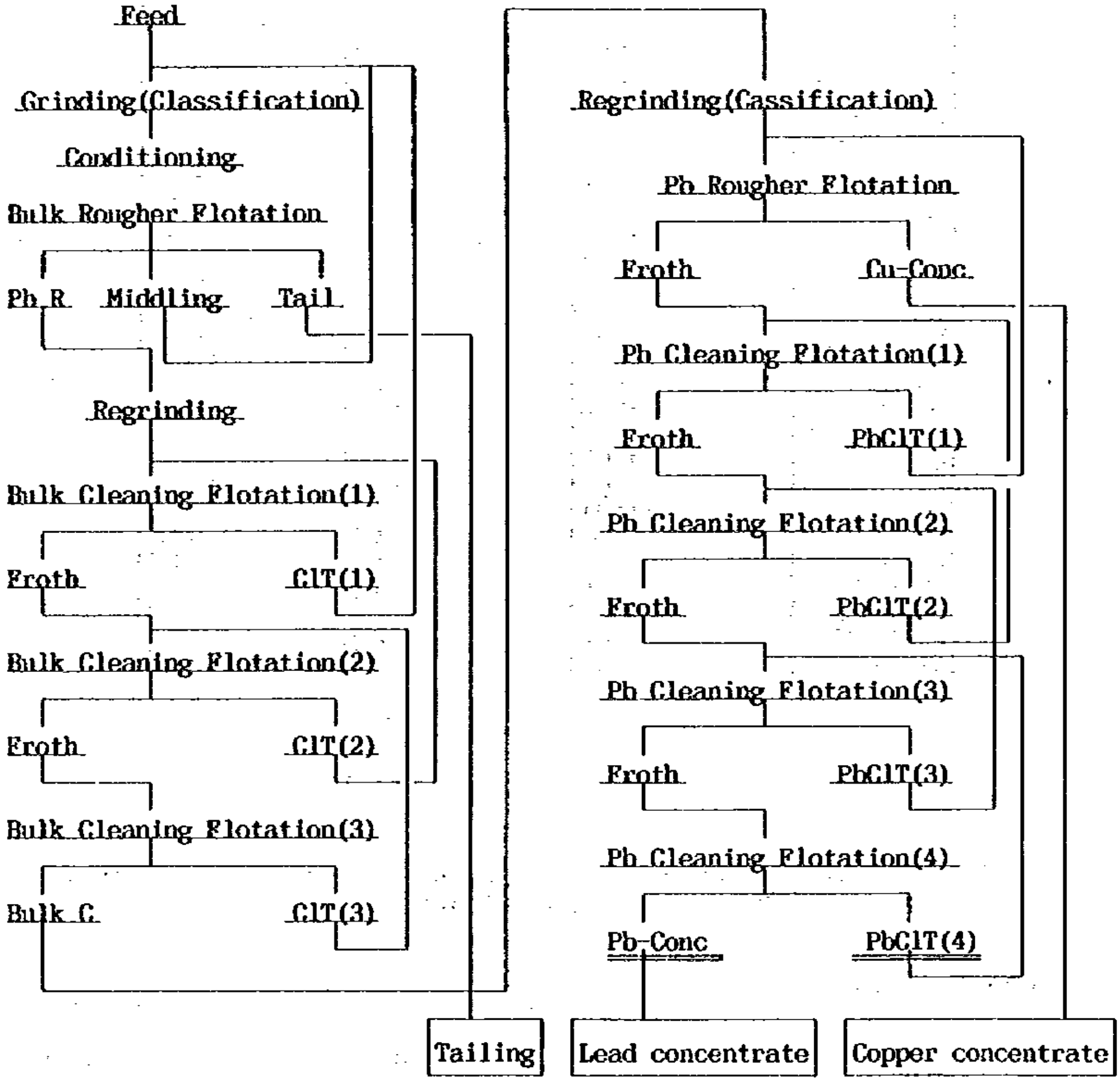


Fig.2-6-36 A Bulk Differential of Complex Ore

Chapter 7 Mining Technology in the Zhezkazgan Copper Mine

The ultimate aim of this project is to make an evaluation of Zhaman-Aibat deposits. In this connection, the existent operating condition, mining technology and operation cost in Zhezkazgan Mine have been investigated which seems to have a very similar geological condition and deposit formation to that in Zhaman-Aibat Mine have been investigated. This year, the investigation on those items centering around the South Mine where the deepest district is being mined has been concentrated particularly. The investigation on the ore dressing plant has also partly carried out.

7-1 Outline of Mine Operation

The mine output from four mines in 1994 and the planned output in 1995 with the remaining ore reserves are listed in Table 2-7-1, and operation of the Zhezkazgan Mine is summarized in Table 2-7-2. Among the four mines (Fig.2-7-2), production of the North Mine was initiated in the 1950's and the West and East Mines commenced their production in the 1960's. The South Mine which started to mining the ore located at greater depth, commenced in the 1970's. Since then, mine output from underground operations has increased remarkably. The output in 1994 was accounted to 16,840 thousand tons with an ore grade of 0.96%Cu. It is reported that approximately 70% of the total mine output was mined by these three underground mines.

The copper ore is transported to No.1 and 2 ore dressing plants in Zhezkazgan City by rail way and after processing in ore dressing plants copper and rhenium (hereinafter written with chemical notation) are recovered in the adjacent smelting plants. On the other hand, complex ore containing Cu, Pb and Zn is processed at No 3 ore dressing plant adjoining the East Mine and Pb and Zn concentrates were sold to the Chimkent smelter. Both mining and ore dressing operations for complex ore are currently suspended due to the present depression of metal markets. The outline of ore dressing is summarized in Table 2-7-3.

The open pit mining is carried out by the conventional methods which consist of the combination of rotary drilling machines, shovel and haulage tracks. Ore is reloaded to the open train wagons at a certain place in pit surface and transported to the ore dressing plants.

Underground mine development methods employ a combination of vertical shafts, haulage drifts with trolley trains (Fig.2-7-3) and underground primary crushers at the bottom of the shaft. A mechanized room and pillar mining method is introduced in ore bodies depending on their morphological characteristics. Most ore bodies are excavated by the trackless mining method utilizing jumbo drills, front-end-loaders and dump trucks. Mined ore is transported to the ore path and is loaded into mine cars at the main haulage level. The rocks are relatively hard. The rock support is performed by rock bolting and reinforced with shotcrete at a weak portion. In a thicker orebody than 8 meters, two slicing method is applied. That is, the upper slice is mined in the first stage by ordinary room and pillar method and the lower one is extracted with bench cutting method after the upper has mined out. Sections with the copper grade over 2.5% or with a thickness over

