for chemical assay. These samples were first semi-quantitatively analyzed for major elements, such as Cu, Pb, Zn and Ag. According to the result of the analysis, samples were selected for quantitative analysis at the laboratory of Zhezkazgangeologiya. Unit core length through the mineralized zone for analysis range from 0.5m to 1.0m in general. Composite samples combining several unit core samples were analyzed for minor elements such as Re,Os,Cd,P,Sb,V,Hg and Se, as well as total and sulphide sulphur. The analytical results were recovered on hand written chemical analysis data sheets labeled with drill hole numbers, depth of samples and other particulars.

A construction of the Zhaman-Aibat data-base has been started by the Japanese Survey team in the Phase I with spread-sheet software for the ore reserve estimation. The data base include approximately 6,700 analytical data from 707 drill holes.

1-6 Ore Reserve Estimation

For the second straight year, the ore reserve estimation (i.e. geological resources estimation) for the Eastern Orebody and main part of the Central Orebody were calculated by the Japanese Survey Team. Total amount of drilling has reached 825 exploration drillholes and core samples.

The Japanese Survey Team started constructing a computer data-base namely Zhaman-Aibat data base with spread-sheet software for the ore reserve estimation of the Zhaman-Aibat deposit. The data base accumulated 3,851 analytical data from 402 drillholes in this year.

The Zhaman-Aibat ore deposits are categorized into four types of ores according to their relative differences in major ore mineral contents. They are 1) copper ore comprising mainly chalcocite and bornite with minor chalcopyrite, 2) complex ores, mainly chalcocite, bornite, galena and sphalerite with minor chalcopyrite, 3) lead-zinc ores, mainly galena, sphalerite, chalcocite and bornite, and 4) silver ores, mainly chalcocite, bornite and native silver, with minor chalcocite. The cut-off grades of these ore types are set at 0.4% Cu for the copper ores, 0.8%Pb+Zn and 0.3%Cu for the complex ores, 1.1%Pb+Zn for the lead-zinc ores and 5g/tAg for the silver ores (Table 2-1-7).

The Eastern Orebody and the main part of the Central Orebody were selected for the ore reserve estimation in this year's study.

1) Calculation methods by the Zhezkazgangeologiya

The method by the Zhezkazgangeologiya was reported in the Phase I report. The points of the calculation procedure will be summarized below :

(1) The delineation of ore zone is basically determined by the geological and mineralogical data and chemical assay data.

(2) According to the resolution of the "State Committee for Reserves" of the State Committee for Geology of the Republic of Kazakhstan, the cut-off grade for each ore deposit is determined.

(3) Correlation of ore-bearing horizon between drill holes is determined stratigraphycally. On each ore body, ore deposit zones are determined by cut-off grade, centering on the ore horizon 4-I in the Zhezkazgan Formation.

(4) A cut-off thickness of ore is incorporated in the determination of ore/waste. Even if the thickness of ore is less than 3 meter and the grade of ore is high, determination of ore/waste will be made by referring to the value of ore thickness(m) × ore grade(%).

(5) In case of the presence of inter band between ore layers and accumulated thickness of them is not exceed 4 meters, and the weighed average of ore is higher than cut-off grade, the zone will be determined to be as ore not waste.

(6) The average grade and length of ore section in each hole are principally applied to the polygon formed by the perpendicular bisectors of the lines jointing the hole to neighboring holes. The lateral limit of each orebody is defined by lines connecting (1) the midpoints between pairs of the drill holes spanning the ore boundary or (2) points 50～150m outside the outer most hole where no drill hole exists beyond this hole.

The estimation is carried out with the following procedure.

(1) A conventional area method is used for the ore reserve estimation

(2) The average grade of the unit ore section is calculated by a length weighted average of the grades of the analytical units. Thickness of ore zone is determined both by cut-off grade and by the judgment of the ore reserve calculator in charge.

(3) The average grade of ore block composed by several unit ore sections is calculated by the length weighted average of grade of each drill hole. The average thickness is calculated by the arithmetic average by the thickness of each drill hole.

(4) The volume of each block is estimating by multiplying the ore area by the average thickness of ore block.

(5) The weight of ore is estimated by multiplying the volume of ore block by average specific gravity (2.600 ton/m^3, measured average).

(6) The metal amount is calculated by multiplying ore weight by average ore grade.

2) Calculation method by the Japanese survey team.

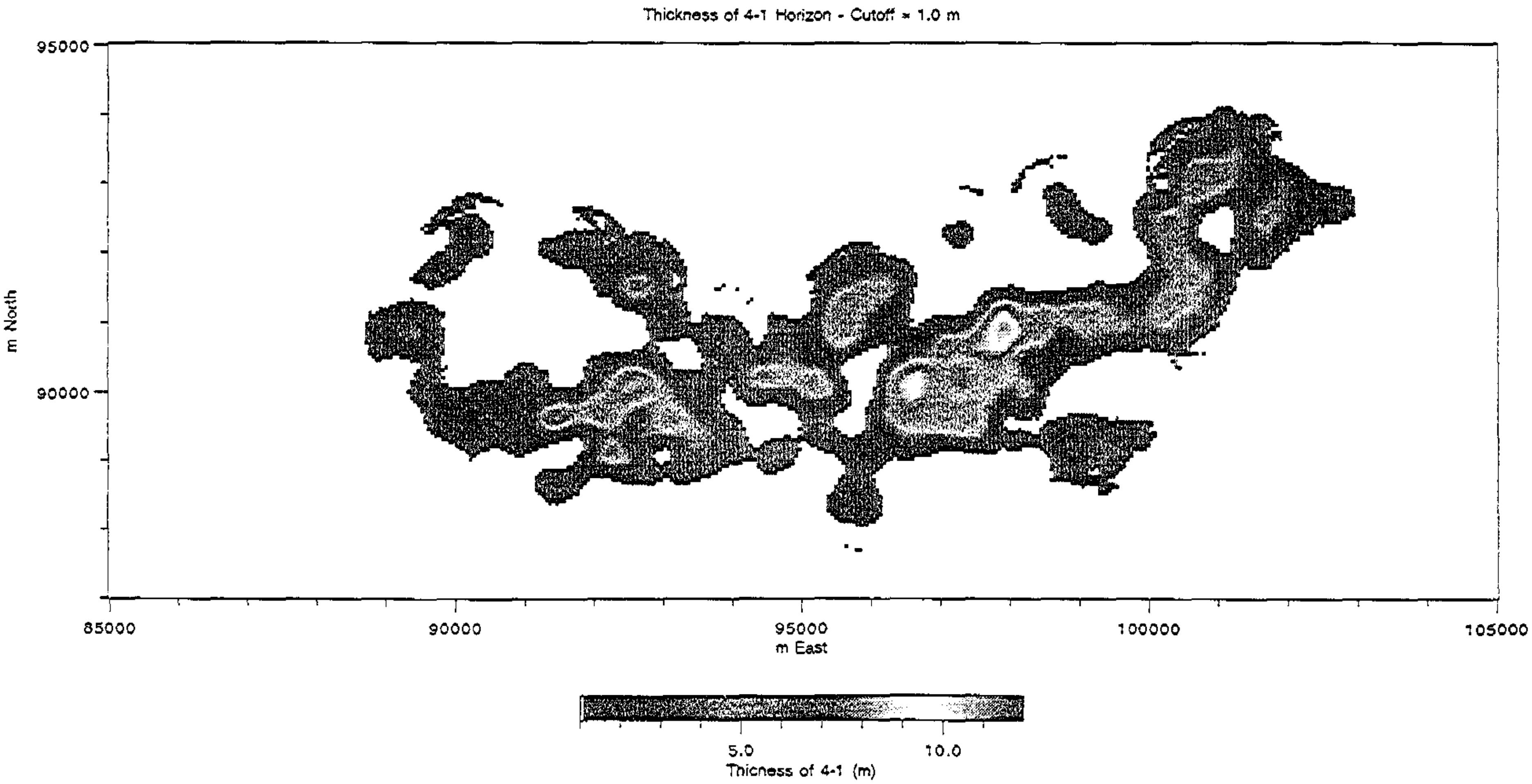


Figure 2-1-11 Contour Map of Ore Thickness(m) of Horizon 4-I in the Zhaman-Aibat Ore Deposit

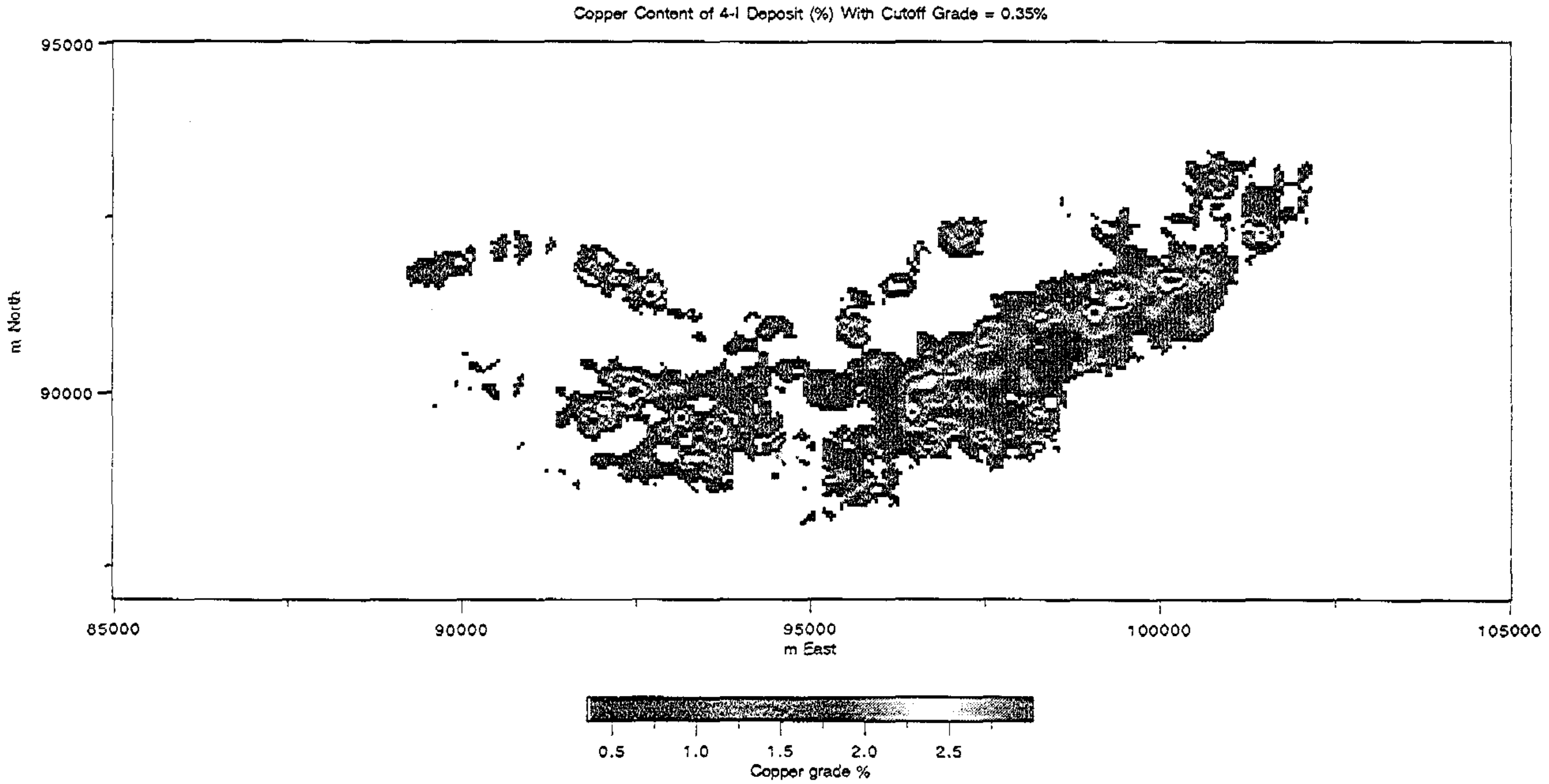


Figure 2-1-12 Contour Map of Copper Grade (%) of Horizon 4-I in the Zhaman-Aibat Ore Deposit

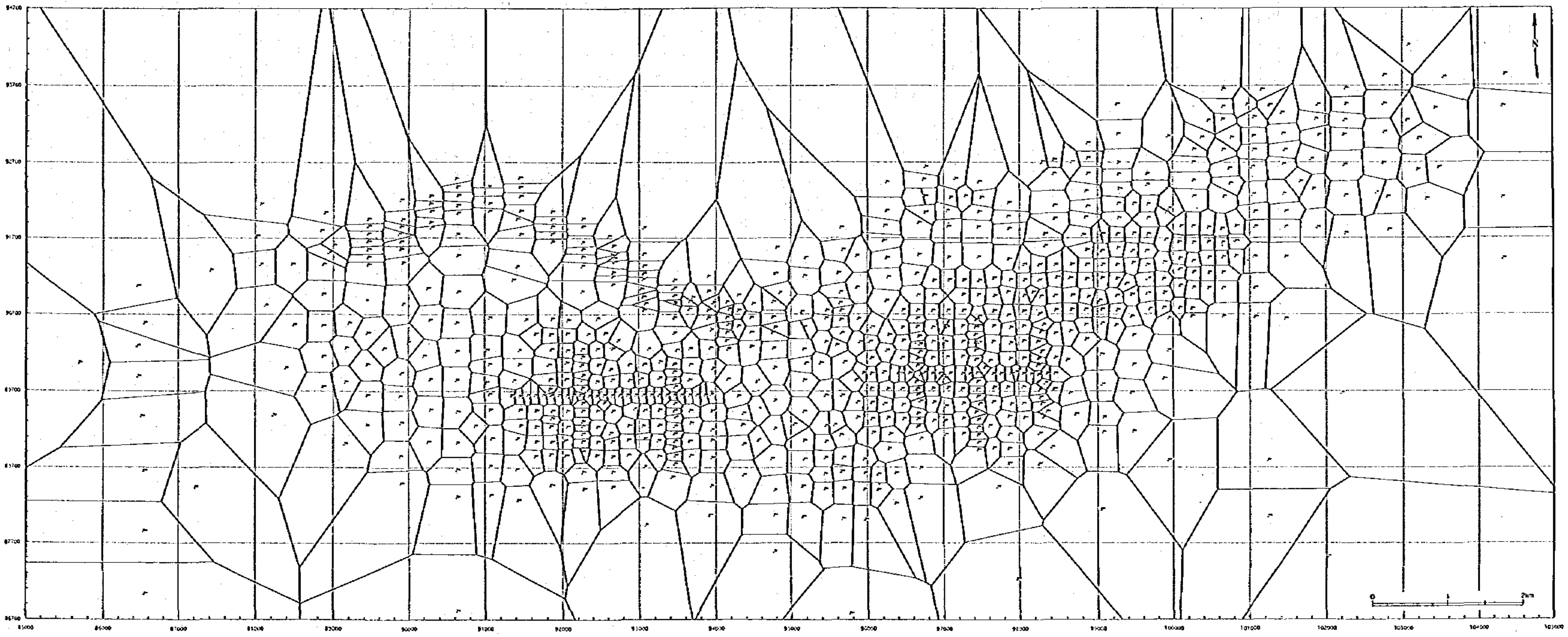


Figure 2-1-13 Polygonal Sub-Blocks of the Zhaman-Aibat Ore Deposit

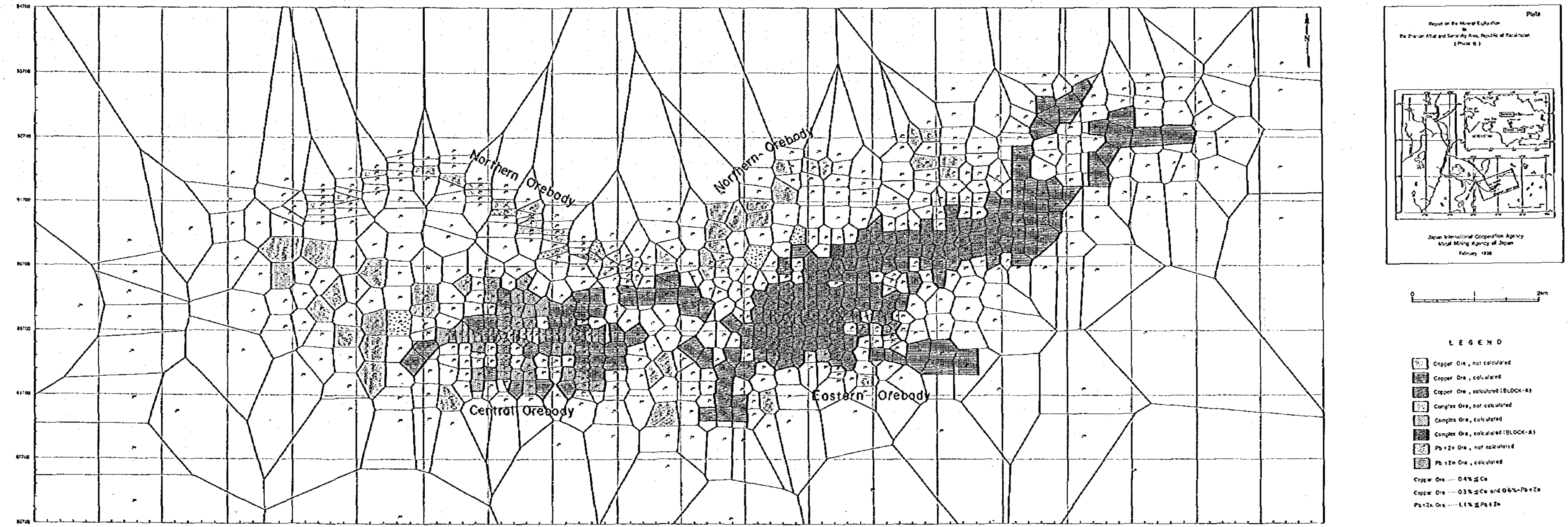


Figure 2-1-14 Interpretation Map for the Ore Reserve Estimation of the Eastern and Main Part of the Central Orebody

Table 2-1-10 Geological (In-Situ) Resources Estimation on the Eastern Orebody of the Zhaman-Aibat Ore Deposit

		Area (sq.m)	Thickness (m)	Ore weight (t)	Cu (t)	Ag (t)
Japanese-Survey Team (1995,96)	Eastern Orebody excect BLOCK-A	5,381,676	4.96m	69,387,490	719,677 1.04%	823.75 11.87g/t
	BLOCK-A	3,365,326	5.45m	47,699,892	876,833 1.84%	438.55 9.19g/t
	Total	8,747,002	5.15m	117,087,382	1,596,560 1.36%	1,262.30 10.78g/t
Zhezkazgan-geologiya (1994)	Eastern Orebody excect BLOCK-A	5,410,699	4.07m	57,311,798	671,590 1.17%	812.18 14.17g/t
	BLOCK-A	3,447,426	5.18m	46,429,900	835,700 1.80%	411.80 8.87g/t
	Total	8,858,095	4.50m	103,741,698	1,507,290 1.45%	1,223.98 11.80g/t

Copper Ore

		Cut-Off Grade				
	Total	1.0%	1.5%	2.0%	2.5%	3.0%
Area (×1000 ㎡)	8,747	5,178	3,286	1,774	1,252	826
Tonnes(×1000)	117,087	64,308	37,283	20,283	12,278	8,383
Cu(%)	1.36	1.94	2.44	3.08	3.64	4.07
Pb(%)	0.05	0.07	0.10	0.09	0.08	0.10
Zn(%)	0.01	0.02	0.02	0.02	0.00	0.01
Ag(g/t)	10.78	15.66	20.26	27.78	32.36	34.09
Cu Metal(×1000ton)	1,597	1,248	921	624	447	341
Pb Metal(×1000ton)	69	47	39	18	9	8
Zn Metal(×1000ton)	21	14	6	5	0	0
Ag(ton)	1,262	1,007	766	563	397	286
Thickness(m)	5.15	4.78	4.43	4.40	3.77	3.90

Table 2-1-11 Geological (In-Situ) Resources Estimation on Main Part of the Central Orebody of the Zhaman-Aibat Ore Deposit

	Area (sq.m)	Thickness (m)	Ore weight (t)	Cu (t)	Pb (t)	Zn (t)	Ag (t)
Cu Ore	1,696,332	3.97	17,505,852	328,542	53,523	12,825	184.97
				1.88%	0.31%	0.07%	10.57g/t
Complex Ore	1,172,863	5.77	17,599,690	271,169	311,434	57,289	192.59
				1.54%	1.77%	0.33%	10.94g/t
Total	2,869,194	4.71	35,105,542	599,711	369,957	70,114	377.56
				1.71%	1.04%	0.20%	10.75g/t

Total

	Cut-Off Grade					
	Total	1.0%	1.5%	2.0%	2.5%	3.0%
Area (×1000 m^2)	2,869	1,606	1,214	747	523	328
Tonnes(×1000)	35,106	18,549	15,132	10,765	7,923	5,076
Cu(%)	1.71	2.67	3.02	3.58	4.06	4.80
Pb(%)	1.04	1.16	1.11	0.95	0.99	0.79
Zn(%)	0.20	0.20	0.22	0.24	0.30	0.03
Ag(g/t)	10.75	14.54	15.27	16.41	18.78	22.47
Cu Metal(×1000ton)	600	496	456	385	322	244
Pb Metal(×1000ton)	370	215	168	102	78	40
Zn Metal(×1000ton)	70	38	34	26	23	1
Ag(ton)	378	270	231	177	149	114
Thickness(m)	4.71	4.44	4.79	5.54	5.82	5.95

For the second straight year, Japanese survey team also carried out the ore reserve estimation and the results were correlated to the results by the Zhezkazgangeologiya. The polygon method was incorporated to the estimation. The procedure of the estimation was summarized below :

(1) The basic data, such as thickness of ore and grade of ore are inputted to the data base both by the Japanese survey team and by the Zhezkazgangeologiya.

(2) The ore block is determined by the polygon method and the perpendicular bisectors of the lines determined on each ore block.

(3) In order to correlate the results of the ore reserve estimation, the outline of the orebodies delineated by the Zhezkazgangeologiya were referred.

(4) The ore reserves and metal amount are calculated on each polygon area. The average specific gravity is determined as 2.600 which is the same figure used by the Zhezkazgangeologiya.

(5) In case of the Central Orebody, ores are classified into two types, namely copper ore (Cu) and complex ore (Cu+Pb+Zn), considering with future mining.

Ore thickness(m) and copper grade(%) of Horizon 4-I in the Zhaman-Aibat ore deposit are shown in Figure 2-1-11 and 2-1-12 and polygonal sub-blocks are represented in Figure 2-1-13.

The results of the ore reserve estimation of the Eastern Orebody and the main part of the Central Orebody are shown in Tables 2-1-10 and 2-1-11. The reserves for different copper cut-off grade are also presented. The estimation results and interpretation on each polygonal sub-blocks are illustrated in Figure 2-1-14. The summary of the results are follows ;

① The Eastern Orebody

The area of the Eastern Orebody calculated by the Japanese survey team is 8,747,002m^2 (98.7% of 8,858,095 m^2 by the Zhezkazgangeologiya) and the average thickness is 5.15m (114.4% of 4.50m). Then the conventional value of density is 2.600 (same value used by the Zhezkazgangeologiya), the total ore reserve is calculated as 117,087,382tons (112.3% of 103,741,698tons). The total metal amounts are estimated as 1,596,560tonCu (105.9% of 1,507,290tonCu), 1,262.30tonAg (97% of 1,223.98 tonAg), respectively. Then the average copper and silver was calculated as 1.36% Cu (94% of 1.45% Cu) and 10.78g/tAg (91% of 11.80g/t Ag), respectively.

It can be seen that overall trends obtained by the Japanese survey team were larger in ore reserve and lower in ore grade due to the discrepancy between the Japanese survey team emphasize and observe the rules of the ore reserve calculation, and that the Zhezkazgangeologiya attaches an importance to the judgment of the chief of ore reserve calculator in charge. However there is no serious problem in this stage of geological resources estimation.

② The Central Orebody

The results are calculated by the Japanese survey team indicates that total area is 2,869,194 m^2 and the average thickness is 4.71m. Then the conventional value of density is 2,600, the total ore reserve calculated 35,105,542tons and total metal amounts were estimated to be as 599,711tonCu, 364,957tonPb, 70,114tonZn, 377.56tonAg, respectively. Thus, the average metal contents are 1.71%Cu, 1.04%Pb, 0.20%Zn, 10.75g/tAg, respectively.

Among these ore reserve, complex ore which is representative ore type in the Central Orebody is estimated to be as 17,599,690tons and the average thickness 5.77m, with an average metal contents of 1.54%Cu, 1.77%Pb, 0.33%Zn, 10.94g/tAg, respectively.

In the Central Orebody, various kinds of ore types, such as copper ore, complex ore, lead-zinc ore, are accumulated and shows complex zonation.

The Zhezkazgangeologiya calculates on each ore type in the same drill hole, even the thickness is so thin. However it is not realistic in future mining to mine so thinner ore and so various ore types at the same time. It is concluded that the results by both calculations are not comparable.

Chapter 2 Drilling Survey

2-1 Summary of Survey

For the purpose of confirming the ore stratigraphy and mode of occurrence of the ore deposits and of sampling the ore deposit, one hole, MJK-1 was drilled (Figure 2-2-1).

Mineralized core samples recovered from this drillhole were used for the ore dressing test. The location (X,Y), elevation (m), drillhole inclination and drilled length(m) are shown in the table below;

Location of Diamond Drilling Holes

DDH Number	Coordinates		Elevation above sea level	Inclination	Hole Depth
	X	Y	(m)		(m)
MJK-1	96.570	90.210	357.04	Vertical	650.5

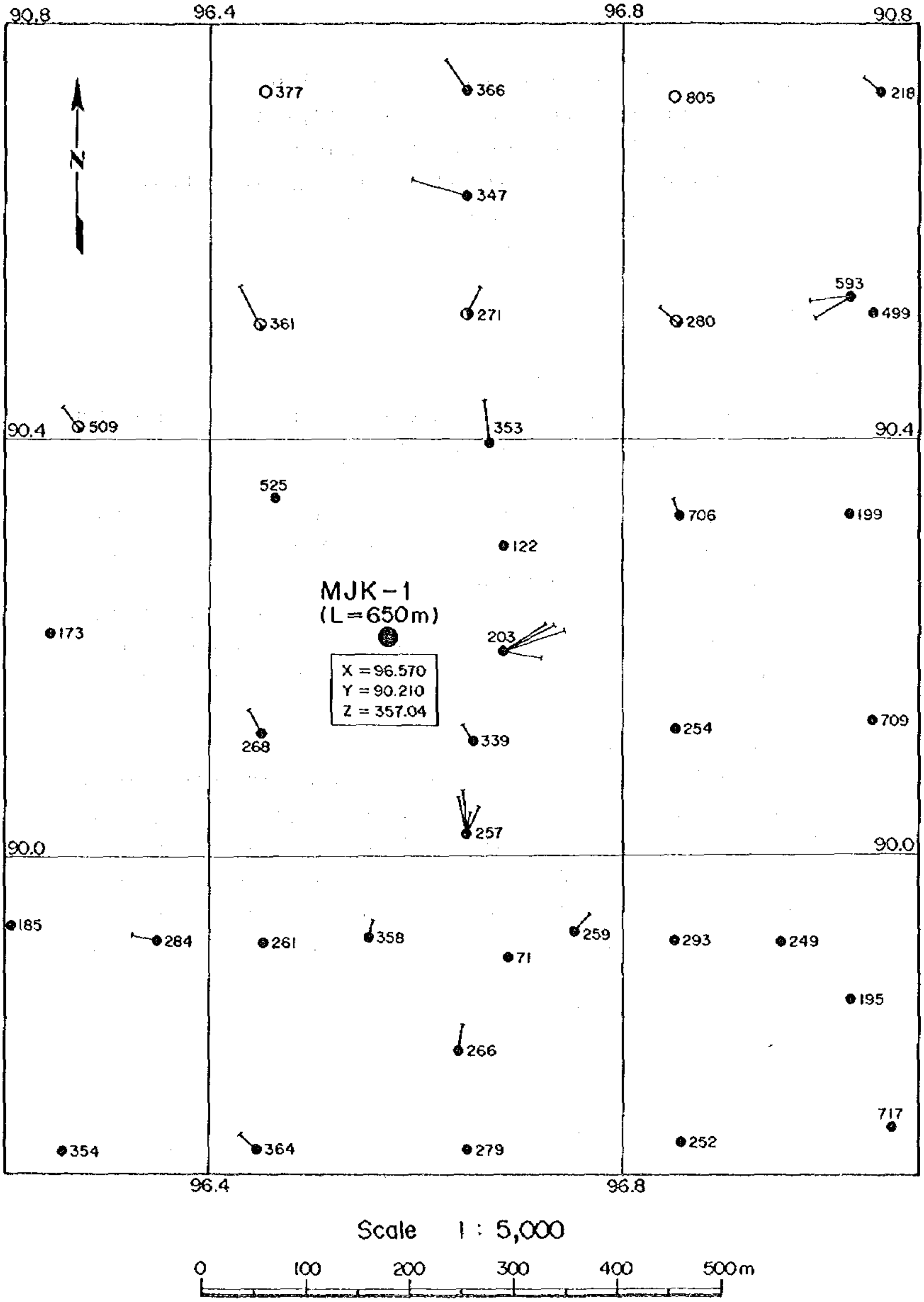


Figure 2-2-1 Location Map of Drill Hole "MJK-1"

The drilling schedule in the 1995 campaign is summarized below;

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

2-2 Survey Method

① Drill progress

The drill progress of MJK-1 is shown in Appendix 2. The itinerary of the drilling engineer of the Japanese survey team is shown below.

Travel (from Tokyo to Zhezkazgan) : July 30, 1995 - August 2,1995
Preparation and agreement : August 2,1995 - August 11,1995
Site survey and management of drilling operation : August 12,1995 - August 22,1995
Data documentation : August 22,1995 - August 23,1995
Dismantle : August 24,1995 - August 28,1995

② Drilling works and drilling team

The drilling works in the Zhaman-Aibat Area are being conducted by the drilling team of the Zhezkazgangeologiya under the supervision of the Japanese survey team. The duration of drilling was 14 days, from August 13, 1995 to August 26,1995.

The drillhole was completed by one drilling rig manufactured in the former USSR. A 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 in the exploration camp.

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 (truck, bulldozer, etc.)	4
Electrician	1
Generator technician	1
Cook	1
Assistant cook	1
Staff for construction/dismantle	

③ Transportation of drilling equipment and water

Drilling equipment for MJK-1 in this year's campaign and consumables including diamond bits are listed in Appendices 3 and 4, respectively. At the drilling site, the drilling rig, drilling pumps, derrick and drilling equipments were transported by two bulldozers. Water for drilling was transported from water wells located 7km to the north to the drilling site by tanker with 5m^3 capacity. The total time required for transportation of drilling water accumulated to 83 hours and 30 minutes and the total volume of water was estimated as 252m^3. Electric power was supplied to the MJK-1 drilling site by an electric generator in the exploration camp via a transformer located 500m from the site.

④ Drilling method

MJK-1 was drilled by conventional methods to 38.3m and the wire-line drilling method was used from 38.3m to 650.5m.

The casing program is detailed below :

(depth)
surface to 4m : 112mm ϕ cemented carbide bit with 108mm ϕ casing, by conventional drilling method
4~38.3 m : 93.3mm ϕ cemented carbide bit with 89mm ϕ casing, by conventional drilling method
38.3~650.5 m : 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 was no interruption by troubles, 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 was 52.0m and the overall core recovery as a percentage of total length of cores was calculated to be as high as 98.5%.

Drilling progress chart and summary of drilling results are shown in Appendices 2 and 5, respectively.

2-3 Survey Results

The geological core logging (scale 1／200) of MJK-1 drillhole is shown in Appendix 6 and detailed geological core logging of the mineralized zone and vicinities are shown in Figure 2-2-2.