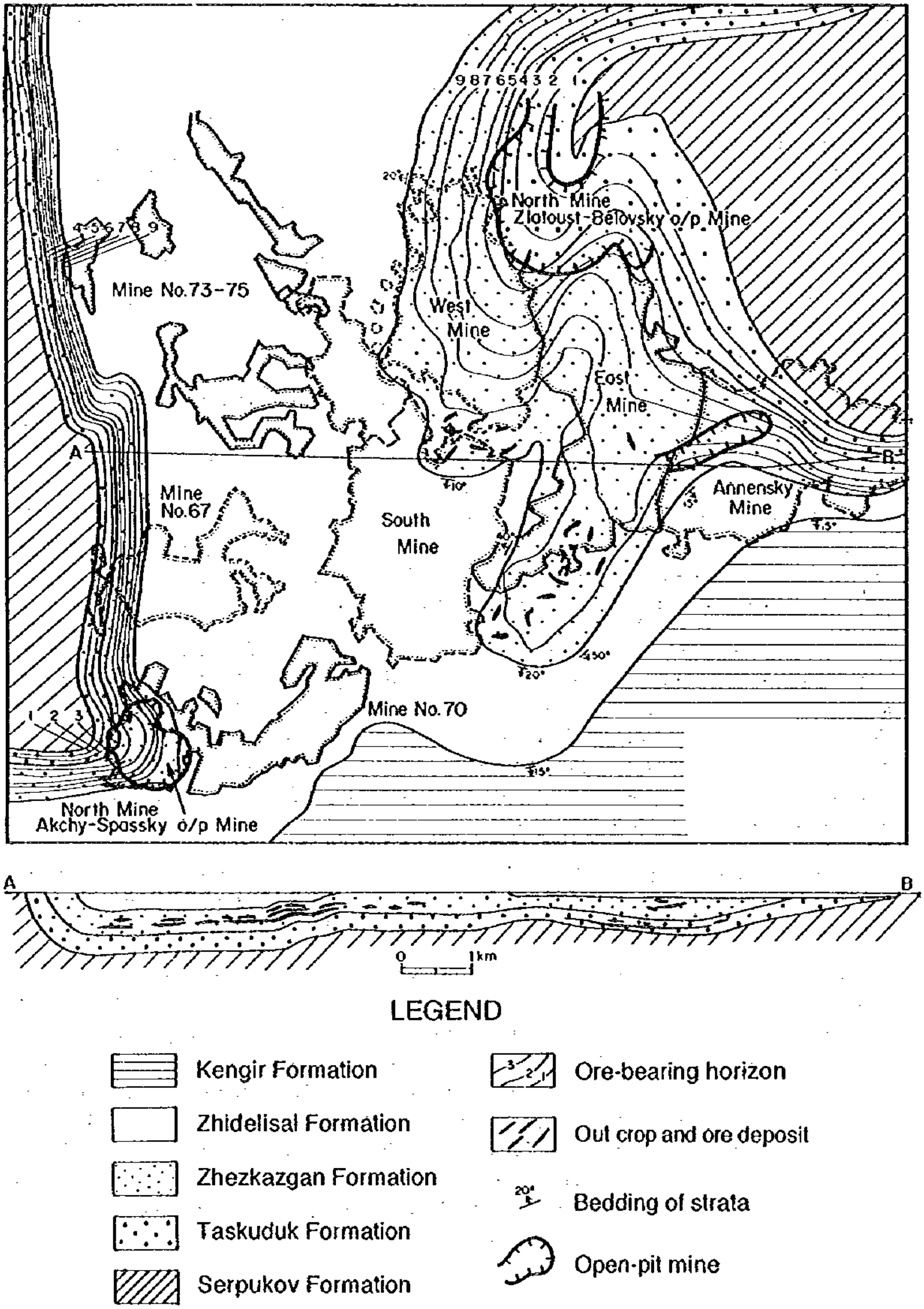


Fig. 2-7-1 Mining Areas of the Zhezkazgan Mine

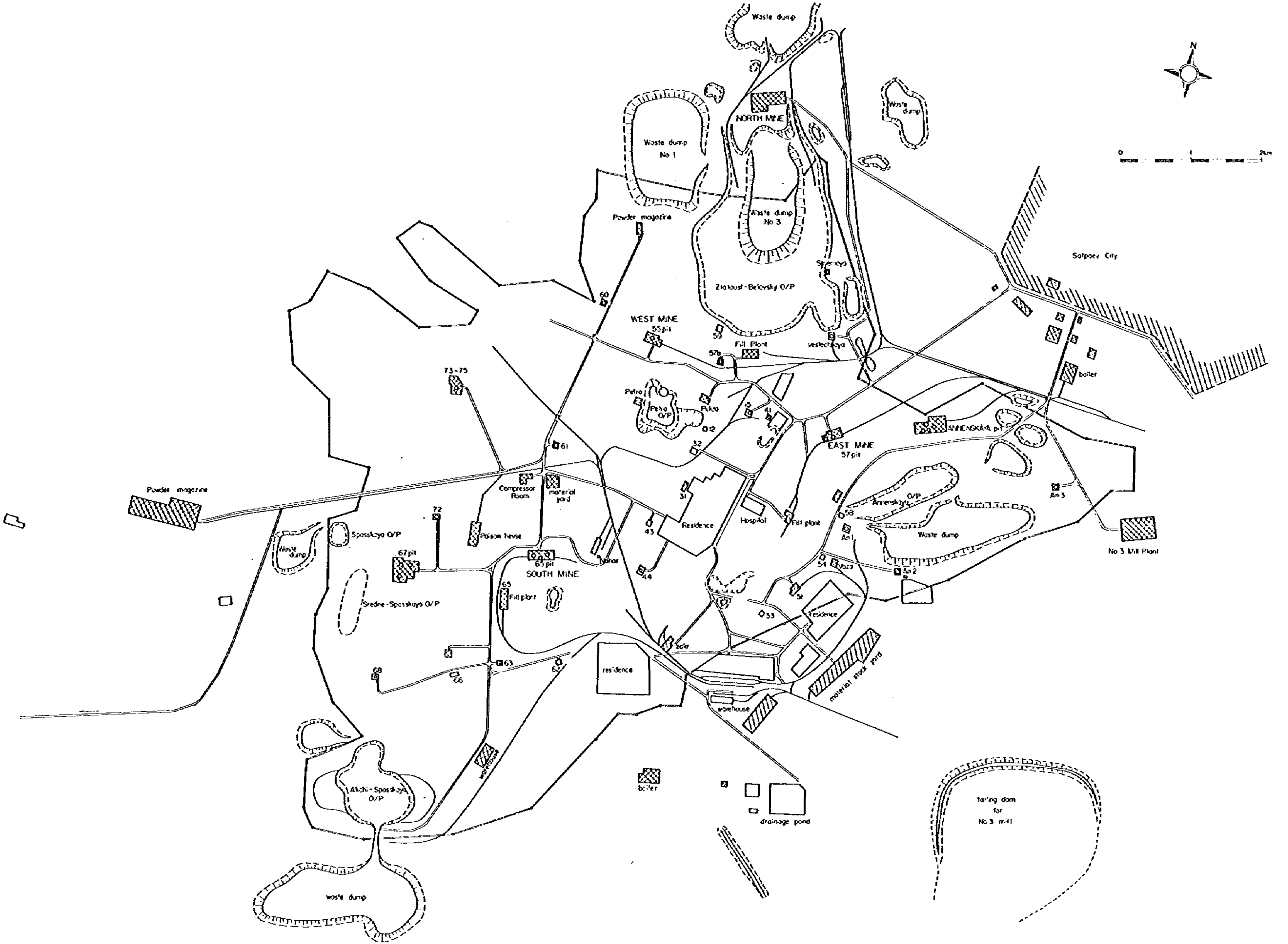


Fig. 2-7-2 Location Map of Mining Facilities in the Zhezkazgan Mine

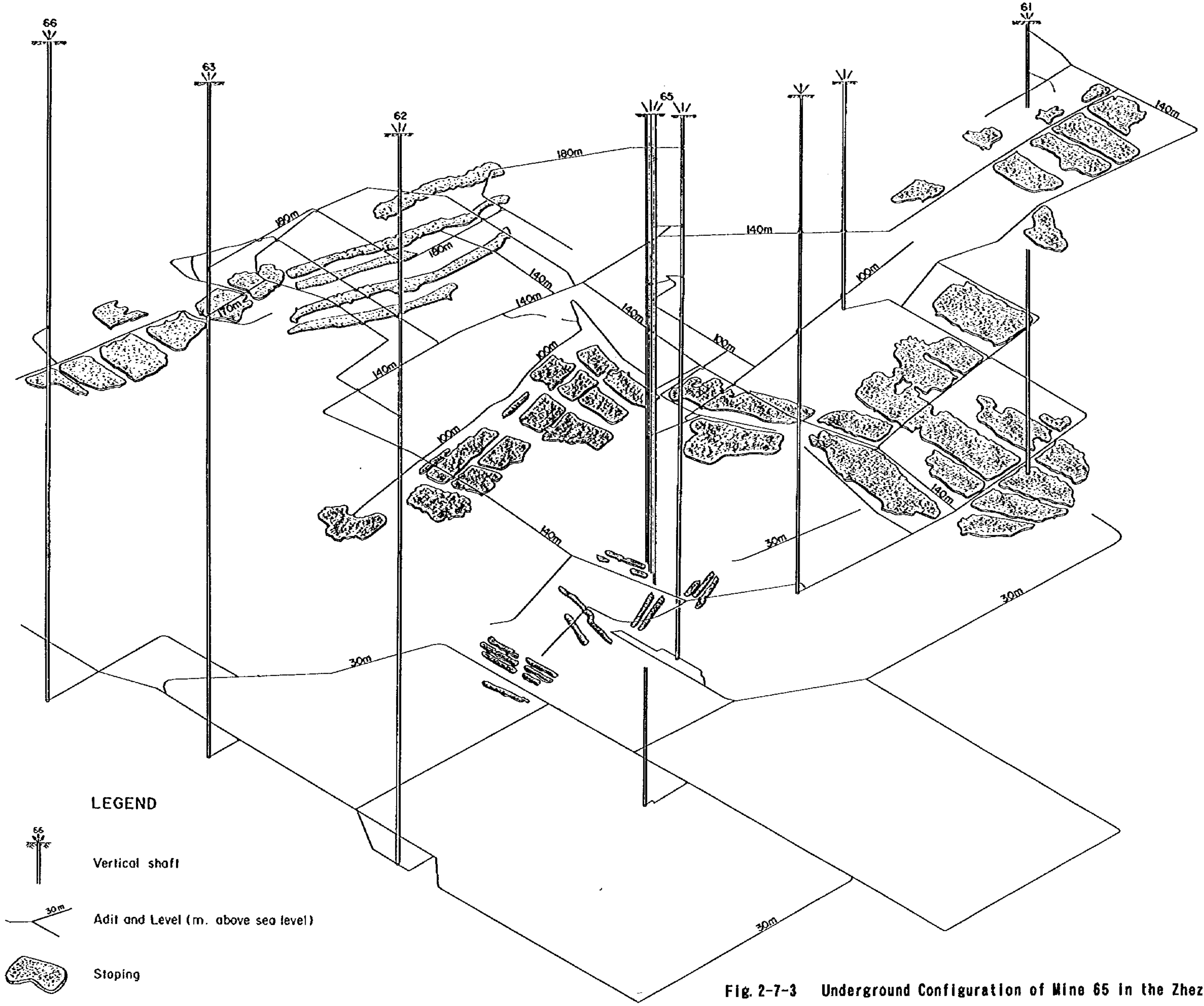


Fig. 2-7-3 Underground Configuration of Mine 65 in the Zhezkazgan Mine

18m have to be exclusively mined by backfilling method which theoretically will leave no pillars. In steeply dipping ores, a sublevel stoping method is introduced. Slime transported from ore dressing plant and mixed with cement, is used as a filling materials.

Mining recoveries depending on morphology of the ore deposit, rock properties, depth beneath the surface, are approximately 80% in underground mines and 96% in open pit mines. The methods and standards of mine development are based on the technical instructions prepared by the Giprotsvetmet Institute in Moscow.

In this 1995's campaign, the Zhezkazgan mine and a part of the ore dressing plant were investigated and data not only on technology but also on costs of operation were obtained. The results are shown in Table 2-7-4 and total employees in mining and ore dressing plant are listed in Table 2-7-5. Details of mining operation cost are discussed in part II, Chapter 3. There were, however, some difficulties to get disclosing data due to the transfer of the management right of the Zhezkazgantsvetmet and peculiar organization system of management. Data obtained in this year are not sufficient to complete our study. Actual mining costs in 1995 (January~July) were 356~418 Tenge/ton-ore in underground mine and 261 Tenge/ton-ore for open pits. There are estimated as 5.9~7.0 $ US/ton-ore and 4.4 $ US/ton-ore, respectively. Compared with the cost of operating mines in the western world, operating costs for open pit of the Zhezkazgan Mine is rather higher. It is because the waste/ore ratio is as high as 7~8 and four small scale open pit mines are operated separately.

Table 2-7-1 Output of the Zhezkazgan Mine