2-3-1 General Geology of Hole MJK-1

0～238.0m	: Reddish brown siltstone with inter layers of reddish brown sandstone (thickness 0.1～5m). 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.
403.20～403.45m	: Greenish-grey, fine-grained sandstone. slightly calcareous, "grey sandstone" appeared.
403.45～438.8m	: 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.
597.6～608.27m	: Dark grey～grey, fine～medium grained sandstone, laminated, mineralized zone (chalcocite≫bornite>galena, chalcopyrite(?), pyrite).
608.27～609.30m	: Light brown～light greenish grey "Raimundo" conglomerate

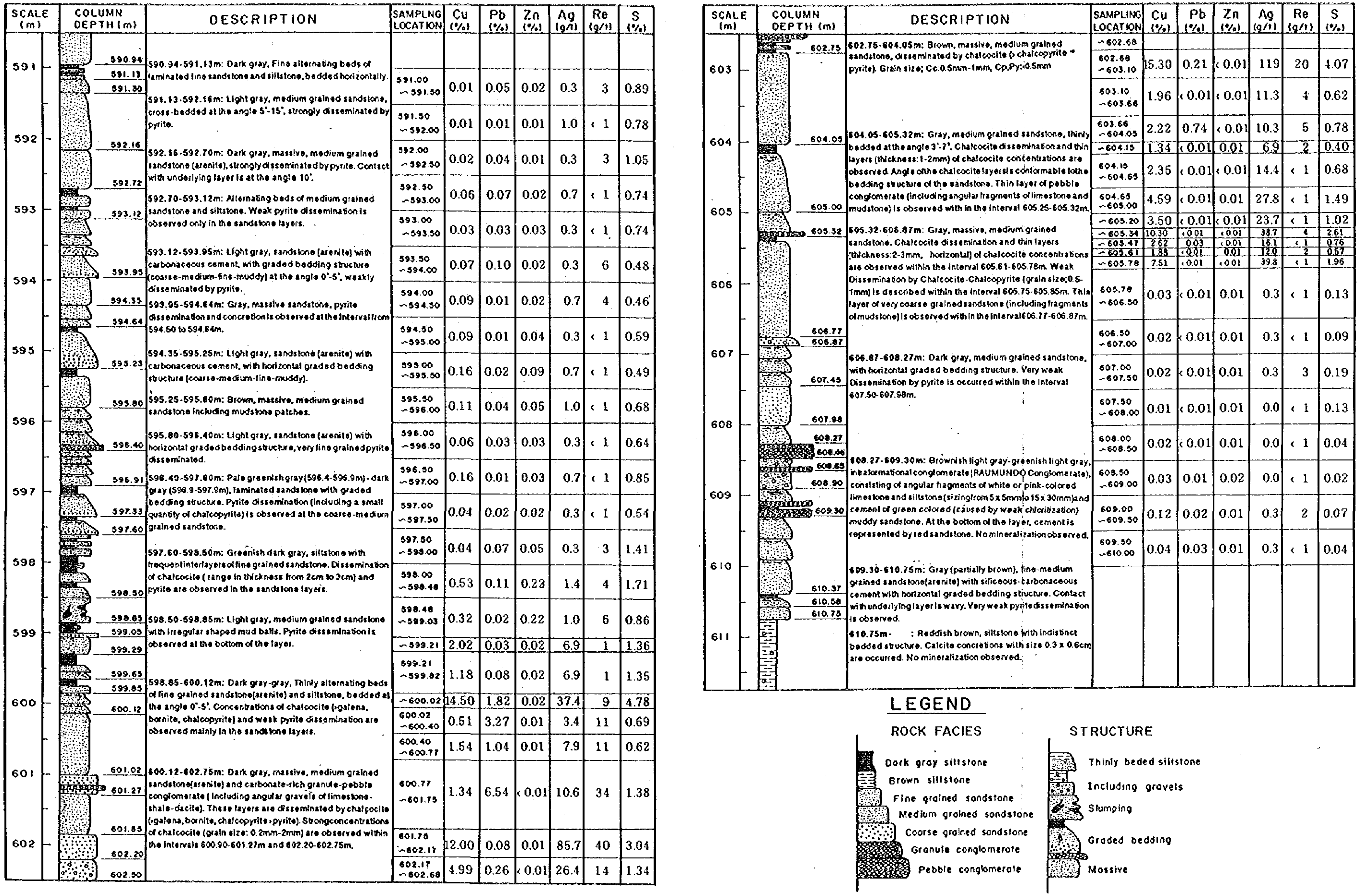


Figure 2-2-2 Detailed Geological Logging for Mineralization Zone of Drill Hole "MJK-1" (1/50)

DESCRIPTION

590.94-591.13m: Dark gray, Fine alternating beds of laminated fine sandstone and siltstone, bedded horizontally.

591.13-592.16m: Light gray, medium grained sandstone, cross-bedded at the angle 5°-15°, strongly disseminated by pyrite.

592.16-592.70m: Dark gray, massive, medium grained sandstone (arenite), strongly disseminated by pyrite. Contact with underlying layer is at the angle 10°.

592.70-593.12m: Alternating beds of medium grained sandstone and siltstone. Weak pyrite dissemination is observed only in the sandstone layers.

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.

593.95-594.64m: Gray, massive sandstone, pyrite dissemination and concretion is observed at the interval from 594.50 to 594.64m.

594.35-595.25m: Light gray, sandstone (arenite) with carbonaceous cement, with horizontal graded bedding structure (coarse-medium-fine-muddy).

595.25-595.80m: Brown, massive, medium grained sandstone including mudstone patches.

595.80-596.40m: Light gray, sandstone (arenite) with horizontal graded bedding structure, very fine grained pyrite disseminated.

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.

597.60-598.50m: 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.

598.50-598.85m: Light gray, medium grained sandstone with irregular shaped mud balls. Pyrite dissemination is observed at the bottom of the layer.

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.

600.12-602.75m: 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.

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

DESCRIPTION

602.75-604.05m: Brown, massive, medium grained sandstone, disseminated by chalcocite (> chalcopyrite = pyrite). Grain size; Cc:0.5mm-1mm, Cp,Py:<0.5mm

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.

605.32-606.87m: 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. This layer of very coarse grained sandstone (including fragments of mudstone) is observed with in the interval 606.77-606.87m.

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.

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.

609.30-610.75m: 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.

610.75m- : Reddish brown, siltstone with indistinct bedded structure. Calcite concretions with size 0.3 x 0.6cm are occurred. No mineralization observed.

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	1.88	< 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	0.04

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.

609.30～605.50m(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.

By the results of MJK-1 drill, the stratigraphy of Horizon 4-I and the type of mineralization were confirmed(Figures 2-2-3 and 2-2-4)

2-3-2 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-I in the Zhezkazgan Formation and is summarized below:

Depth : 598.00m～605.78m
Depth of mineralization : 7.78m
Metal content : 3.78%Cu, 1.17%Pb, 0.03%Zn, 22.7g/tAg, 11.2g/tRe
Ore minerals : chalcocite≫bomite>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.

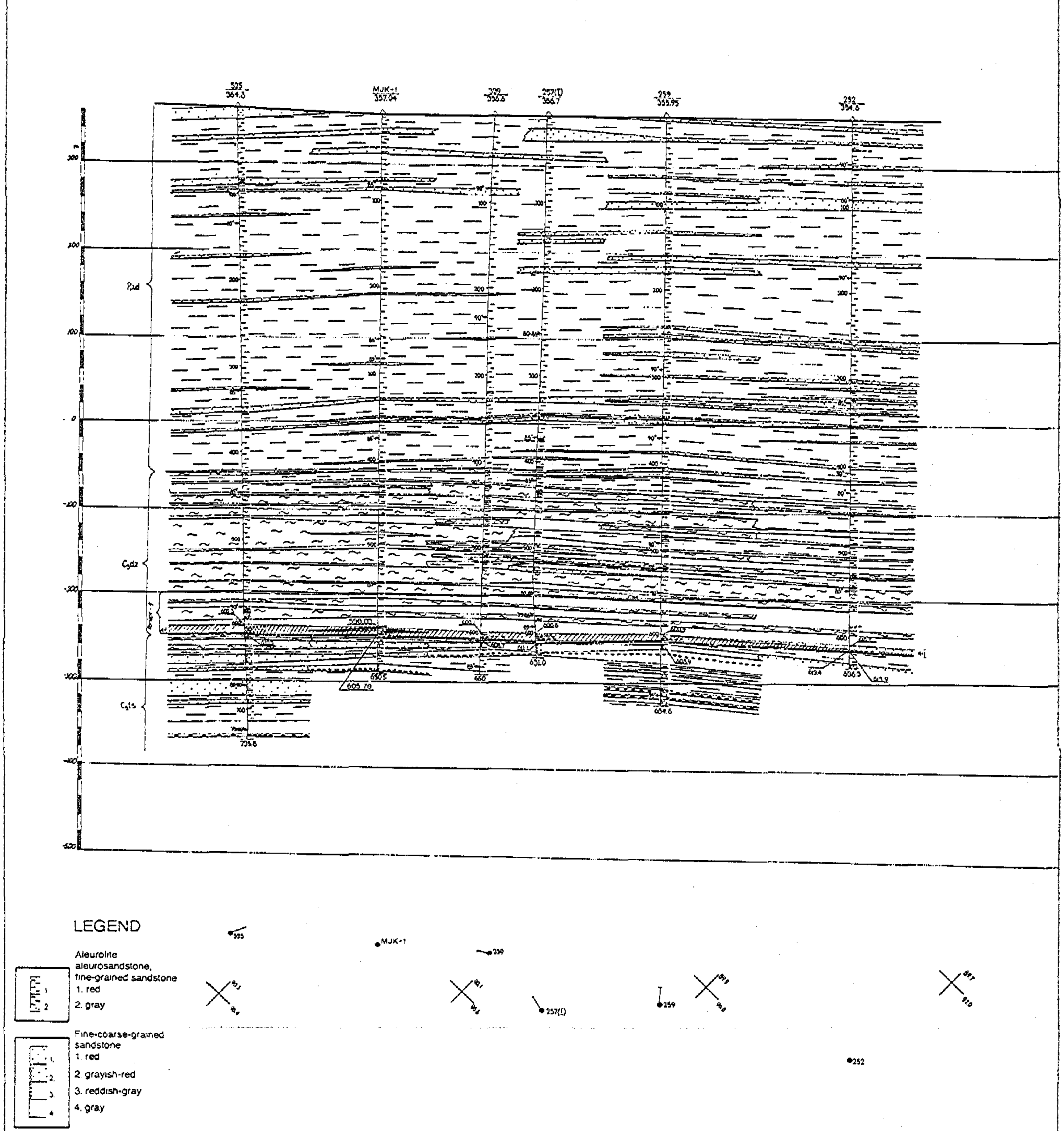
2-4 Laboratory Tests

2-4-1 Chemical Analysis

(1) Chemical assays of MJK-1 core

① Sampling procedure

SCHEMATIC GEOLOGICAL CROSS-SECTION (525-MJK-1-339-257(I)-259-252)



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-I	balance
339	600.1	608.7	8.6	8.6	100	2.76	-	-	0.70	8.10	0.80	copper	4-I	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-I 4-I	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-I	balance
252	610.7	615.4	4.7	4.4	94	0.39	-	-	0.08	1.58	0.09	copper	4-I	off-balance in the roof
	615.4	615.9	0.5	0.5	100	3.04	-	-	0.33	17.0	0.75	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	4-1	balance

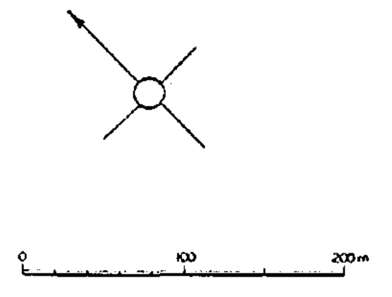
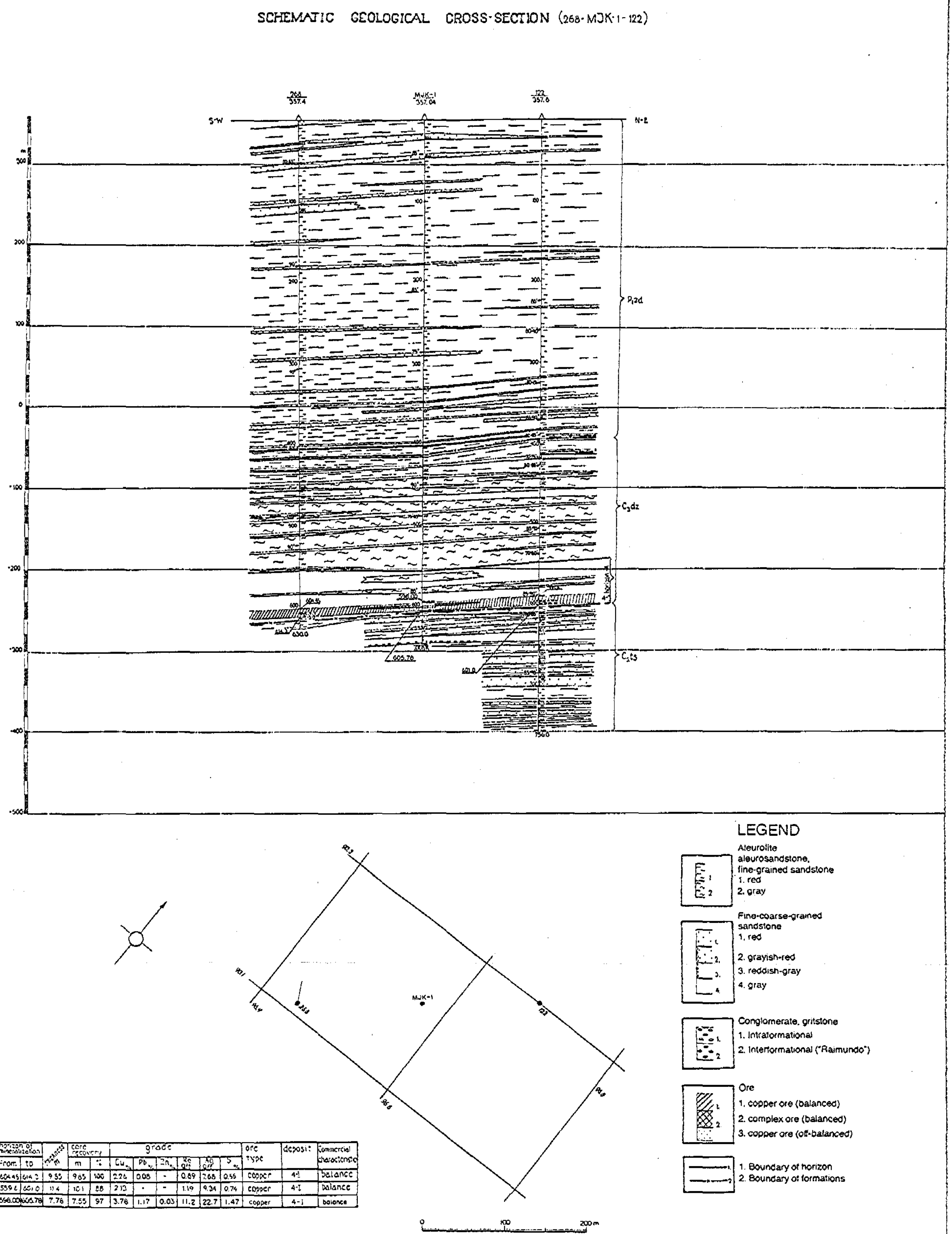


Figure 2-2-3 Geological Cross-Section along Line DDH525(Northwest)- DDH252(Southeast)



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	614.0	9.55	9.65	100	2.26	0.08	-	0.89	7.68	0.55	copper	4-I	balance
122	559.6	601.0	11.4	10.1	88	2.13	-	-	1.19	9.34	0.74	copper	4-I	balance
MJK-1	596.00	605.78	7.76	7.55	97	3.78	1.17	0.03	11.2	22.7	1.47	copper	4-I	balance

Figure 2-2-4 Geological Cross-Section along Line DDH268(Southwest)- DDH122(Northeast)

76 samples were taken at the depth 435m to 642.3m including dense sampling of the mineralized zone of MJK-1 (Appendix 6). In order to compare the chemical 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;

435～585m : 31 samples, continuous sampling with 0.3m sampling interval.
591.0～598.0m : 14 samples, continuous sampling with 0.5m sampling interval
598.0～605.78m : 21 continuous samples in the mineralization zone, split sampling interval ranges 0.13～0.98m due to the lithology and mineralization.
605.78～610.0m : 8 samples, continuous sampling with 0.5m sampling interval.
621.00～642.0m : 2 samples, spot sampling.

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

elements	methods			detection limits	upper limit
Au	: Fire Assay	AAS	(ppb)	5	10000
Ag	: Aqua-Regia digest	AAS	(oz/ton)	0.01	20.0
	Fire Assay -	Gravimetric	(g/ton)	3	10000
Cu	: Aqua-Regia digest	AAS	(%)	0.01	100.0
Pb	: Aqua-Regia digest	AAS	(%)	0.01	100.0
Zn	: Aqua-Regia digest	AAS	(%)	0.01	100.0
Re	: Neutron Activation Analysis		(ppm)	1	10000
FeO	: Titration		(%)	0.01	100.0
Fe (total)	: Titration		(%)	0.01	100.0
S (sulphide)	: Gravimetric		(%)	0.01	100.0
S (sulphate)	: Gravimetric		(%)	0.01	100.0
S (element)	: Gravimetric		(%)	0.01	100.0

② Result of Assay

The assay result of 80 samples are shown in Appendix 7. In the main mineralization zone, the assay results revealed that the highest copper content was 15.3% and the lowest 0.32%, The weighed average Cu content was 3.78% with the thickness of 7.78 meters. Pb assay results showed that the highest content is 6.54% and the lowest is lower than 0.01% and weighted average Pb content in the mineralization zone was 1.17%. On the other hand, the highest Zn content is 0.22% and the lowest was lower than 0.01% weighted average Zn content was 0.03%. The highest Ag content was as high as 118.6g/t and weighted average Ag content was 22.7g/t.

(2) Whole rock analysis

5 representative samples taken at the depth from 440m to 606.5m, were assayed for whole rock analyses. All samples are correlated to "grey sandstone". Assay results are shown in Appendix 8. 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.

2-4-2 Petrological and Mineralogical Examinations

1) Microscopic observations of this 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 12 and 13.

Specimen 95-TS-06 was taken at the depth of 601.5m in the middle part of the mineralization zone. 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. 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.

2) Microscopic observations of polished sections.

9 specimens from the mineralization zone of MJK-1 and one (1) 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

Appendices 9 and 10.

The identified minerals are listed below ;

primary minerals : chalcocite series minerals; such as chalcocite, digenite, djurleite, bornite, galena, chalcocite, 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.

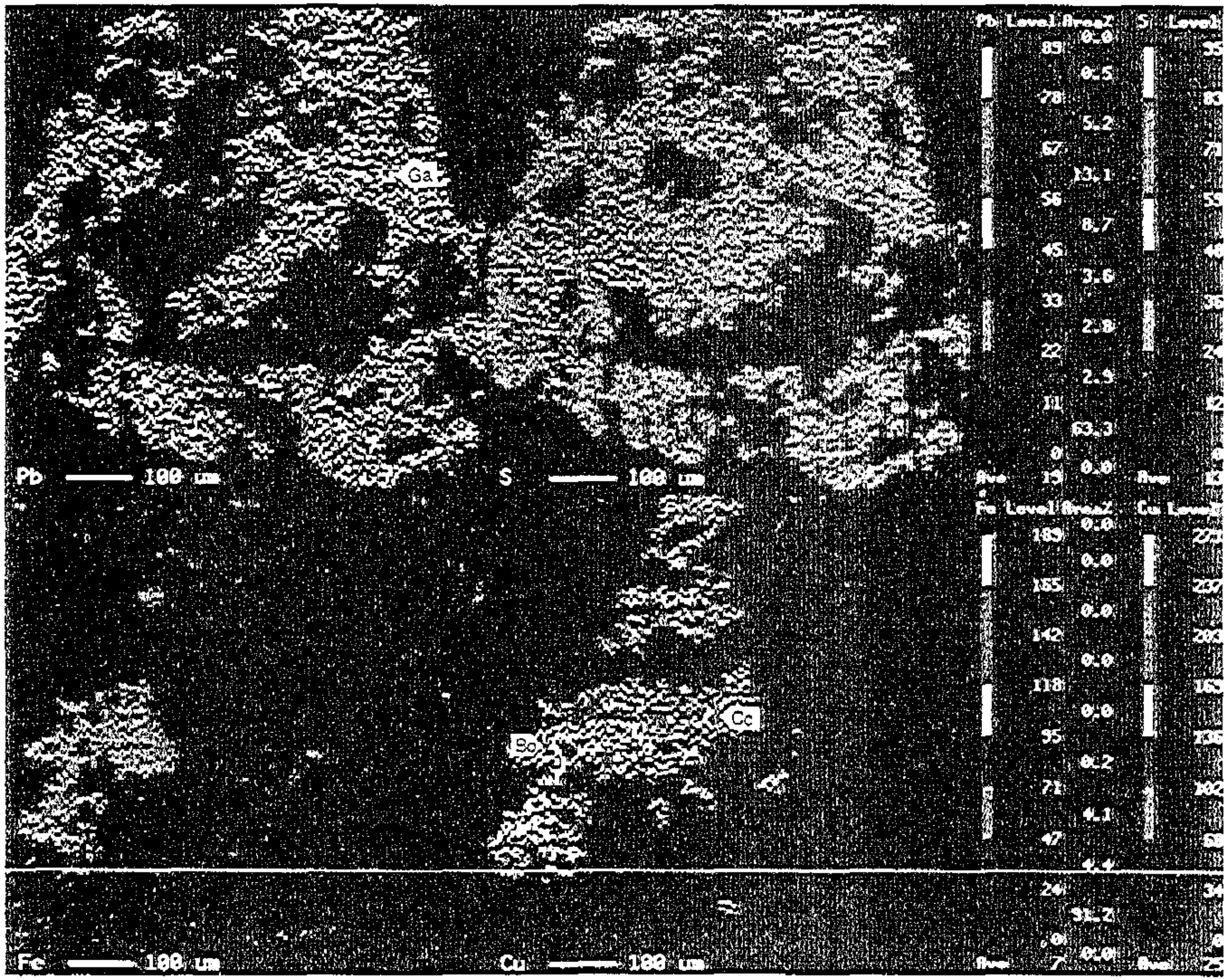
3) Qualitative analysis of ore minerals by electron probe microanalyser

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 Figures 2-2-5(1), 2-2-5(2) and 2-2-6. The results were checked both by quantitative analysis (Appendix 11) and by X-ray diffraction analyses.

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.

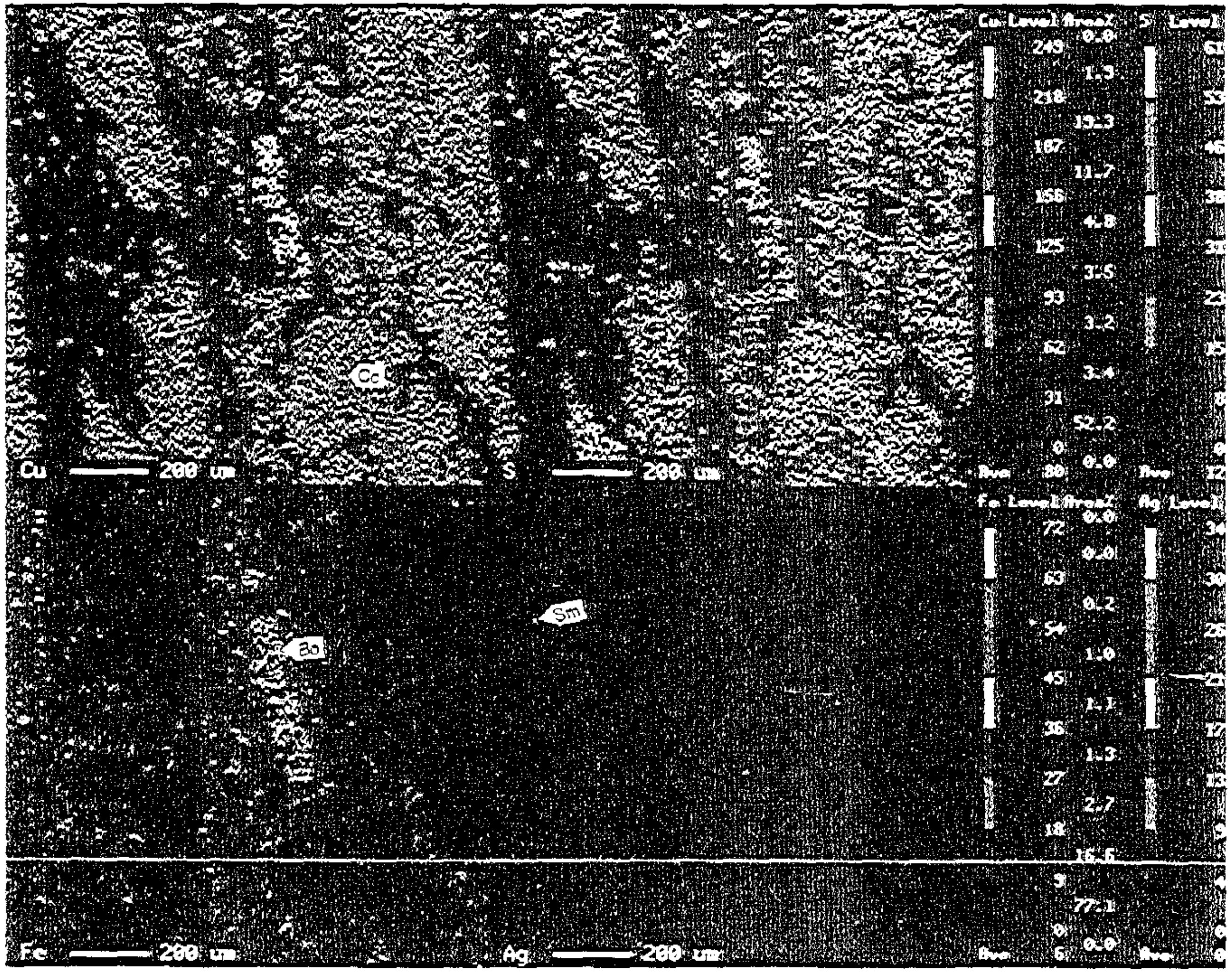
Chapter 3 Mining Technology and Mining Cost in the Zhezkazgan Copper Mine

The ultimate aim of this project is to make an evaluation of Zhaman-Aibat deposits. In the next year 1996, estimation of minable ore reserves of the deposits and a conceptual plan of development will be carried out. In this connection, the existent operating



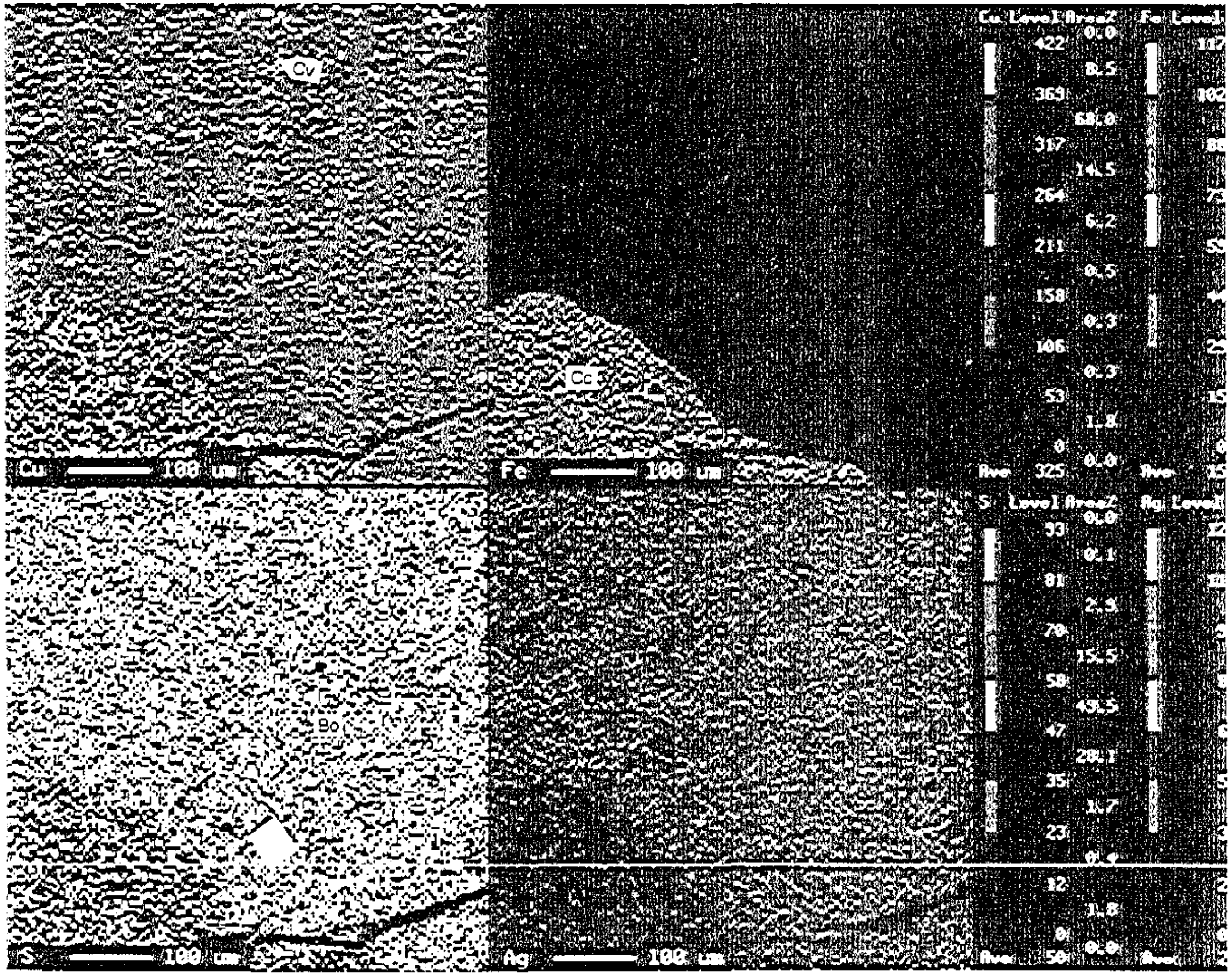
Sample No. :95-EP-01
Location : Eastern Orebody
DDH : MJK-1
Depth : 600.0m
Field code : No.19
Ore type : Cu ore
Minerals :
Cc : chalcocite
Bo : bornite
Ga : galena

Figure 2-2-5 (1) Electron Microprobe X-ray Color Image of Cu Ore by Drill Hole MJK-1 in the Eastern Orebody



Sample No. : 95-EP-02
Location : Eastern Orebody
DDH : MJK-1
Depth : 602.00m
Field code : No.23
Ore type : Cu ore
Minerals :
  Cc : chalcocite
  Bo : bornite
  Sm : stromeyerite

Figure 2-2-5 (2) Electron Microprobe X-ray Color Image of Cu Ore by Drill Hole MJK-1 in the Eastern Orebody



Sample No. : 95-EP-03
Location : Zhezkazgan South Mine
Ore type : Cu Ore
Minerals :
Cc : chalcocite
Bo : bornite
Cv : covellite

Figure 2-2-6 Electron Microprobe X-ray Color Image of the High Grade Ore in the Zhezkazgan Mine

condition, mining technology and operation cost in Zhezkazgan Mine (Figure 2-3-1) 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.

3-1 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 2-3-1, and operation of the Zhezkazgan Mine is summarized in Table 2-3-2. Among the four mines(Figure 2-3-2), 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 2-3-3.

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 (Figure 2-3-3) 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