	1994		Plan in 1995		Remaining Reserves	
	×1000 tons	Cu%	×1000 tons	Cu %	×1000 tons	Cu %
East Mine	4,401	1.15	5,200	1.00	* 62,855	1.29
West Mine	2,531	1.11	3,000	0.84	37,768	1.02
South Mine	4,746	1.06	6,000	1.10	136,224	1.09
North Mine	5,162	0.64	3,850	0.52	61,434	0.68
Total	16,840	0.96	18,050	0.90	298,281	1.04

* : showing only for Annensky District but unknown for main mine.

Table 2-7-2 Summary of the Zhezkazgan Mine Operation (1)

Mine Name	East Mine	West Mine	South Mine	North Mine
Starting Year	1967	1965	1975	1956
Mine Output (×1000t, planned in 1995)	5,200	3,000	6,000	1,250 (KZB) 2,600 (ACK)
Daily Output (t)	17,049	9,836	19,672	12,623
Grade (%Cu)	1.00	0.84	1.10	0.44 (KZB) 0.60 (ACK)
Cut-off (%Cu)	0.4	0.4	0.4	0.2
Remaining Ore Reserve (×1000t)	62,855 (Annensky)	37,768	136,224	61,434
Ore grade (%Cu)	1.29	1.02	1.09	0.68
Total Employees	2,200	701	2,032	1,712
Mining Workers	NA	237	223	250
Co-Workers	NA	322	1,471	1,255
Staff, Engineers	300	142	338	207
Operating Schedule (Day/Year)	305	305	305	350
Operating Schedule (Hour/Shift)	6	6	6	12
Operating Schedule (Shift/Day)	3	3	3	2
Skip Shaft	57, 57 bis 42	55, 31	45, 65, 67	
Mining Method	Panel/Pillar Room/Pillar Slicing	Panel/Pillar	Panel/Pillar Room/Pillar Slicing	Open Pit Strip. Ratio W/O = 7.8
Ore Recovery (%)	82.7	80.4	82.8	96.0
Waste Dilution (%)	5.7	6.3	5.2	7.4
Working Depth (m below surface)	420	320	450	280
Pillar Diameter (m)	8~12	8~12	8~12	
Pillar Spacing (m)	20	20	20	
Maximum Working Height(m)	18	18	18	15 m Bench H.
Minimum Working Height (m)	4	4	4	10m Bench Width
Maximum Height per one Shot Explosive (m)	7	7	7	Pit slope, 42°
Powder Factor (kg/t-ore)	2.4	2.2	1.9	1.55
Drilling Dia /Length	43mm×4m	43mm×4m	43mm×4m	250mm×18m
Burden/Spacing	1 m	1 m	1 m	3~4m×6~8m
Filling Materials	1st. Slime + Cement 2nd. Excavated waste*	1st. Excavated rock*	1st. Slime + Cement 2nd. Excavated waste*	

* Prevented against surface subsidence

NA : not available

Table 2-7-2 Summary of the Zhezkazgan Mine Operation (2)