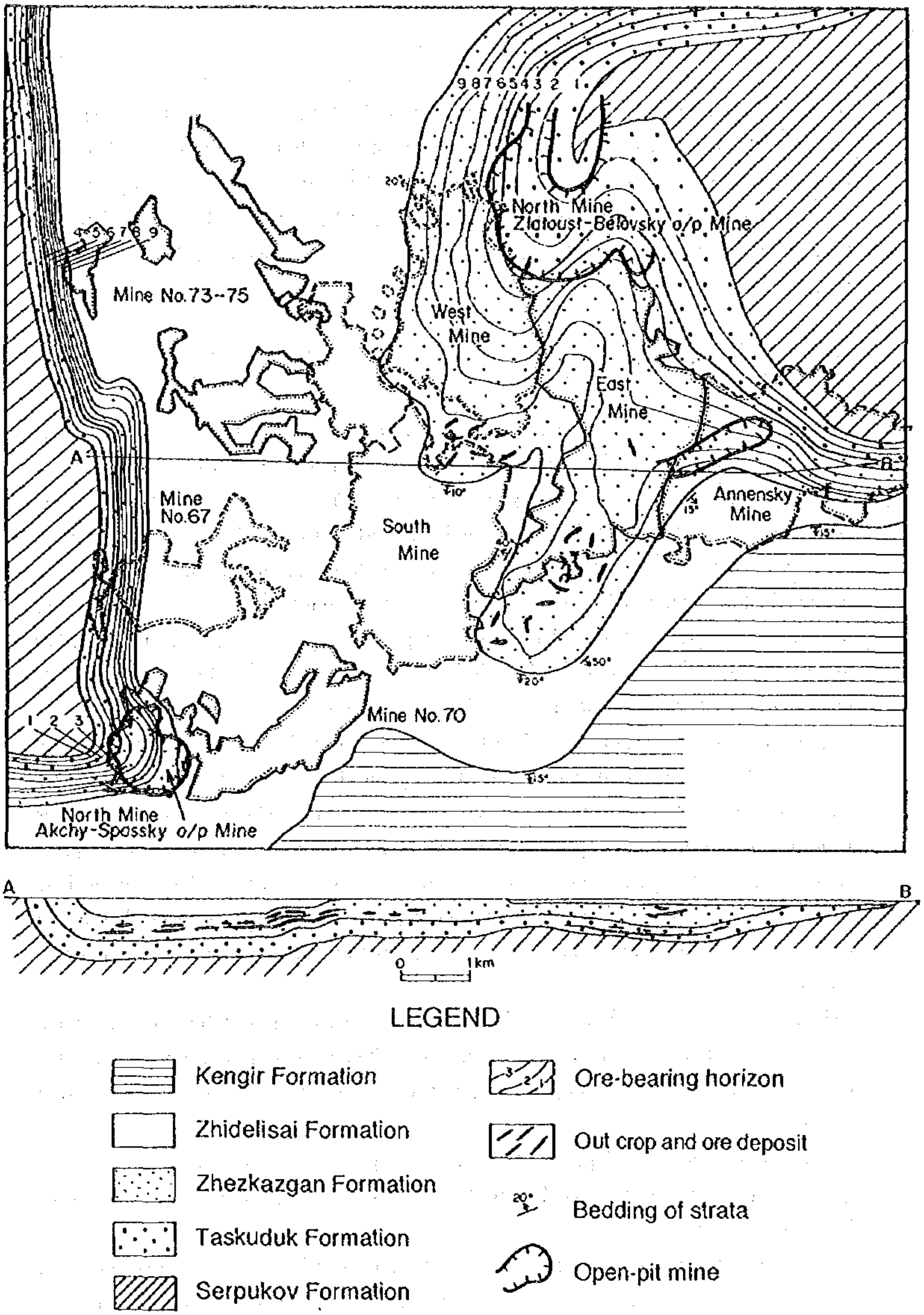


Figure 2-3-1 Mining Areas of the Zhezkazgan Mine

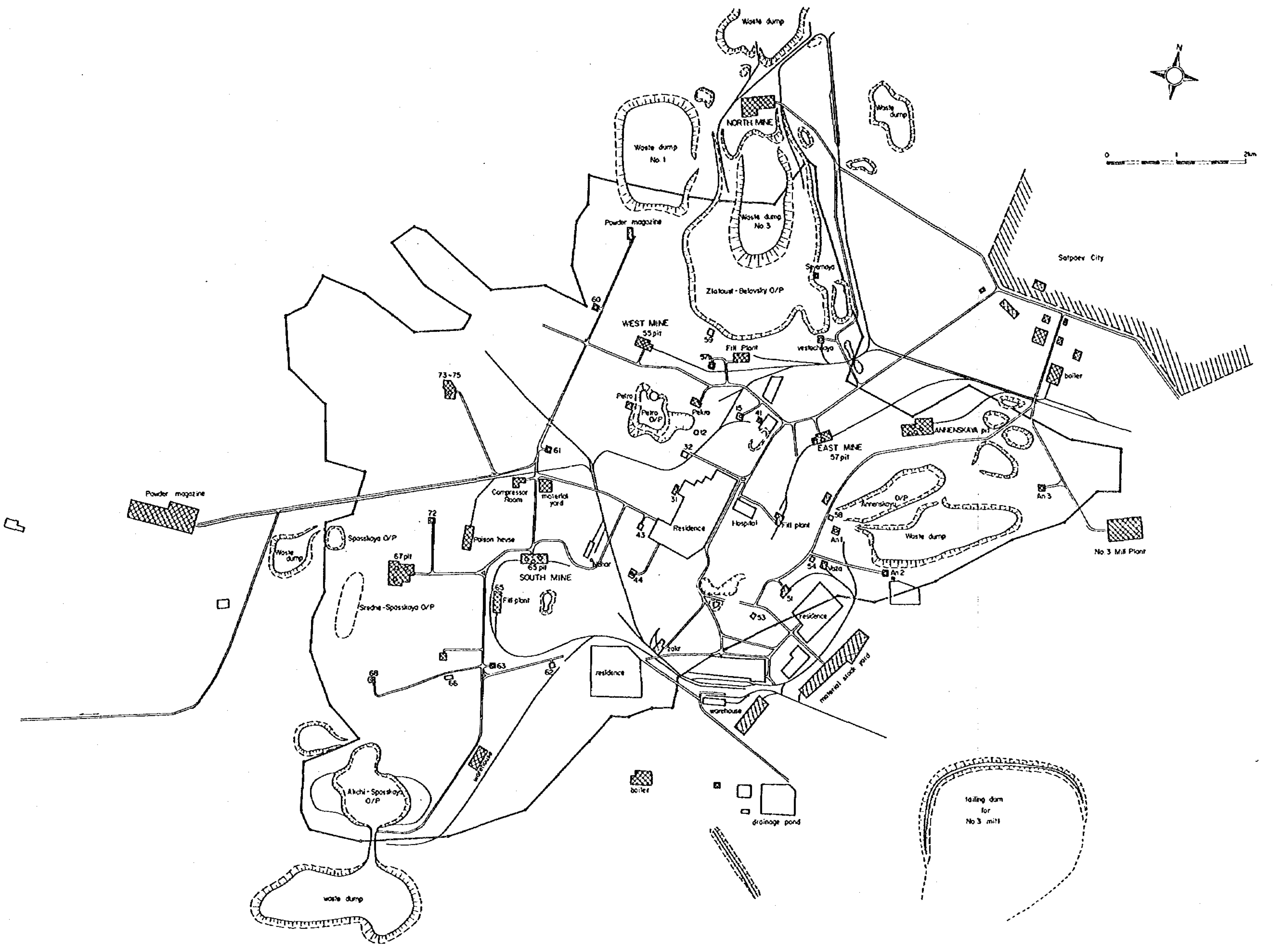


Figure 2-3-2 Location Map of Mining Facilities in the Zhezkazgan Mine

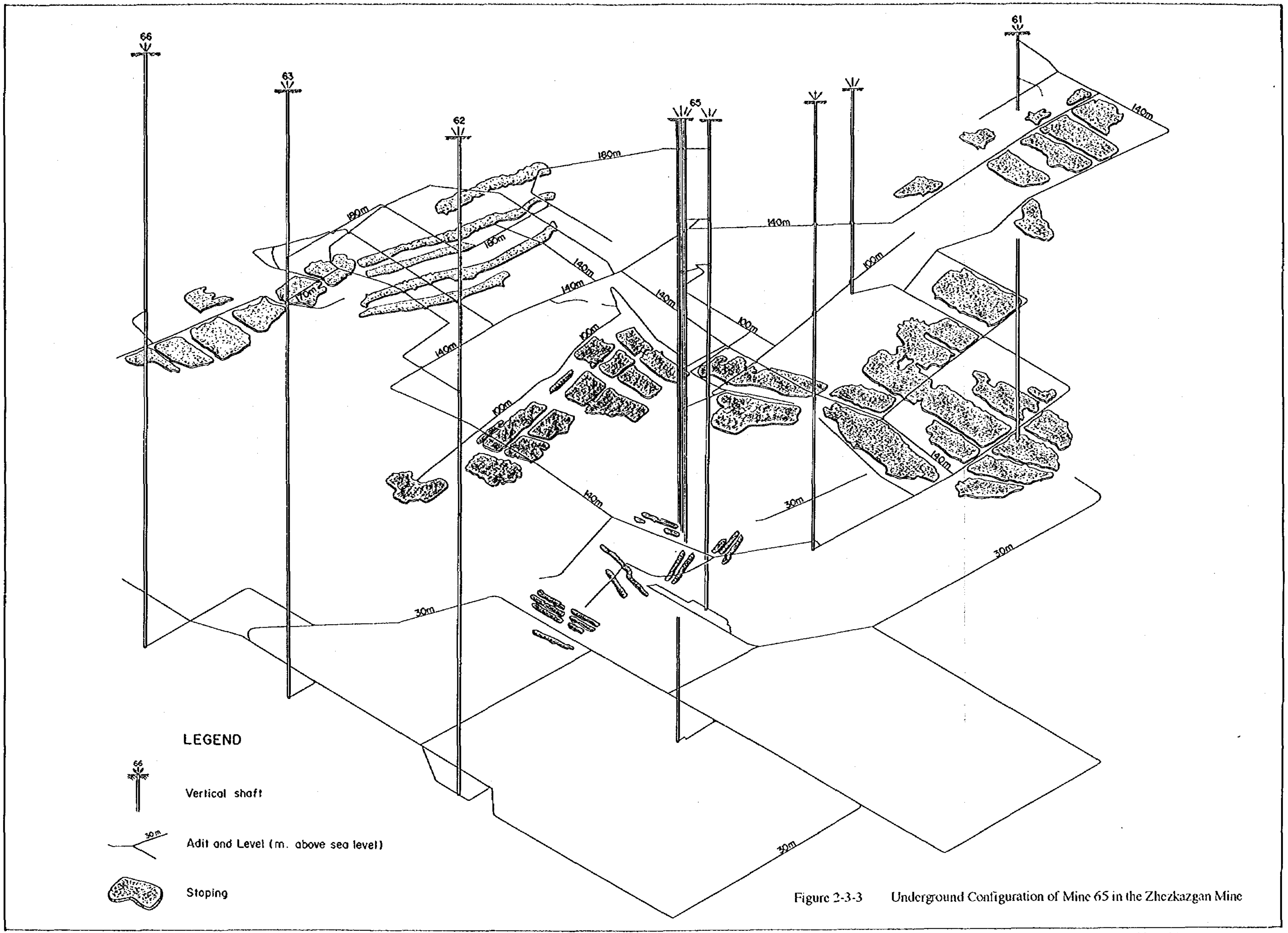


Figure 2-3-3 Underground Configuration of Mine 65 in the Zhezkazgan Mine

Table 2-3-1 Output of 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 2-3-2 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 2-3-2 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 2-3-3 Summary of 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

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 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 year'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 2-3-4 and total employees in mining and ore dressing plant are listed in Table 2-3-5.

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 2-3-4 Operation Cost of the 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 2-3-5 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

3-2 Mining Technology and Cost

As stated in the beginning, the final goal of this project is to evaluate the Zhaman-Aibat deposits and produce a conceptual development design. For this purposes, it is important to select a suitable mining method for the deposits and estimate a reasonable mining cost. In this connection, we have investigated this year such issues putting emphasis on the existing mining operation costs as well as mining technology in Zhezkazgan deposits that seem to have a very similar deposit formation to those of Zhaman-Aibat. Our investigation excludes smelting cost and the initial capital expenses.

3-2-1 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.

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.

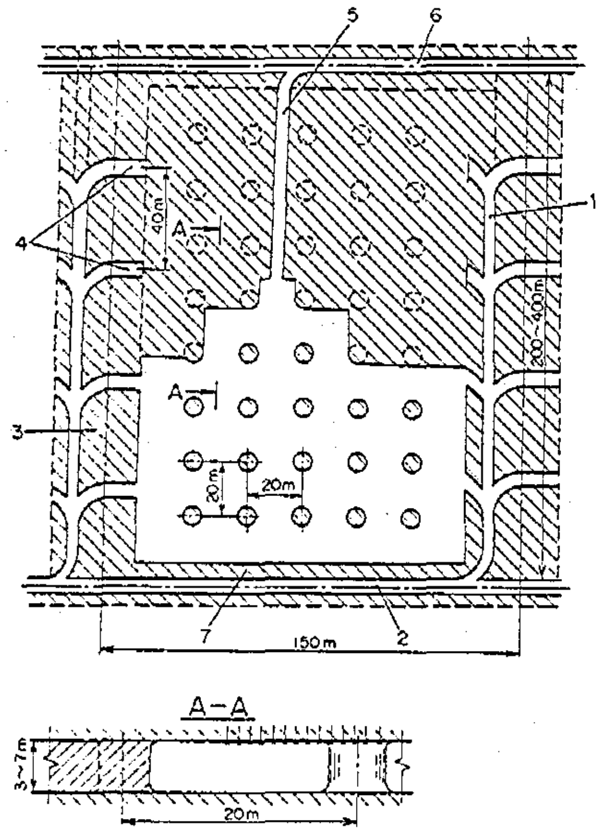
(1) Panel and Pillar Mining Method (One of the Room and Pillar Mining Methods)

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 Figure 2-3-4. 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 Figure 2-3-5. 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.

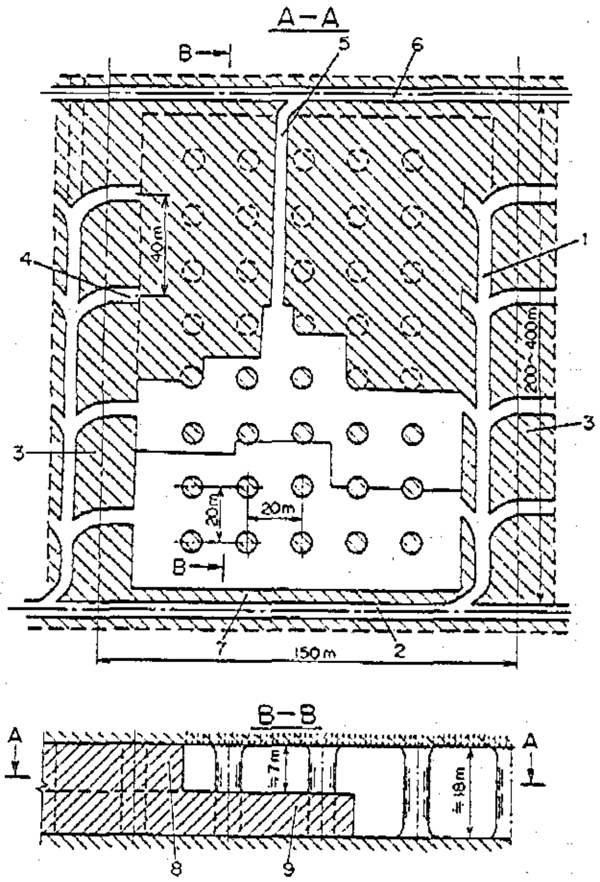
(2) Room and Pillar Mining Method (with Cut and Fill Mining Method)

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 Figure 2-3-6, 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



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

Figure 2-3-4 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 sloping for benching
9. bench

Figure 2-3-5 Panel and Pillar Mining Method
⟨ orebody with a gentle inclination, over 8m in thickness, andCu content lower than 2.5% ⟩

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.

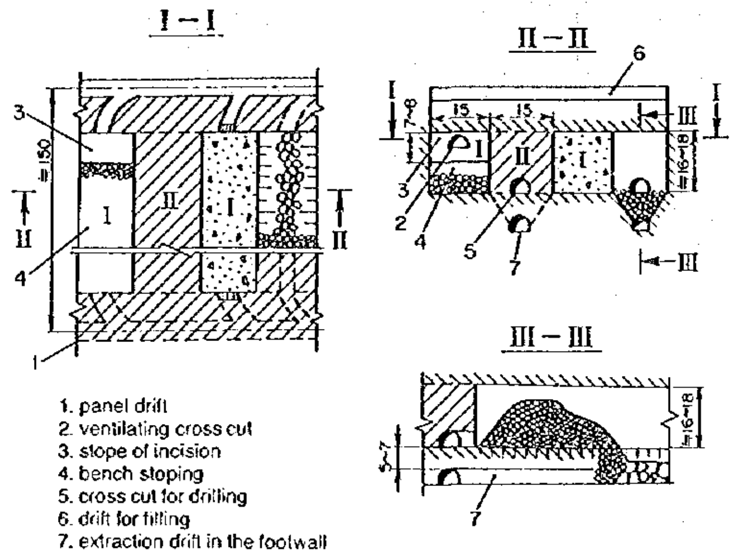


Figure 2-3-6 Room and Pillar Mining Method (with filling and cut and fill stoping in the primary chambers and Cu content higher than 2.5%)

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 m² is sunk down from the filling level (Figure 2-3-7).

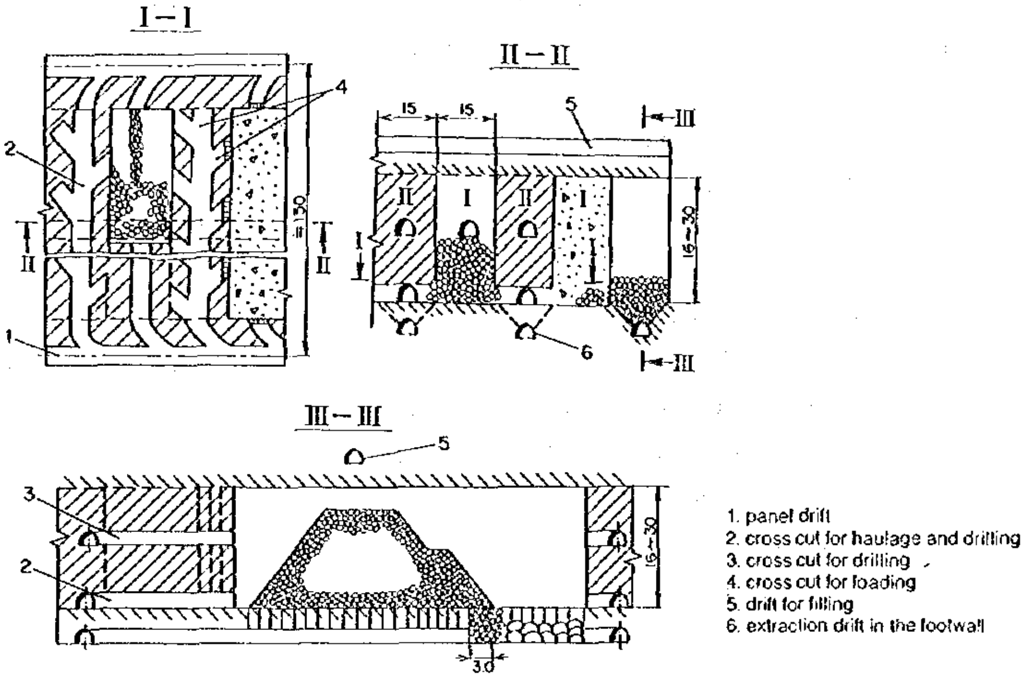


Figure 2-3-7　Room and Pillar Mining Method
(stoping height higher than 16～18m)

(3) *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 Figure 2-3-8, 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.

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.