Mine Name	East Mine	West Mine	South Mine	North Mine
Supporting Method	Cement mortar type rock bolts ϕ 16~18 mm×2~3m Shotcrete (t = 20mm)	Cement mortar type rock bolts ϕ 16~18 mm×2~3m Shotcrete (t = 20mm)	Cement mortar type rock bolts ϕ 16~18 mm×2~3m Shotcrete (t = 20mm)	
Nos. of Equipment				
Drills	NA	21	47	22
Front-End Loaders	NA	7	13	Excavator 32
Gathering Loaders	NA	12	25	2
Dump Trucks	NA	20t - 24	20t - 54 40t - 1	40t - 56 110t - 16
Nos. of Shafts	13	11	13	
Main Shaft Depth (m)	400	300	450	
Shaft Diameter (m)	5.5~7.0	4.5~7.0	6.0~7.0	
Total Capacity of Winding machines(kW)	12,450	3,550	11,250	
Ventilation (m^3/min.)	107,400	52,200	93,600	
Drainage (m^3/hr)	625	208	504	
Power Consumption (MWH/month)	7,000	3,500	8,300	
$US1.00=60Tenge				
Production Cost (Tenge/t-ore)	356	418	414	261
Actual Cost in July ($US/t-ore)	5.9	7.0	6.9	4.3

NA : not available

Table 2-7-3 Summary of the Zhezkazgan Ore Dressing Plants

Ore Dressing Plants	No.1	No.2	No.3
Constructed in	1953	1971	1986
Plant Capacity (×1000t/year)	8,100	14,200	4,200
Feeding Rate (t/day)	22,000	40,000	Suspended now (5,000 for plan)
Feed Grade (%Cu)	0.9~1.2	0.6~1.2	
Ore Type	Cu	Cu	Cu, Pb, Zn
Concentrate Grade(%Cu)	37.5	37.5	
Mill Recovery(%)	90 ~ 92	86 ~ 90	
Water Consumption (m^3/t-ore)	4	4	3.5
Electric Consumption (KWH/t-ore)	38	38	40
Total Employees	No.1 & No.2 : 1917		411
Ore Dressing Cost	201 Tenge/t-ore ($US3.35/t)	201 Tenge/t-ore ($US3.35/t)	NA

NA : not available

Table 2-7-4 Operation Cost of Mining and Ore Dressing (Tenge/ore-ton)

	Actual Records in July 1995	Actual Records January ~ July	
Mining			
East Mine	812.29	366.89	Underground mining
West Mine	450.17	356.06	Underground mining
South Mine	618.87	418.04	Underground mining
North Mine	275.77	260.69	Open pit mining
Ore transportation	20.97	From mine site to No.1 & 2 ore dressing plants	
Ore Dressing No.1 & 2	201.67	171.40	Cu concentrate

Table 2-7-5 Employees in Mining and Ore Dressing Plant (Aug. 1995)

	Staff	Workers	Total
East Mine	300	1,900	2,200
West Mine	142	559	701
South Mine	338	1,687	2,025
North Mine	207	1,505	1,712
No.1 & 2 Ore Dressing	180	1,737	1,917
No.3 Dressing	63	348	411
Total	1,230	7,736	8,966

7-2 Mining Technology

All Zhezkazgan mines carry out the underground mining operations except for the North Mine, while at Zhaman-Aibat the deposits are located 500～700 m below the surface with a stratification of 6 m thickness or less. In consequence, it seems to be reasonable for the Zhaman-Aibat mine to apply the same mechanized room and pillar mining method as that being now adopted in the Zhezkazgan mine. The critical question of this mining method is that pillar size becomes bigger and hence the ore recovery decreases the deeper the working place gets. The cut and fill mining method might be more suitable to improve such lower ore recovery. This method must however lead to increased mining costs in this case and the problem of profitability might appear.

Such being the case, it has been finalized that we have to investigate first a mining method to be applied to Zhezkazgan deposits this year in order to develop the conceptual design next year.

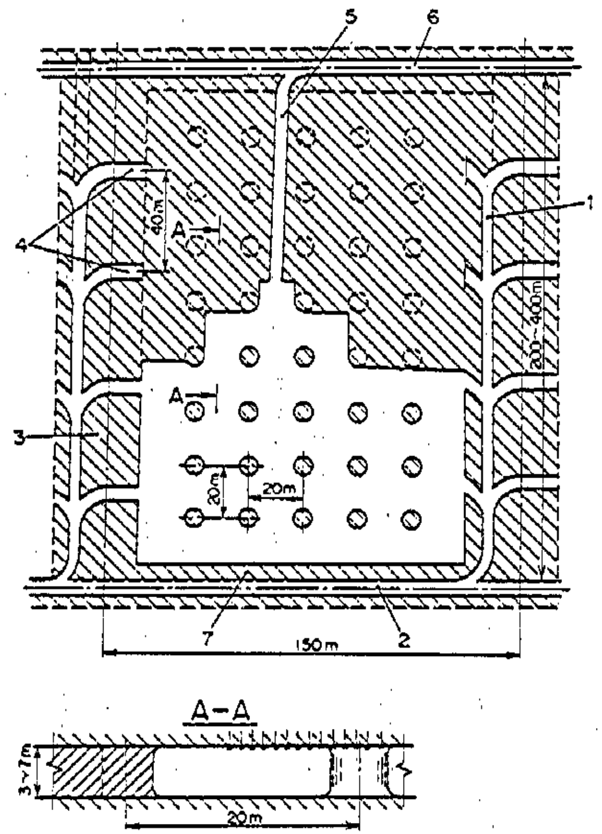
7-2-1 Underground Mining Method

The underground mining method being applied in Zhezkazgan deposits is divided into 3 classes depending on ore grade and profitable working output. While the panel and pillar method that is one of the room and pillar mining methods is being used in a general way, a room and pillar method (also referred to us a cut and fill method) is being applied for 2.5 % or higher Cu grades. It is prohibited to work inside chamber spaces higher than 18 m. In consequence, when the deposit layer is thicker than 18 m or the working height increases to more than 18m due to steep dip (in the case of a steep dip stratification), it is prescribed that a sub level is to be arranged at every 18 m of vertical height and a slicing method by downward mining is to be adopted (by means of upward drilling). Each mining method follows the practical guidelines established in the late U.S.S.R. The basic technology such as working procedure, filling method or pillar arrangement is to be applied corresponding to the procedures and methods outlined in the governmental tentative guide of 1986 as well as the design guide produced by Giprotsvetmet Research in Moscow.

Panel and Pillar Mining Method (One of the Room and Pillar Mining Methods)

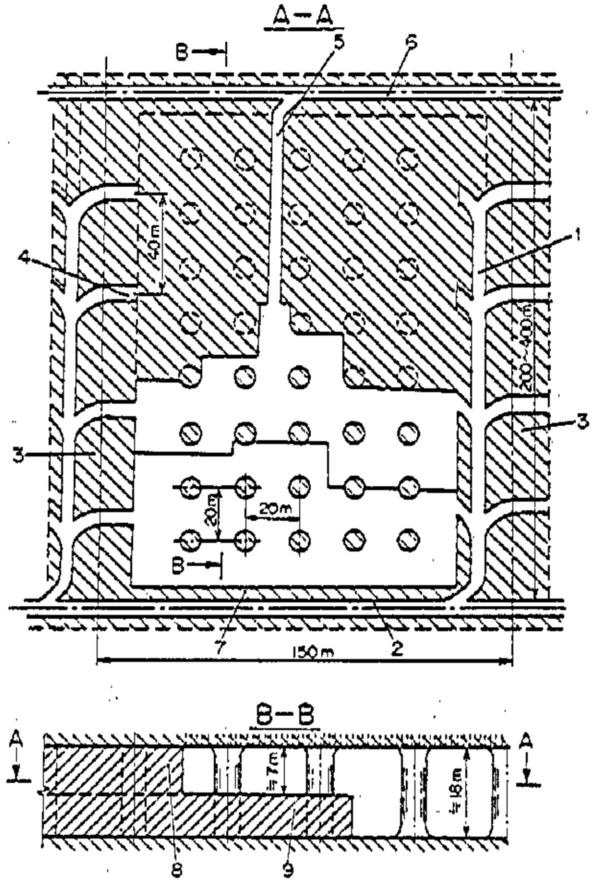
This mining method is being practiced for more than 75% of the output in Zhezkazgan deposits and is applied to ore bodies with a gentle slope and stratification thinner than 18 m with 2.5% or less Cu content. The panels are with 150～250 m × 200～400 m as shown in Fig.2-7-4. A panel drift for ventilation and haulage is excavated in a barrier pillar between adjoining panels. After that, thirling drifts for extraction are driven toward the ore body from the panel drift at 40 m intervals and are utilized for future access thereto.

Mining operation can be conducted to a maximum thickness of 8 m from the roof edge and barrier pillars remain after mining. Mining output is restricted by drilling capacity. After mined out, rock bolts (ϕ 20 mm×2500 mm) are installed in the roof at 1.2 m spacing. Rock bolt installation is reinforced with 20 mm thick cement mortar at weak positions such as at red sandstone or aleurolite. Pillars in panel are arranged at 20 m spacing between both centers. When the thickness exceeds 8 m, the lower part is mined out after the upper part is mined as shown in Fig.2-



1. panel drift
2. transport drift
3. barrier pillar
4. extraction drifts into a panel
5. ventilating drift
6. sectional ventilating drift
7. safety pillar

Fig. 2-7-4 **Panel and Pillar Mining Method, orebody with a gentle inclination, thinner than 8m and Cu content lower than 2.5%**



1. panel drift
2. transport drift
3. barrier pillar
4. extraction drifts into a panel
5. ventilating drift
6. sectional ventilating drift
7. safety pillar
8. upper stoping for benching
9. bench

Fig. 2-7-5 **Panel and Pillar Mining Method, orebody with a gentle inclination, over 8m in thickness, and Cu content lower than 2.5%**

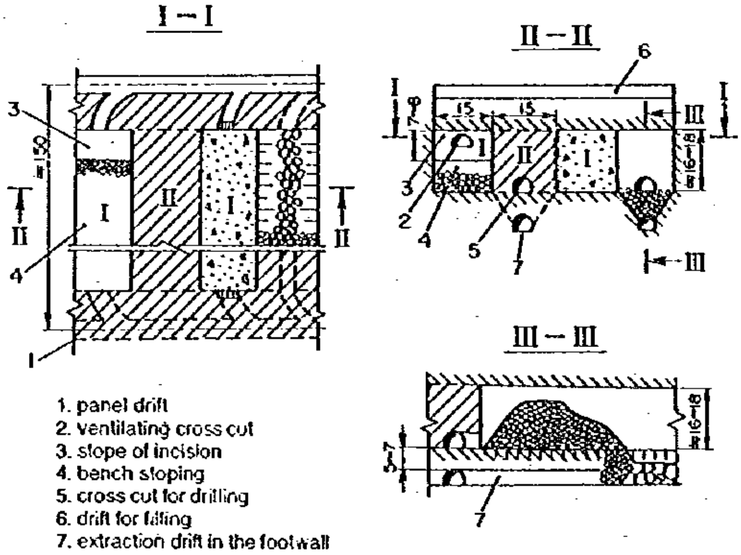
7-5. While the chamber space is usually not filled with waste material, it must be filled up when there is a structure or installation on the surface above the layer. A mixture of tailing slime and cement is used as filling material. Pillar size will be explained later.

Room and Pillar Mining Method (with Cut and Fill Mining Method)

This method is used for more than 2.5%Cu content by which about 20% of the output of the South Mine is obtained. As shown in Fig.2-7-6, after being mined out primarily by the panel and pillar method, a filling level is arranged 5～6 m above the hanging wall of the chamber space. After that, three injection bores of 105 mm diameter are drilled from the filling level toward the mined out space every 50 m. One of these bores is connected with a hydraulic transportation pipe for injection of filling material to be supplied from the filling plant on the surface. The other two bores are used for water removal. On the other hand, concrete partitions are installed at each cross cut inlets to the chamber space to prevent the filling material from flowing out. A drainage pipe is buried in each partition. When filling materials reaches the partition, a drainage hole is drilled into the panel. Filling capacity is nominally 30,000 m^3/month, but it reaches only 60 % in actual fact. The filling plants are located on the surface of the East and South Mines. No filling method is used at the West Mine. A typical compound of filling material after mined out primarily consists of the ratio of 1,200 kg/m^3 tailing slime: 150 kg/m^3 cement: 440 kg/m^3 water.

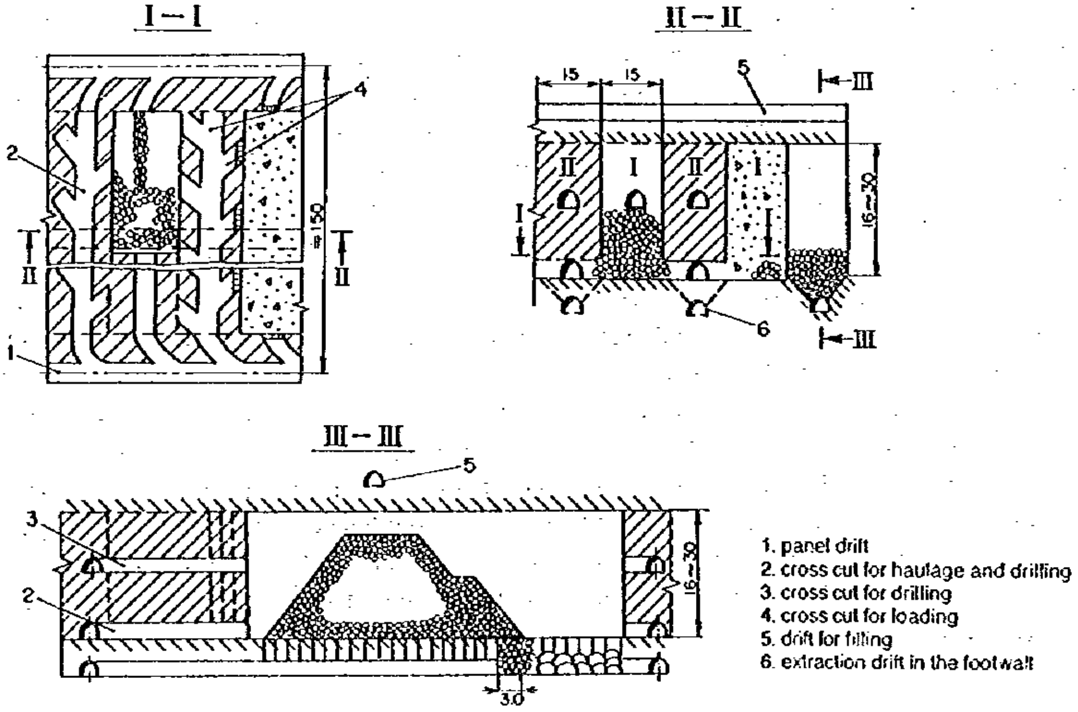
Secondary mining is started 6 months or later from the completion filling. A typical uniaxial compressive strength of filling material at that time is designed at 4 MPa. Curing strength of the material is changed on the basis of the design guide depending on working height and host rock properties. Secondary mining must be started by excavating a blasting drift with small sectional area in the pillar. After reaching the neighboring panel, a free face is secured by means of raising a slot into the panel pillar by using a handy hammer. Upward drilling in the shape of a fan (semicircular) and blasting the pillar, swell portion of blasted ore is recovered by using a gathering loader. Excavation is made safely by means of retreating. It is not permitted to enter into chamber spaces after they have been blasted due to a fragility of filling material. Therefore, the gathering loader is permitted to enter the chamber space up to only 5 m in front of the operator cabin.