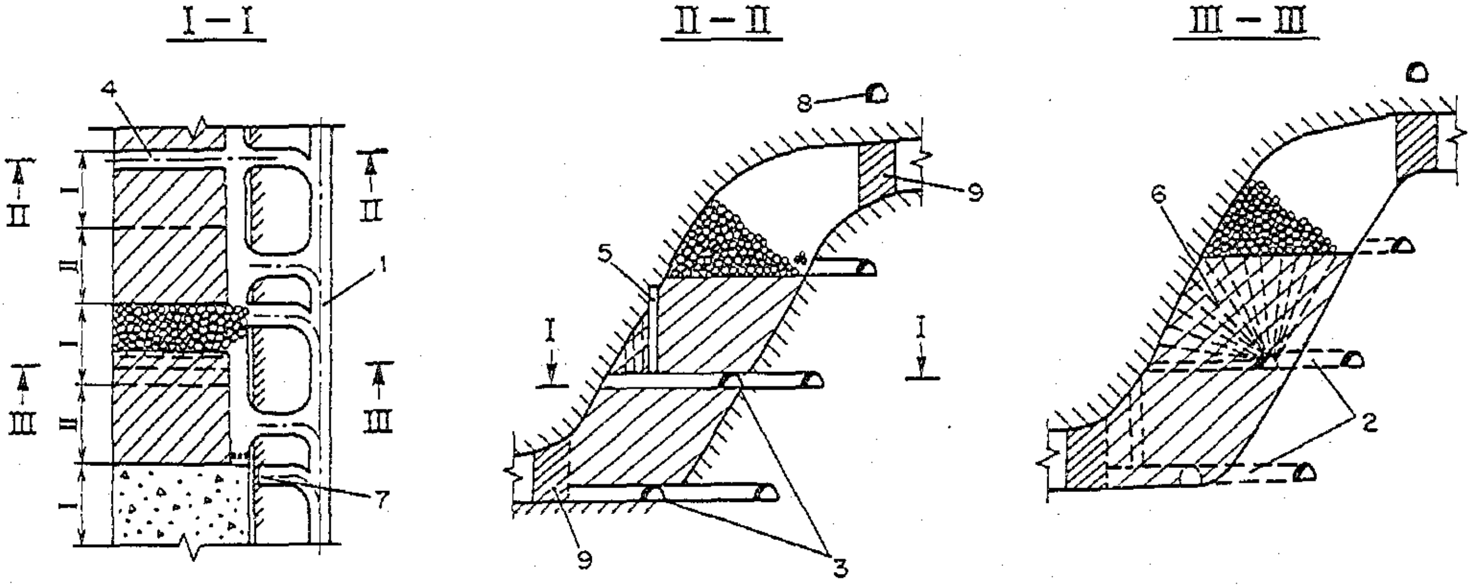
(1) Designing (Rib) Barrier Pillar

$$A=(K_{H}'\gamma\ h\ H\ L\ \ П_{Ь}\ K_{\alpha}/K_{TP}\,б_{н})1/2$$

		(Constant)
A	: Rib pillar width (m)	
K_{H}'	: Load coefficient on pillar	1
γ	: Rock density (t/m³)	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
$П_{Ь}$	: Safety coefficient of rib pillar	3
K_{α}	: Influence coefficient by stratiformed dip	
K_{TP}	: Influence coefficient by crack	
$б_{H}$	: Uniaxial compressive strength of grey sandstone (t/m²)	

(2) Designing Room Pillar

$$D=(4\ K_{H}\gamma\ \ h\ \ H\ \ S_{ON}\ П_{MK}\ \ K_{\alpha}/\pi\ \ б_{H}\ \ K_{TP}\ \ K_{NP}\ \ K_{NM}\ \ K_{K})1/3$$



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

Figure 2-3-8 Sublevel (Slicing) Stoping Method (with filling and orebody steeper inclination)

D	: Pillar diameter (m: diameter)	
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) $20 \times 20 = 400m^2$	
Π_{MK}	: Safety coefficient of pillar	2
K_α	: Influence coefficient by stratiformed dip	
π	: Circular constant	
б$_H$	: Uniaxial compressive strength of grey sandstone (t/m^2)	
K_{TP}	: Influence coefficient by crack	
	＊Less cracked grey sandstone	0.63
	＊More cracked grey sandstone	0.4
K_{NP}	: Coefficient of large waste band such as aleurolite or red sandstone	0.9
K_{NM}	: Influence coefficient by blasting	0.85~0.9
K_K	: Influence coefficient by contact	
	＊Grey sandstone	1
	＊Red sandstone	0.7

(3) Finalizing Parameter

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

K_H : Load coefficient on pillar

L/H (Width/Depth)	1.00	0.66	0.5	0.4	0.33
K_H (Hanging wall of grey sandstone)	0.7	0.6	0.55	0.5	0.45
K_H (Hanging wall of red sandstone)	0.85	0.8	0.77	0.75	0.72

The following values are given from past records.

б$_H$: Uniaxial compressive strength of hanging wall.

Depth (m)	150~200	300	400	500
б$_H$ (MPa)	200	230	240	245

* Calculation example in case of a gentle dip where foregoing parameters are applied.

The calculated examples of pillar size and ore recovery are shown in Table 2-3-6

K_α	: Influence coefficient by stratiformed dip	0.1
K_T	: Influence coefficient by crack	
	Less cracked grey sandstone	0.63

K_{NP} : Coefficient between walls such as aleurolite or red sandstone 0.9

K_{NM} : Influence coefficient by blasting 0.9

K_K : Influence coefficient by contact Grey Sandstone 1

D : Room pillar diameter (m)

A : Rib pillar width (m)

R : Ore recovery (%)

Table 2-3-6 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

3-2-2 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 2-3-4. Actual records of underground mining operation from January to July 1995 show 400 Tenge/ton-ore ($us6.70). 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.

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 2-3-7). 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

* This figure is over 35% of the expectation.

Table 2-3-7 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	

Actual records of unit cost from January to August 1995:

Electric Power	51,019 MW	21.3 KW/t
Drainage	$2.645 \times 10^6 m^3$	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.

2) Operative Expenses at North Mine

Operative expenses from January to July 1995 at North Mine are shown in Table 2-3-8.

Table 2-3-8 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

3-3 Outline of Ore Dressing Plant

A flow sheet of the ore dressing process is shown in Figure 2-3-9.

3-3-1 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 Т (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%

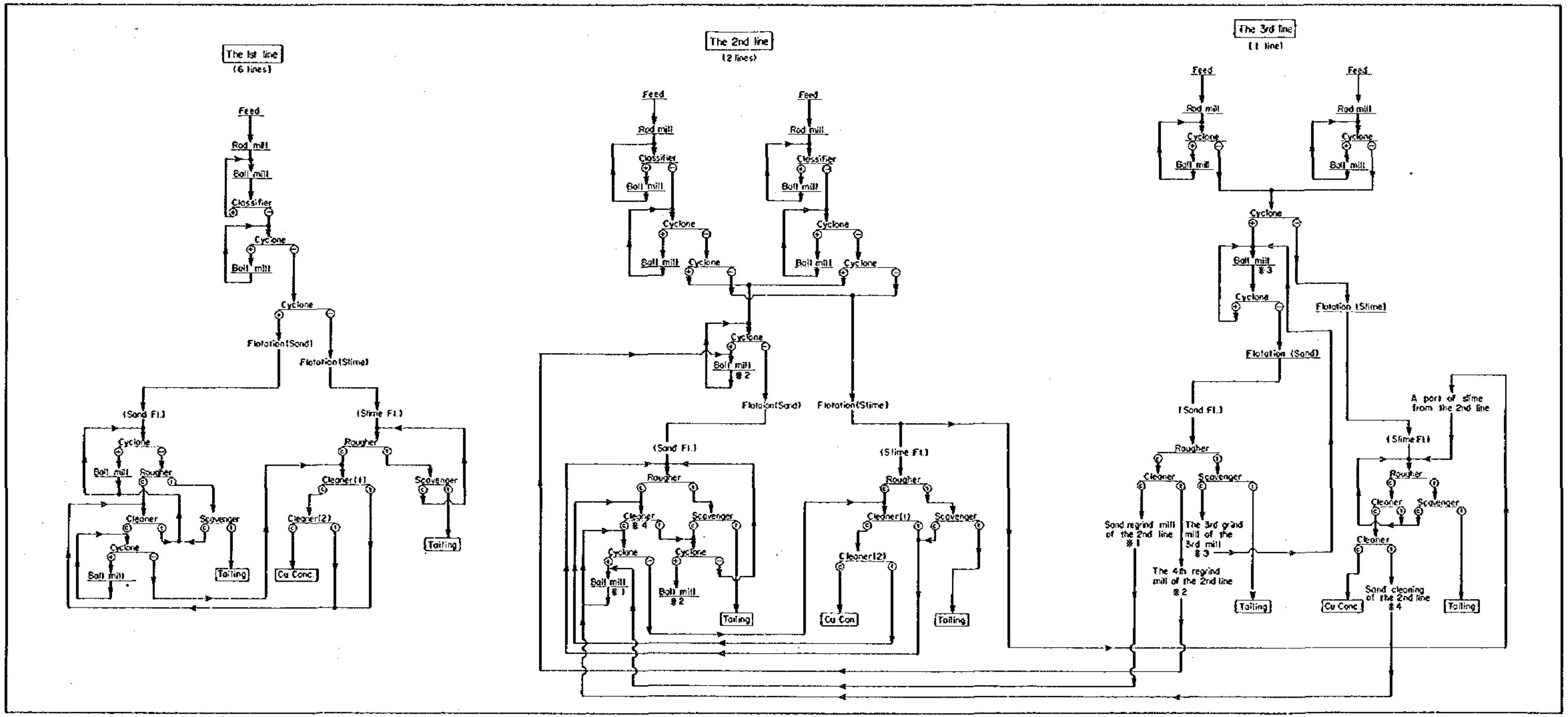


Figure 2-3-9 Flow Chart of Zhezkazgan Ore Dressing Plant

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.

3-3-2 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 З Т М 1,500/180×2 units with over 1,500 mm feeding size

Secondary crushing : Crusher К С П-2,200 Г Р ×8 units

Tertiary crushing : Crusher К М Л - 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 32m^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 36m^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 32m^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, ФМР-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 m^3/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).

3-3-3 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:

Primary crushing	: Medium sized crusher К С П-2200×2 units
Secondary crushing	: Small crusher К М П-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 Б О У-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 2-3-5).

3-4 Future Issues and Suggestion

As stated before, the ultimate aim of our present examination is to evaluate the Zhaman-Aibat deposits and to produce a conceptual design to develop it. For this purpose, the operative situation of Zhezkazgan deposits has been investigated this year because it seems to have a very similar deposit to that of the Zhaman-Aibat deposits. 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.

$$A=(K_H{}' \ \gamma \ h \ H \ L \ \Pi_{\text{ь}} \ K_\alpha / K_{TP} \ \text{б}_H)1/2$$

$$D=(4 K_H \ \gamma \ h \ H \ S_{ON} \ \Pi_{MK} \ K_\alpha / \pi \ \text{б}_H \ K_{TP} \ K_{NP} \ K_{NM} \ K_K)1/3$$

A : Rib pillar width (m)
D : Pillar size (min diameter)
K_H : Load coefficient on pillar <GL −600 m− 0.4>
<GL −700 m− 0.35>
$K_H{}'$: Load coefficient <1>
γ : Host rock density per unit volume (t/m^3) <2.6>
h : Pillar height, layer thickness or mining height (m)
H : Covering soil height above layer, mining depth (m)
L : Pillar spacing between both centers (m) <150>
$\Pi_{\text{ь}}$: Safety coefficient of rib pillar <3>
S_{ON} : Hanging wall area on pillar (m^2) <20 m×20 m = 400 m^2>
Π_{MK} : Safety coefficient of pillar <2>
K_α : Influence coefficient by layer dip <Gentle dip 0.1>
K_{TP} : Influence coefficient by crack <Less cracked grey sandstone 0.63>
K_{NP} : Coefficient of large waste band such as aleurolite, red sandstone <0.9>
K_{NM} : Influence coefficient by blasting <0.9>
K_K : Influence coefficient by contac <Grey sandstone 1>
π : Circle constant

σ_H : Uniaxial compressive strength of grey sandstone (t/m^2) <2500>

* D : Diameter of room pillar (m), A: rib pillar width (m), R: Ore recovery (%)

Mining Depth(m)	Layer Thickness(m)	2	4	6	8	10	12
	D	4.7	5.9	6.8	7.5	8.1	8.6
600	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
	D	4.8	6.0	6.9	7.6	8.1	8.6
700	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

Chapter 4 Ore Dressing Test

4-1 Test Samples

4-1-1 Composite Samples and Chemical Analyses of Ore

The amount of the composite samples 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, <0.1g/tAu, 12g/tAg (Table 2-4-1).

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.

Table 2-4-1 Chemical Analyses of Test Samples

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 : %

4-1-2 X-Ray Diffractometer Analyses of Ore

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 14)

4-1-3 Microscopic Observation of Polished Ore

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 which va- ried 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 par-ticle size of 1 to 300 μ m, most of which varied from 20 to 100 μ m. Galena was mostly scat-tered in the country rocks, but was occassionally associated with the chalcocite, chalco- pyrite and pyrite (Appendix 15)

The structures of the main minerals are shown in Table 2-4-2.

Table 2-4-2 The Structures of Main Ore Minerals

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
Bonite	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

4-1-4 The Analyses by EPMA (Quantitative Analyses)

The results of the analyses of EPMA are shown in Table 2-4-3 (Appendix 15).

Table 2-4-3 The Results of the Analyses of EPMA (Wt.%)

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					Galena
d		13.6		86.6			Galena
e	80.7	19.3		86.4			Chalcocite
f	64.1	24.7	11.2				Bornite
g		13.6					Galena
h	79.9	20.1		86.4			Chalcocite
i	80.3	19.7					Chalcocite
j	80.0	20.0					Chalcocite
k	80.5	19.5					Chalcocite

4-2 Preliminary Ore Dressing Tests

4-2-1 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 sciving 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 groundproduct was 80 μ m. The test sample was ground under the same condition for 15 and 20 min- utes, 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

$$12.1 \times \frac{19.1}{15.0} = 15.4 \text{ KWH/t}$$

Generally speaking, this sample ore was rather hard.

4-2-2 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 16).

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.

4-2-3 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 Figure 2-4-1.

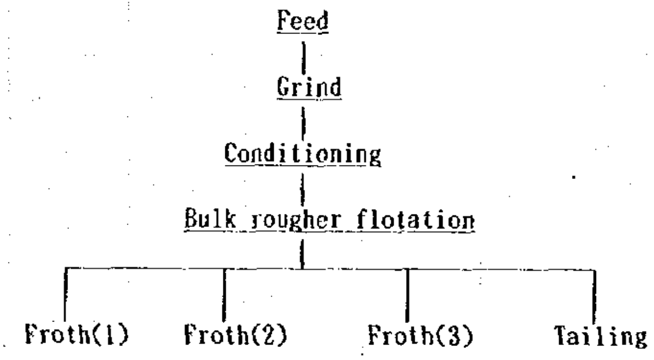


Figure2-4-1 The Flowsheet of Bulk Rougher Flotation

The results of the bulk rougher flotation are shown in Table 2-4-4.

The relation between the flotation size and recovery of the froth is shown in Figure 2-4-2.

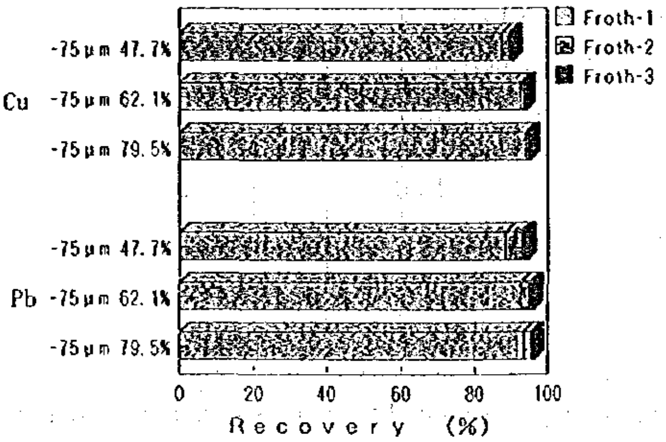


Figure 2-4-2 The Relation Between the Flotation Size and Recovery of the Froth

Table 2-4-4 The Results of the Bulk Rougher Flotation

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		

The weight % of structural minerals of bulk concentrate are shown in Table 2-4-5.

Table 2-4-5 The Weight % of Structural Minerals of Bulk Concentrate

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

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.

4-2-4 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 andthe test results are shown in Figure 2-4-3 and Table 2-4-6.

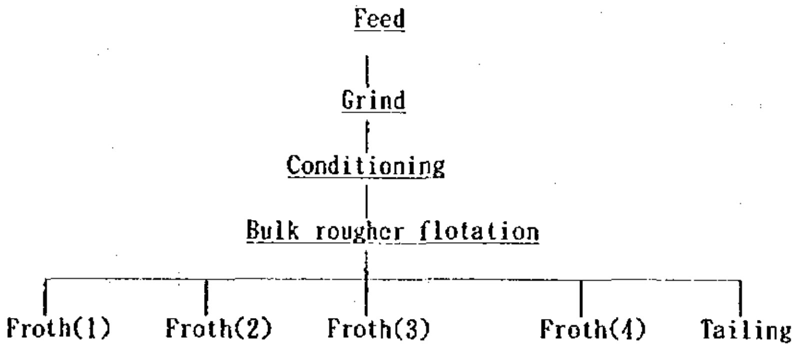


Figure 2-4-3 Flowsheet of Flotation Speed Test

Table 2-4-6 The Results of Flotation Speed Test

Test No.	Product	Wt.%	Grade %		Recovery %		Cum. Recovery %	
			Cu	Pb	Cu	Pb	Cu	Pb
KS－4 (-200mesh 64%)	Feed	100.00	1.70	0.49	100.00	100.00		
	Froth(1)	6.14	24.50	7.09	88.29	88.74	88.29	88.74
	Froth(2)	1.90	2.03	0.96	2.27	3.73	90.56	92.47
	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		
KS－5 (-200mesh 78%)	Feed	100.00	1.70	0.49	100.00	100.00		
	Froth(1)	5.87	26.23	7.53	90.63	90.63	90.63	90.63
	Froth(2)	2.30	1.61	0.93	2.18	4.38	92.81	95.01
	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		

Both copper and lead floated with recovery 88～90% in the first 3 minutes indicating thatthis 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 timewas sufficient to optimise recovery.

4-2-5 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 Figure 2-4-4.

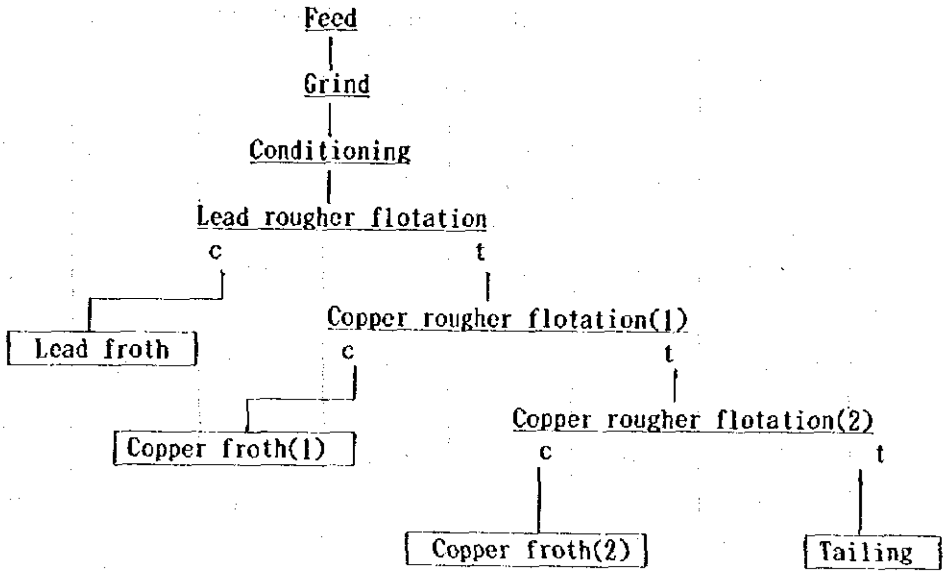


Figure 2-4-4 Flowsheet of Straight Differential Flotation Test

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 2-4-7.

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.

Table 2-4-7 Test Results of Straight Differential Flotation Test

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		
K S − 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		
K S − 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		
K S − 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		

4-2-6 Size Analyses of Flotation Tailing

Size analyses and chemical analyses of size fractions were carried out on the tailings oftest KS-1,2,3. The results are shown in Appendix 18.

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.

4-3 Differential Flotation Test

4-3-1 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 paticlesize of regrinding was investigeted. The flowsheet of the bulk differential flotation testis shown in Figure 2-4-5.

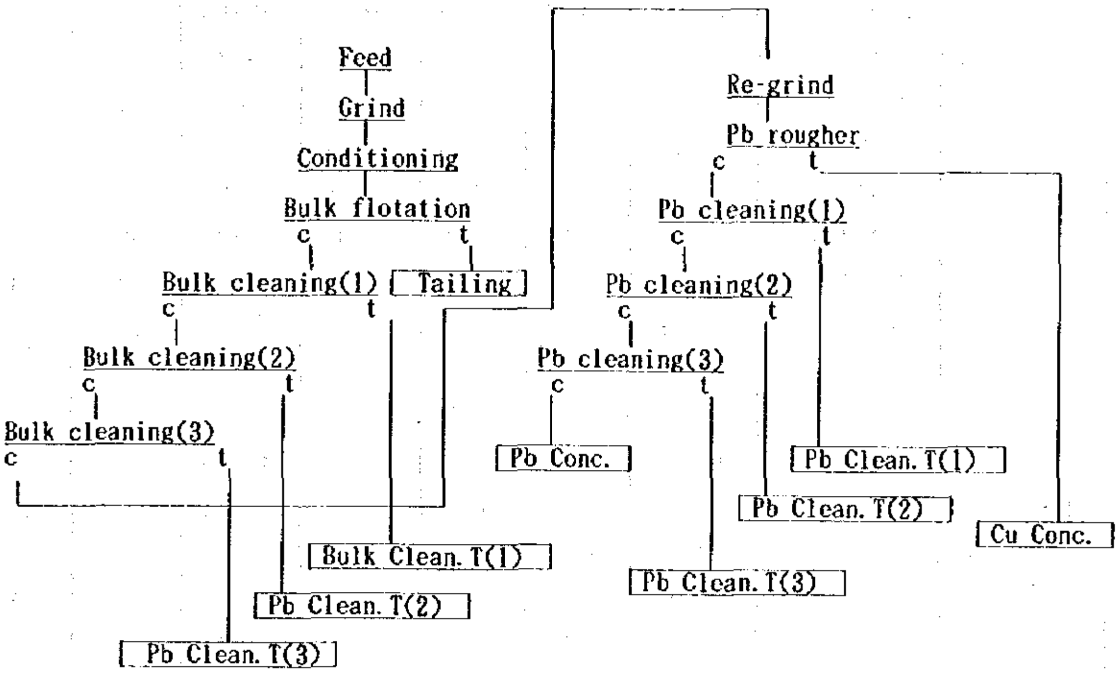


Figure 2-4-5 Flowsheet of the Bulk Differential Flotation Test

The test results of lead-copper separation differential flotation were carried out with various regrinding sizes in three stages is shown in Table 2-4-7.

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 notsufficient, 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.

Table 2-4-8 The Results of Bulk Diffrential Flotation Test (Effect of Particle Size)

Test No.	Product	Wt.%	Grade %		Recovery %		Cum. Recovery %	
			Cu	Pb	Cu	Pb	Cu	Pb
	Feed	100.00	1.74	0.50	100.00	100.00		
KS－10	Pb Conc.	2.05	31.10	17.10	36.48	69.45	36.48	69.45
(-200mesh	Pb clean. T-2	0.56	49.80	1.56	15.93	1.73	52.41	71.18
86.3%)	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		
	Feed	100.00	1.70	0.49	100.00	100.00		
KS－11	Pb Conc.	1.64	28.90	20.50	27.62	68.34	27.62	27.62
(-200mesh	Pb clean. T-2	0.49	53.90	2.31	15.26	2.28	42.88	70.62
93.7%)	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		
	Feed	100.00	1.70	0.49	100.00	100.00		
KS－12	Pb Conc.	1.21	27.30	22.00	19.41	54.70	19.41	54.70
(-200mesh	Pb clean. T-2	0.50	52.40	4.87	15.52	5.04	34.93	59.74
95.1%)	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		

4-3-2 Bulk Differential Flotation Test (Investigation of Collectors in Copper-Lead Separation Flotation)

In order to investigate collectors, four different kinds; Sodium isopropylxanthate (NaIPX), NaIPX + Aeropromoter242 (AP242), Aeropromoter 3418a (AP3418a) and Sodium ethyl xanthate (NaEX), were tested. The re-grinding size was 80% passing size 27 μ m and the flow was as same as 4-3-1. The test results are shown in Table 2-4-9.

The difference between the colletors was not significant, and the copper grade in the lead concentrate was lowest when NaIPX was used. From that, NaPIX 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 is as follows.

4-3-3 Confirmation Test (Investigation of the Best Conditions)

Based on the above observations, the best conditions to get the most desirable copper andlead 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 sulfite, sodium carbonate and sodium silicate as condi- tioners. 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 2-4-11.

Table 2-4-9 The Results of Bulk Diffrential Flotation Test (Effect of Collectors)

Test No.	Product	Wt %	Grade %		Grade %		Grade %	
			Cu	Pb	Cu	Pb	Cu	Pb
KS-13 (NaIPX+ AP242)	Feed	100.00	1.67	0.50	100.00	100.00		
	Pb Conc.	0.82	13.10	44.10	6.43	73.04	6.43	73.04
	Pb clean. T-2	0.46	50.90	3.41	14.02	3.17	20.45	76.21
	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		
KS-14 (NaIPX)	Feed	100.00	1.68	0.47	100.00	100.00		
	Pb Conc.	0.78	9.17	43.30	4.29	71.83	4.29	71.83
	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		
KS-15 (AP3418a)	Feed	100.00	1.70	0.48	100.00	100.00		
	Pb Conc.	0.94	13.70	38.00	7.63	73.83	7.63	73.83
	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		
KS-16 (NaEX) 86.3%)	Feed	100.00	1.69	0.49	100.00	100.00		
	Pb Conc.	0.75	11.10	43.00	4.95	68.14	4.95	68.14
	Pb clean. T-2	0.35	40.80	5.77	8.59	4.32	13.54	72.46
	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 2-4-10 Combined Copper Concentrate

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 2-4-11 Test Results of the Bulk Differential Flotation

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
	Tailing	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		

Combining the lead cleaning tailing and the copper concentrate, the grade and recovery ofthe copper concentrate is as follows.

Table 2-4-12 Combined Copper Concentrate

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

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 sulfite, sodium carbonate and sodium silicate as conditioners. Flotation time was 17 min- utes. After re-ginding of the lead rougher froth the lead concentrate was gained by clea-ning 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 Figure 2-4-6 and an example of test results is shown in Table 2-4-13.
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.

4-4 Determination of the Flotation Flow Process

A high grade of lead concentrate (higher than 50%Pb) was not obtained by the bulk differ-ential 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 copperis 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 Figure2-4-7.

Table 2-4-13 Straight Differential Flotation Test Results

Test No.	Product	Wt.%	Grade %		Recovery %		Cum. Recovery %	
			Cu	Pb	Cu	Pb	Cu	Pb
KS－22	Feed	100.00	1.63	0.48	100.00	100.00		
	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		

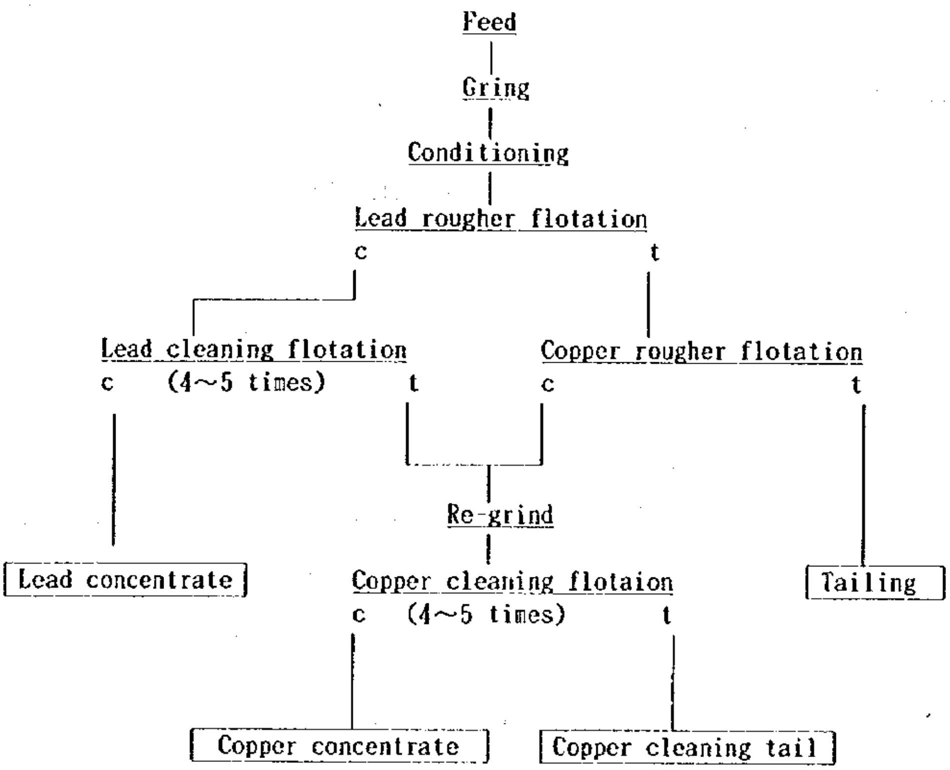


Figure2-4-6 Straight Differential Flotation Test Flowsheet

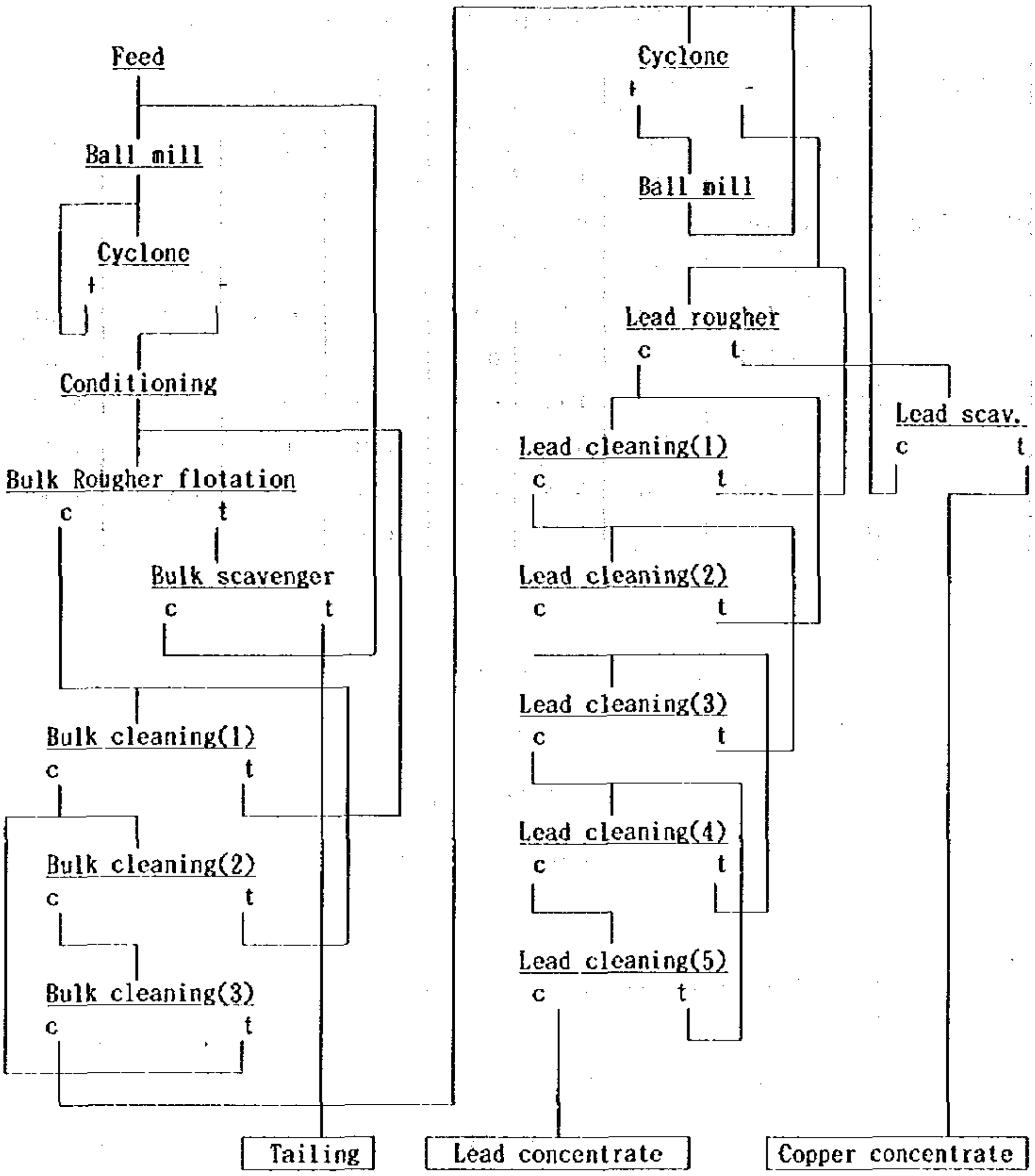


Figure 2-4-7 Optimum flowsheet

4-5 Chemical Analyses of Copper and Lead Concentrates

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

Table 2-4-14 Chemical Analyses of Copper and Lead Concentrate

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; %

4-6 Comment and Recommendation

The amount of sample received this year was less than 23 kg. It took about two months to get export permission for this sample. So there was not enough time to fully test and evaluate the results. It is necessary to complete the investigations by the begining of the 3rd year research.

This year, "copper ore",the representative ore type in the Eastern Orebody was investigated. But it is considered that there are many other types of ore such as "complex ore (Pb+Zn+Cu)", "copper-silver ore (Cu+Ag)" and so forth within this deposit. Those estimated ore reserve is several ten million tons in in-situ base. For the purpose of ore dressing tests more samples will be required and more practical mineral processing evaluation should be done by flotation tests of other ore types.

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.

PART III

CONCLUSION AND RECOMMENDATION

PART Ⅲ CONCLUSION AND RECOMMENDATION

Chapter 1 Conclusion

The result of the present study leads to the following conclusions.

(1) Zhaman-Aibat copper deposit represents resources that are essential to the economy of the Republic of Kazakhstan. It is recommended to proceed not only with the exploration of new ore deposits but also the evaluation of the known, unexploited deposits.