In parallel with secondary mining, an extraction level is driven leaving pillars with 5～6 m in the foot wall where, by using the gathering loader, ore is recovered at this level. Assuming 5～6 m of sill pillar as a plate, blasted ore being put thereon is cascaded to the extraction level when the pill pillar is removed by similar way to draw out a plate. In order to prevent dispersion of ore the sill pillar is crushed by blushing toward or below the mined out chamber. Ore dropped below is loaded on dump trucks with the gathering loader. As personnel are not permitted to enter chamber spaces after being mined out, one shot of blasting allows only 5 m or shorter progress. Chamber spaces after secondary mining are filled with a mixture of tailing slime and little cement when there is a structure or installation on the surface or with a mixture of waste rock and slime when no structure or installation. When filling with waste rock only, a slot of about with 5 m^2 is sunk down from the filling level (Fig.2-7-7).



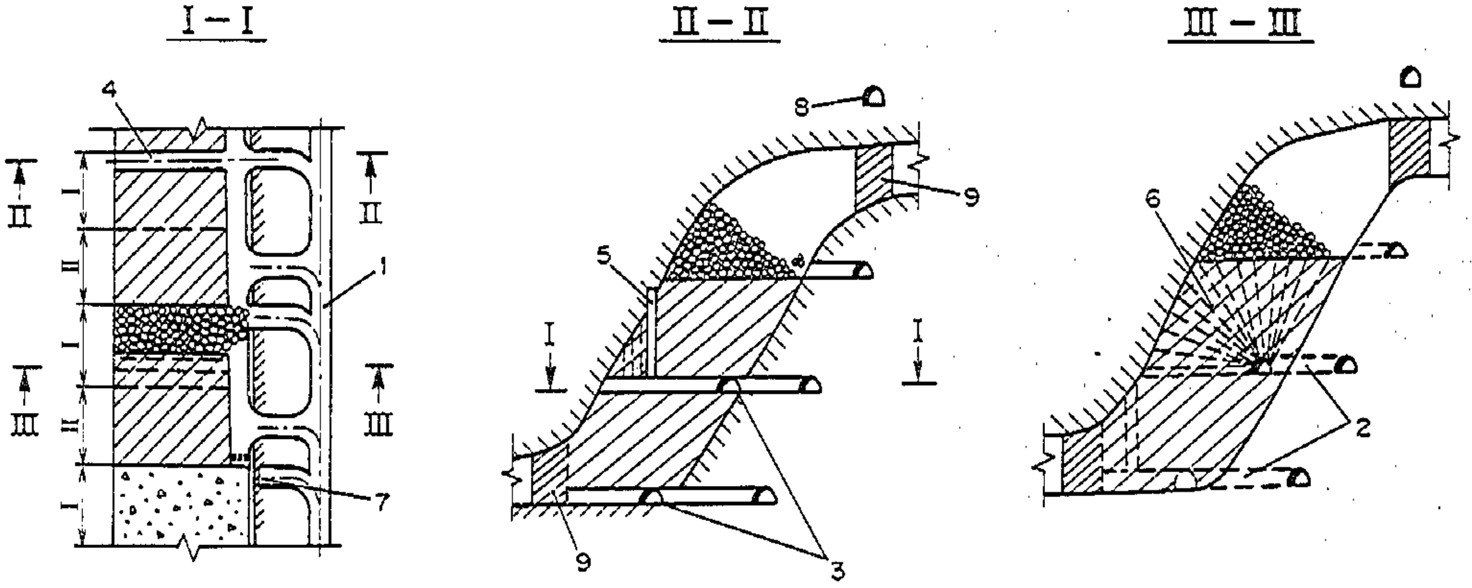
1. panel drift
2. ventilating cross cut
3. stope of incision
4. bench stoping
5. cross cut for drilling
6. drift for filling
7. extraction drift in the footwall

Fig. 2-7-6 **Room and Pillar Mining Method with filling and cut and fill stoping in the primary chambers and Cu content higher than 2.5%**



1. panel drift
2. cross cut for haulage and drilling
3. cross cut for drilling
4. cross cut for loading
5. drift for filling
6. extraction drift in the footwall

Fig. 2-7-7 **Room and Pillar Mining Method (stoping height higher than 16~18m)**



1. delivery drift
2. cross cut for loading
3. drift for drilling
4. cross cut for forming free face
5. slot raise
6. drill hole
7. partition for filling
8. drift for filling
9. partitioning pillar

Fig. 2-7-8 Sublevel (Slicing) Stoping Method (with filling and orebody steeper inclination)

Slicing Mining Method

This mining method is adopted for stratifications with thickness more than 18 m or with dip steeper than 50 degrees where working sections reach 100 m in height. As shown in Fig.2-7-8, an access level and a sublevel are driven in the foot wall and ore body, respectively, at 18~20 m spacing, where mining operation is advanced from the top to the bottom, while drilling direction is upward. Swelled portion of blasted ore is recovered with the gathering loader also in this case and almost all ore is extracted finally at the extraction level driven in the foot wall by means of retreating. The chamber space is filled up in the same way as that of the room and pillar mining method. This method is now being practiced in the East and South Mines.

7-2-2 Pillar Design

Pillar size in Zhezkazgan is determined on the basis of the design standards stipulated in 1984. Applicable conditions for the standards will be a dip of less than 15 degrees, 3~18 m of layer thickness and less than 500 m of rock covering (mining depth). It is, therefore, not applicable to the No.67 district of the South Mine nor to the Annensky district of the East Mine where working places are expected to become deep than 500 m. If applied in these districts, it is sure that ore recovery will decline remarkably. Now the central research is reconsidering parameters for these areas.

Designing (Rib) Barrier Pillar

$$A = (K_H' \cdot \gamma \cdot h \cdot H \cdot L \cdot \Pi_b \cdot K_a / K_{TP} \cdot \sigma_H)^{1/2}$$

A : Rib pillar width (m)
K_H' : Load coefficient on pillar :1
γ : Rock density (t/m^3) :2.6
h : Pillar height, layer thickness or mining height (m)
H : Rock covering above mining place, mining depth (m)
L : Rib pillar spacing between both centers (m) :150
Π_b : Safety coefficient of rib pillar :3
K_a : Influence coefficient by stratiformed dip
K_{TP} : influene coefficient by crack
σ_H : Unaxial compressive strength of grey sandstone (t/m^2)

Designing Room Pillar

$$D = (4K_H \cdot \gamma \cdot h \cdot H \cdot S_{ON} \cdot \Pi_{MK} \cdot K_{\alpha} / \pi \cdot \sigma_H \cdot K_{TP} \cdot K_{NP} \cdot K_{NM} \cdot K_K)^{13}$$

D : Pillar diameter (m)
K_H : Load coefficient on pillar
γ : Rock density (t/m^3) :2.6
h : Pillar height, layer thickness or mining height (m)
H : Rock covering above mining place, mining depth (m)
S_{ON} : Loaded roof area on pillar (m^2) :400
Π_{MK} : Safety coefficient of pillar :2
K_{α} : Influence coefficient by stratiformed dip
π : Circular constant
σ_H : Unaxial compressive strength of grey sandstone (t/m^2)
K_{TP} : influcne coefficient by crack
- * Less cracked grey sandstone :0.63
- * More cracked grey sandstone :0.40

K_{NP} : Coefficient of large waste band such as aleurolite :0.9
K_{NM} : Influence cofficient by blasting :0.85-0.90
K_K : Influence cofficient by contact
- * Grey sandstone :1.0
- * Red sandstone :0.7

Finalizing Parameter

Coefficient will vary depending on panel depth (H) and width (L).

L/H Ratio	1.00	0.66	0.50	0.40	0.33
K_H (Gray Sandstoone)	0.70	0.60	0.55	0.50	0.45
K_H (Red Sandstoone)	0.85	0.80	0.77	0.75	0.72

The following values are given from past records.

Depth (m)	150~200	300	400	500
σ_H (MPa)	200	230	240	245

The calculated examples of pillar size and ore recovery are shown in Table 2-7-6

Table 2-7-6 Examples of Pillar Size and Ore Recovery

Depth (m)	Layer Thickness (m)	2	4	6	8	10	12	14	16	18
	D	4.0	5.0	6.0	6.5	7.0	7.5	8.0	8.0	8.5
200	A	6.0	8.5	10.5	12.0	14.0	17.0	19.5	22.5	25.0
	R	92.7	89.7	87.3	85.4	83.0	79.9	77.0	75.0	71.9
	D	4.0	5.5	6.0	7.0	7.5	8.0	8.0	8.5	9.0
300	A	7.0	9.5	12.0	13.5	14.0	17.0	19.5	22.5	25.0
	R	92.0	88.2	86.3	83.3	81.9	78.7	77.0	73.6	70.6
	D	4.5	5.5	6.5	7.0	7.5	8.0	8.5	9.0	9.5
400	A	8.0	11.0	13.5	15.5	17.0	19.0	20.5	22.5	25.0
	R	90.5	87.9	84.4	82.0	79.9	77.3	74.9	72.3	69.1
	D	4.5	6.0	6.5	7.5	8.0	8.5	9.0	9.0	9.5
500	A	8.5	12.0	15.0	17.0	19.0	21.0	23.0	24.0	25.5
	R	90.1	86.3	83.3	79.9	77.3	74.6	72.0	71.3	68.8

7-3 Mining Cost

When the Zhaman-Aibat deposits are evaluated, mining costs must be considered as a key factor. Mining costs per ton of ore are shown in Table 2-7-4. Actual records of underground mining operation from January to July 1995 show 400 Tenge/ton-ore ($US 6.70). Mining costs reportedly reach 700 Tenge only in July which is nearly double the average which suggests the records may be unreliable. The expenses given are specified only for underground mining at the South Mine and the open pit mining at the North Mine. Itemized costs of drilling, loading and transportation are, however, unknown and are not sufficient for us to argue a possibility or applicability of the cut and fill mining method. Re-examination must be required.

7-3-1 Production Cost at South Mine

An operating budget is worked out by each mine individually and is adopted after approval by the planning manager of the headquarters of Zhezkazgantsvetmet (Table 2-7-7). Construction plans are formulated by each mine although cost estimation are carried out at the headquarters. For example, a shaft development cost is included in the budget of the development division of the headquarters. The mines themselves are in a position to manage the development works but not to control the expenses. The capital expenses and the content of the project are administered and allotted by the planning division of the headquarters.

A breakdown of the actual expenses at South Mine could not be obtained. Only data of last August was available. Administration fees for the headquarters are not included in these operation costs. The scope of the costs includes all activities from exploration in mine to feeding ore into shaft hopper.

Actual records in August 1995 : 413 Tenge/ton-ore*
Plan in August 1995 : 305 Tenge/ton-ore
* over 35% of the expectation.

Actual records of unit cost from January to August 1995;

Electric Power	51,019 MW	21.3 KW/t
Drainage	2.645 million m^3	1.104 m^3/t

Actual output was expected to be about 2.4 million tons corresponding to 60% achievable for 4 million tons of budget.

7-3-2 Operative Expenses at North Mine

Operative expenses from January to July 1995 at North Mine are shown in Table 2-7-8.

Table 2-7-7 Budget of South Mine

(January~July 1995: 3.5 Million Tons of Planned Output)

Unit: Tenge/ ton-ore

Items	Unit Cost	Remarks
Material	18.70	Refund of exploration cost (mining tax: 1% of sales)
Supplies	9.07	Commodity cost using for mining
Engineering service	0.51	
Energy	100.68	Power & fuel expenses
Labor	37.03	Wages, bonus & premium for miners
Filling	14.72	Material, fabricating & transport. costs
Exploration	2.58	Exploration in the mine
Liquidation for drifting cost	4.62	
Depreciation	21.99	
Internal expenses	160.21	Expenses else than mining and drifting, administration, commodity tax, repair
Drifting cost	237.85	
Deduction sold ore	Δ241.07	Value of extracted ore from drifting
Total Operative Costs	366.89	

Table 2-7-8 Comparative Table between Budget and Actual Records at North Mine

(Unit: ×1000 Tenge, Unit Cost: Tenge/Ton-Output)

	Actual Records		Budget	
Items	Amount	Unit Cost	Amount	Unit Cost
Sales	691,731	600,063		
Production Cost	545,227	260.87	540,748	258.73
Profit on Mine Site	146,504	70.10	59,315	28.31

* Output: 2.09 million tons, Delivery: Unknown

	Items of Operation Costs			
Items	Actual	Unit Cost	Budget	Remarks
Exploration	17,672	8.46	16,595	* 1
Operative supplies	58,095	27.80	66,100	* 2
Fuel	63,603	30.43	69,200	
Power	26,461	12.67	28,600	
Heating	18,966	9.07	19,010	* 3
Labor cost	126,322	60.44	105,750	
Insurance premium	47,000	22.49	33,840	
Bonus for elder	6,209	2.97	6,210	
Depreciation	29,995	14.35	29,995	
Repairing cost payable	4,713	2.25	3,586	
Maintenance supplies	93,452	44.71	104,309	
Supplies	1,234	0.59	1,240	Meal to be supplied
Penalty	467	0.22	200	* 4
Maintenance for building	2,150	1.03	2,500	
Internal service, etc.	48,888	23.39	53,613	
Total	545,227	260.87	540,748	

* 1: Mining tax payable to the State Government

*2: Various mining supplies such as explosives

*3: Heated water from boiler plant

* 4: Penalty against bulky ore treatment & environmental contamination

7-4 Outline of Ore Dressing Plant

A flow sheet of the ore dressing process is shown in Fig.2-7-9.

7-4-1 No.1 Ore Dressing Plant

Daily 22,000 tons of ore with 0.9～1.2% of feed grade from underground mining is processed in this ore dressing plant where 37.5%. Cu concentrate can be obtained at 90～92% mill recovery.

Crushing process: Ore is crushed through 3 crushing stages as follows;

Primary crushing : Crusher КМЛ - 900×2 units with over 900 mm feeding size
Secondary crushing : Crusher КМЛ - 200×4 units
Tertiary crushing : Crusher КМЛ - 2,200 T (18.0 mm)×8 units

Grinding process: Ore crushed is ground through 2 milling stages as follows;

Primary grinding: Primary grinding process consists of 12 ball mills (3,200 mm ϕ ×3,100 mmL and 22 m^3 capacity) and 3 units of rod mills (3,200 mm ϕ ×3,380 mmL and 25 m^3 capacity) with a closed circuit combined with a spiral type classifier. Ore is ground to the grain size of 35% under 200 mesh.

Secondary grinding: Secondary milling process consists of 8 ball mills (3,600 mm ϕ × 4,000 mmL and 36 m^3 capacity) and 3 other mills (3,200 mm ϕ ×3,100 mmL). Ore is further ground to the grain size of 65 % under 200 mesh.

Flotation process: A sand/slime flotation method which treats sand and slime independently is applied in this process. Air blowing flotation machines, ФПМ-3.2 (3.2 m^3 capacity of sand flotation) and ФПМ-6.3 (6.3 m^3 capacity of slime flotation) are used.

For the purpose of enhancing Cu grade, rougher concentrate is reground to the grain size of 90～95% under 200 mesh by using 8 ball mills (2,700 mm ϕ ×3,600 mmL) and is cleaned three times. Cu concentrate grade increases 37～38 % at 91.5～92.0% mill recovery. Cu concentrate is mixed with that from No.2 ore dressing plant and transported hydraulically to the smelter and is smelted after thickened, filtered and dried. Other useful minerals contained in the Cu concentrate are Ag, Re and S.

7-4-2 No.2 Ore Dressing Plant

Daily 40,000 tons of ore from open pit and underground mining is transported by dump cars on the track. Crude ore grade is 0.6～1.2 % Cu from which a Cu concentration of 37.5% can be produced at 86～90% mill recovery.

Crushing process: Ore is crushed from over 1,500 mm to under 35 mm through 3 crushing stages as follows;

Primary crushing : Crusher У 3 Т М 1,500/180×2 units with over 1,500 mm feeding size

Secondary crushing : Crusher КСП-2,200 ГР ×8 units
Tertiary crushing : Crusher КМЛ - 2,200 Т × 10 units and inertia type crusher (Impact type crusher?) × 2 units, crushed to under 35 mm

Grinding process: Grinding process is roughly divided into 3 systems

No.1 grinding system (6 lines): Secondary grinding of feed ore is performed in a closed-circuit consisting of ball mill and spiral type classifier (3,000mm ϕ with 2 spirals) after passing through the primary grinding of rod mill with 32 m^3 capacity (in open circuit). Tertiary grinding is carried out in a closed circuit consisting of a cyclone (750 or 1,000mm ϕ) and a ball mill (3,600 mm ϕ ×4,000 mmL and 36 m^3 capacity). After that, ground ore is divided by the cyclone to two flotation lines, slime from the overflow and sand for the underflow.

No.2 grinding system (2 lines): After passing through 2 rod mills with 32 m^3 capacity arranged in parallel (in open circuit), secondary grinding is done in a closed circuit consisting of ball mill (3,600 mm ϕ ×4,000 mmL and 36 m^3 capacity) and spiral type classifier (3,000 mm ϕ with 2 spirals). And then it is subjected to tertiary grinding with cyclones (750 or 1,000 mm ϕ) and ball mills (3,600 mm ϕ ×4,000 mmL and 36 m^3 capacity). The ground ore is then separated into slime and sand flotation lines by cyclones. The overflow goes directly to the slime flotation line while the under flow passes through another ball mill (in a closed with the cyclone) before entering the sand flotation line.

No.3 milling system (1 line) : After primary grinding with 2 rod mills with 32 m^3 capacity arranged in parallel, ore is ground in a closed circuit consisting of ball mill and cyclone. Ground ore is divided by the cyclone to slime and sand flotation lines. The cyclone overflow passes directly to the slime flotation line while the cyclone underflow undergoes tertiary grinding in another ball mill (in a closed circuit combined with the cyclone) before entering the sand flotation line.

Flotation process: Flotation process belongs to each grinding process and consists of 3 lines which are complex and are characterized by applying sand/slime flotation method in general. Sand and slime are separately treated in operation of rougher and cleaning flotation. Rougher concentrate is reground and, then admixed with rougher concentrate from slime line, and Cu concentrate can be given by cleaning. Air blowing flotation machines, ФПМ-16, type ФП М-6.3 and mechanical agitating flotation machines, Ф М Р-6.3 are used. The feed grade is 0.95% and Cu concentrate 36.37% at 88.5~89.0% recovery. Cu concentrate is mixed with that from No.1 ore dressing plant and transported hydraulically to the smelter.

Water consumption at both No.1 and No.2 ore dressing plants is 4.0 m3/ton-ore processed and electric power consumption is 38.0 KWH/ton-ore processed. The plant employs 1,737 operating and maintenance workers and 180 office workers, (1,917 persons in total) (Table 2-3-5).

7-4-3 No.3 Ore Dressing Plant

This ore dressing plant processes complex ore (Cu+Pb+Zn)and began operating in September 1986. Daily treatment ore amounts to 5,000 tons. Ore crushed at the underground crushing plant is secondarily crushed here. After that, it is ground through 3 stages of grinding process and then Pb, Cu and Zn concentrates are obtained through the differential flotation method. Each concentrate is supplied to the smelter after being thickened and filtered.

Crushing process: Ore is crushed through 3 crushing stages as follows;

Primary crushing : Crusher КМЛ - 900×2 units with over 900 mm feeding size
Secondary crushing : Small crusher КМП-22001×2 units by which under 16.0 mm grain size can be given through the closed circuit combined with the screen.

Grinding process:

Primary grinding: Rod mill (3,600 mm φ ×5,500 mmL and 49 m^3 capacity) ×1 unit that constitutes the closed circuit with the spiral type classifier

Secondary grinding: Ball mill (4,500 mm φ ×6,000 mmL and 85 m^3 capacity)×1 unit that constitutes the closed circuit with the cyclone having 1,000mm φ.

Tertiary grinding: ditto

Re-grinding: Ball mill (2,700 mm φ ×3,600 mmL)×3 units

Air blowing and mechanical agitating flotation machines, ФПМ-12.5 and ФПМ-3.2 are used in this flotation process. Concentrate is thickened through a thickener of 18 m diameter and is filtered with 6 units of ЪОУ-40. Water consumption is 3.5 m^3/ton-ore processed and electric power consumption is 40.0 KWH/ton-ore processed. 348 operation and maintenance workers and 63 office workers, (411 persons in total), are employed(Table 2-7-5).

7-5 Future Issues and Suggestion

From the investigation it became clear that no pillar design standards is not finalized in their hand which is to be applied for room and pillar mining method at 500 m or more below the surface, while the mining area of the South Mine planned for the coming year and a part of the Annensky district are already under development rushing into a mining area deeper than 500 m below the surface and they are to be in a position to review the calculation parameters as well as experimental equations which have been applied in their past design. On the other hand, it is also a fact that there is no economic justification for the choice of 2.5% Cu grade as the criterion for applying the cut and fill mining method. For this reason, they have been operating by "from top to down" management system. It is said that the Zhaman-Aibat deposits have mining depth of 500~750 m with a thickness of 10 m or less. When the foregoing pillar design equation is applied, the following pillar size and ore recovery are obtained.

Mining Depth(m)	Layer Thickness(m)	2	4	6	8	10	12
600	D	4.7	5.9	6.8	7.5	8.1	8.6
	A	9.4	13.4	16.3	18.8	21.1	23.1
	R	85.9	79.9	75.3	71.5	68.1	65.1
700	D	4.8	6.0	6.9	7.6	8.1	8.6
	A	10.2	14.4	17.7	20.4	22.8	25.0
	R	84.9	78.6	73.7	69.7	66.1	63.0

Applying the current pillar design equation, ore recovery is expected to be 70% or less at 600 m or deeper depth with 10 m or more layer thickness when the primary mining operation is only considered. In consequence, recovery can not be improved unless pillars are recovered. It thus becomes important to fill up chamber space after they are primarily mined out to recover the pillars. The question then arises as to strength of filling material, filling method and ore recovery method. At present, it is prohibited in Zhezkazgan mines to enter into an area when secondary mining is being operated. For this reason strength of the filling material will be enough at max. 4 MPa. In addition, the filling level is driven into the hanging wall in order to fill the chamber space more compactly, while the extraction level is being driven into the foot wall aiming at recovering ore more safely. Such method is practiced for 2.5% or higher Cu grade.

In order to select a suitable mining method to be applied at Zhaman-Aibat deposits the following items must be investigated and examined in the coming year.

1) To obtain physical properties of host rocks
2) To study the current pillar design equation and re-examine the parameters
3) To investigate various expenses for secondary mining
4) To investigate filling material (kind of materials, strength and prices)
5) To examine procurement methods of the filling material
6) To study heavy-duty machines for mining

It is needless to say that the profitability must be considered to plan the deposit development, to which a lot of construction expenses must be required for infrastructure, ore dressing plant and mining field. It is difficult to get the data for development expense at existing operating mine. These expenses must be investigated at the headquarters or engineering division.

1) Mine development : Constructing shaft, driving structural level, installing skip hoist
2) Ore dressing plant : Flow sheet, equipment, tailing dam
3) Road and rail way : Transporting concentrate
4) Power and water supply : Transmission line, sub-station and water supply pipe line
5) Company's housing quarters and mine site building
6) Tax and accounting system