(2) By the results of previous data analysis, drilling survey and various kinds of laboratory tests, it is proved that the Zhaman-Aibat copper deposit is a type of stratiform copper deposit and has many analogies of geology and mineral deposition to the Zhezkazgan copper deposit.

(3) For the purpose of obtaining representative samples of copper ore for geological and mineralogical studies and ore dressing test, MJK-1 (depth : 650m, vertical) was drilled in the Eastern Orebody. Total core recovery was as high as 98.5% and drillhole inclination ranged from 0° 30' to 1° 15'. Copper mineralization could be observed between the depth from 598.0m to 605.78m. The mineralized core length reached 7.78m with average grades of 3.78%Cu, 1.17%Pb, 0.03%Zn and 22.7g/tAg. Chalcocite was the predominant ore mineral, accompanied by a small amount of bornite and galena. Rare abundance of chalcopyrite, electrum, stromeyerite, gersdoritite-cobalite series mineral, mere noticed. The ore was a typical copper ore.

(4) The ore dressing test sample was taken from the boring cores of the deposit and some important deta for deposit evaluation were obtained by the ore dressing tests. The ore grade was 1.69%Cu, 0.51%Pb, 0.03%Zn and 12g/tAg. The main copper mineral was chalcocite and small amounts of galena were observed. The ore was rather hard with the work index 15.4KWH/t. The optimum flotation process is bulk differential flotation and the copper concentrate of 39%Cu and 1%Pb with copper recovery 86% and the lead concentrate of 48%Cu and 11%Pb with lead recovery 67% were obtained. There are much other types of ore body such as "complex ore (Pb+Zn+Cu)", "copper-silver ore (Cu+Ag)" and so forth within this deposit. So, more ore dressing tests should be done for those other ore types.

(5) The investigation of the Zhezkazgan operating mine concluded that mining technologies being used are applicable to the exploitation of the Zhaman-Aibat copper deposit. It is concluded that the mining costs, or capital cost and a part of the operation

cost should be investigated in 1996 for the economic evaluation.

(6) For the ore reserve evaluation scheduled in Phase III, construction of the "Zhaman-Aibat data base" has started and approximately 75% of the data on drilling and chemical assays were input to the data base. It is essential for ore reserve evaluation and future planning that the data base be completed at an early stage.

(7) For the second straight year, the ore reserves (i.e. geological resource estimation) for the Eastern and the main part of the Central Orebody were calculated by the polygon method using 302 drillholes with 3,564 point data sets.

The results of the Eastern Orebody showed that the ore reserve as copper ore is calculated as 117million tons with an average grade of 1.36%Cu and 10.78g/tAg, and an average thickness is 5.15m.

On the other hand, the results of the Central Orebody indicated that the total ore reserve for copper ore is calculated as 35 million tons and the average grades are 1.71%Cu, 1.04%Pb, 0.20%Zn and 10.75g/tAg with an average thickness of 4.71m. The ore reserve for the complex ore is calculated as 17.6million tons with an average grade of 1.54%Cu, 1.77%Pb, 0.33%Zn, 10.94g/tAg and an average thickness is 5.77m.

It can be seen that the overall results obtained by the Japanese survey team were larger in the ore reserve and lower in ore grade than those obtained by counterparts due to the discrepancy of ore/waste definition between the estimations.

(8) In the Phase I report, it was pointed out that all works from assay data sheet preparation to the ore reserve estimation are done by hand and generate several data sheets. Errors, omissions and inconsistencies in the hand written data sheets reduce the reliability of each data. It is therefore recommended to adopt a computer system for the management of exploration data.

In response to this recommendation, the Government of Japan donated a set of personal computer systems including computers and printers to the Zhezkazgangeologiya. The system is being adopted for geophysical data analysis and processed the previous survey data.

There are, however, some problems related to a lack of personal computer sets, software and skill in computer operation. Incorporation of computer systems into the daily exploration works is urgently required.

Chapter 2 Recommendation for Phase III Survey

Based on this year's results, it is recommended to carry out the following exploration works for the Phase III(1996) campaign.

(1) Study and evaluate previous survey data, in particular, data evaluation on the Central and Northern Orebodies is needed.

(2) Drilling survey

It is recommended to drill in order to obtain basic data and ore samples for ore dressing tests in the Central Orebody for complex ore and in the Northern Orebody for copper-silver ore.

(3) Ore dressing test

By using the mineralized core samples of complex ore and copper-silver ore, ore dressing tests should be carried out in order to confirm the basic parameters of ore reserve evaluation.

(4) Ore reserve evaluation

① study the present drilling interval by the geostatistical approach.

② investigation of the capital and operating costs

③ minable ore reserve calculations

④ conceptual design and economic evaluation of the Zhaman-Aibat Mine.

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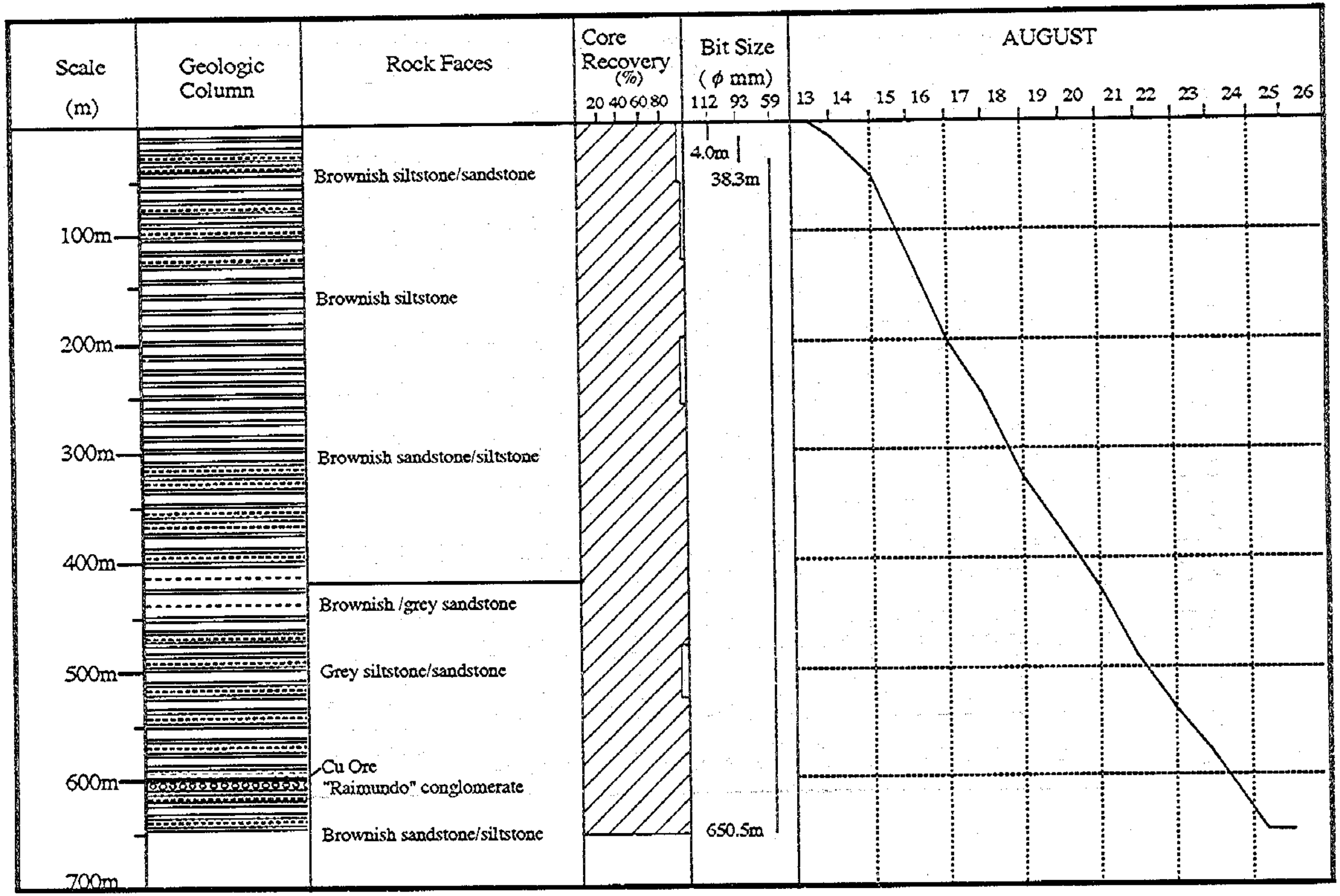
Appendices

Appendix 1 Assay Results of Check Analysis of Ore Samples from the Zhaman-Aibat Ore Deposit

Sample Location				Dzezukazgan Labs				Chemex Labs				
Drill No.	Spl. No.	From (GL-m)	To (GL-m)	Cu %	Pb %	Zn %	Ag g/t	Cu %	Pb %	Zn %	Ag g/t	Au g/t
364	16624	621.20	621.70	1.65	<0.05	<0.05	4.2	1.65	<0.01	0.02	4.1	<0.005
364	16625	621.70	622.30	3.70	<0.05	<0.05	16.5	3.62	0.01	<0.01	10.5	<0.005
364	16626	622.30	622.80	11.06	<0.05	<0.05	50.0	10.20	0.02	<0.01	42.3	<0.005
364	16627	622.80	623.30	5.53	<0.05	<0.05	35.0	5.46	0.01	<0.01	22.8	<0.005
364	16628	623.30	624.00	11.55	<0.05	<0.05	67.0	11.80	0.01	0.01	49.2	<0.005
364	16629	624.00	625.00	4.44	<0.05	<0.05	18.5	4.59	0.01	0.01	20.4	<0.005
364	16632	626.00	626.50	0.47	<0.05	<0.05	3.4	0.52	<0.01	0.01	1.6	<0.005
266	30225	613.20	613.80	2.04	<0.05	<0.05	6.0	1.39	<0.01	0.01	5.6	<0.005
266	30229	615.30	615.90	1.87	<0.05	<0.05	5.0	1.88	0.01	0.01	4.7	<0.005
266	30230	615.90	616.60	4.56	<0.05	<0.05	18.0	4.55	0.01	0.01	15.9	<0.005
266	30231	616.60	617.10	0.50	<0.05	<0.05	4.0	0.49	<0.01	0.01	2.4	<0.005
266	30232	617.10	617.80	2.21	<0.05	<0.05	15.5	2.10	<0.01	0.01	23.1	<0.005
266	30233	617.80	618.80	5.28	<0.05	<0.05	23.5	5.13	<0.01	0.01	27.6	<0.005
279	30637	613.80	614.30	0.35	<0.05	<0.05	2.0	0.33	<0.01	0.01	0.7	<0.005
279	30640	615.80	616.30	3.72	<0.05	<0.05	19.5	3.35	<0.01	0.01	21.4	<0.005
279	30642	616.90	617.50	1.90	<0.05	<0.05	11.5	1.88	0.01	0.01	11.5	<0.005
252	13989	615.40	615.90	3.04	<0.05	<0.05	17.0	3.12	<0.01	0.01	15.9	<0.005
254	30010	597.05	597.70	2.62	0.15	<0.05	10.0	2.65	0.01	0.01	10.6	<0.005
254	30011	597.70	598.20	3.40	<0.05	<0.05	9.5	3.45	0.01	0.01	8.5	<0.005
254	30012	598.20	598.70	0.90	<0.05	<0.05	3.5	0.94	<0.01	0.02	1.6	<0.005
254	30013	598.70	599.20	0.23	<0.05	<0.05	0.5	0.23	0.01	0.02	<0.3	<0.005
254	30014	599.20	599.70	3.26	<0.05	<0.05	7.5	3.16	0.01	0.02	9.7	<0.005
254	30015	599.70	600.25	6.36	<0.05	<0.05	10.5	6.29	0.01	0.01	11.0	<0.005
245	15540	598.50	599.00	4.67	<0.05	<0.05	10.0	4.19	<0.01	0.02	14.7	<0.005
398	18005	528.90	529.90	2.88	<0.05	<0.05	4.5	2.93	<0.01	0.02	4.5	<0.005
398	18008	531.50	532.00	0.95	<0.05	<0.05	3.4	0.99	<0.01	0.01	3.4	<0.005
567	111883	523.45	524.10	1.50	<0.05	<0.05	4.4	1.34	0.01	<0.01	4.6	0.025
567	111887	526.25	526.80	0.43	<0.05	<0.05	3.0	0.45	<0.01	<0.01	2.6	0.025
726	120448	574.10	574.60	3.05	<0.05	<0.05	16.0	3.35	<0.01	0.01	12.0	<0.005
726	120449	574.60	575.10	6.25	<0.05	<0.05	21.7	5.98	<0.01	0.01	20.5	<0.005
726	120453	576.60	577.70	2.61	<0.05	<0.05	4.4	2.87	<0.01	<0.01	3.4	<0.005
255	30042	600.60	601.50	3.16	<0.05	<0.05	7.3	3.01	0.01	<0.01	7.5	<0.005
402	18072	550.10	550.60	1.23	<0.05	<0.05	2.6	1.17	0.02	0.01	1.9	<0.005
402	18073	550.60	551.10	1.84	0.22	<0.05	4.6	1.93	0.06	0.01	3.5	<0.005
402	18074	551.60	552.60	6.36	0.42	<0.05	15.0	6.26	0.11	0.02	10.9	<0.005
402	18075	552.60	553.60	4.36	0.27	<0.05	10.0	4.53	0.07	0.02	6.8	<0.005

Appendix 2 Drilling Progress by Hole



Appendix 3 List of Drilling Equipments

Article	Model	Specification	Quantity
Drilling machine	Z I F - 650 M	Capacity : ϕ59mm 800m Inner diameter of spindle : 63.5mm Spindle speed : 81～800 rpm Weight : 2800kg	1 set
Power unit	A-2-ч 2-4	Electric Motor Revolution : 1450rpm Related power : 30 KW 380V	1 set
Drilling pump	NB-320/100	Type : 3 cylinder single acting Volume (max) : 320 ℓ /min Pressure (max) : 63 kg/cm^2	1 set
Power unit	4Λ200-M б U Z-220/380V	Electric Moter Revolution : 100rpm Related power : 22KW 380V	1 set
Water supply pump	б-12-33A	Type : turbine Volume (max) : 50 ℓ /min Pressure (max) : $50kg/cm^2$	1 set
Power unit	AO2-У 1-6	Electric motor Revolution : 960rpm Related power : 3 KW	1 set
Wire line hoist	K-б 3×25+1×16		1 set
Derrick	m R U б U-18/20	Pipe structual derrick	1 set
Generator	б ms-13-41 12 Om-4	Diesel engine Revolution : 500rpm Related power : 320KVA Weight : 4080kg	1 set
Drill rod	CCK-59		650m
Water tank		$9m^3$	1 set

Appendix 4 Amount of Consumed Materials of Drilling Survey

Article	Unit	Quantity
Diamond Bit 59mm	Pcs	10
Cemented carbide bit 112mm	Pcs	1
do. 93mm	Pcs	3
Diamond reaming shell 59mm	Pcs	2
Core lifter	Pcs	13
Core lifter case	Pcs	6
Core box	Pcs	130
Lost circulation mtarial	Kg	100
Diesel	l	8000
Gasoline	l	2800
Engine oil	l	400

Appendix 5 Operational Results of Drilling Survey

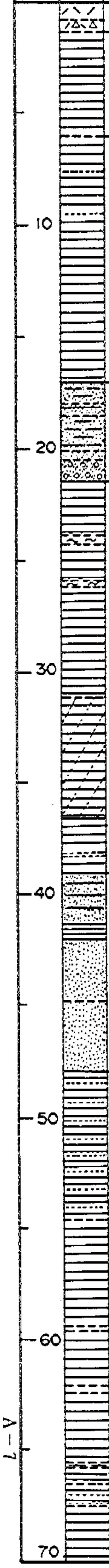
Item	Drilling hole No.			MJK · 1
Drilling Data	Drilling length		(m)	650.5
	Core length		(m)	640.55
	Core recovery		(%)	98.5
	Depth by	112mm size	(m)	4.0
	do.	93mm size	(m)	34.3
	do.	59mm size	(m)	612.2
	Cashing pipe	108mm	(m)	4.0
	do.	89mm	(m)	38.3
	Drilling mashine			ZIF · 650
Working Period	Working Period			8.13~8.26
	Actual Working Days		(d)	14
	No Working Days		(d)	0
	Total		(d)	14
	Actual Working Days	Mounting	(d)	0.5
		Drilling	(d)	12.5
		Dismounting	(d)	0.5
		Others	(d)	0.0
		Total	(d)	13.5
	Drilling length / Working Period		(m/d)	46.5
	Drilling length / Drilling days		(m/d)	52.0
	Drilling length / Drilling shifts		(m/s)	26.0
Working Time	Drilling		(h)	167°05'
	Hoisting & Lowering rod etc.		(h)	132°55'
	Repairing		(h)	0°00'
	Sub total		(h)	300°00'
	Mounting		(h)	12°00'
	Dismounting		(h)	12°00'
	Others		(h)	0°00'
	Total		(h)	324°00'
	Drilling length / Drilling hour		(m/h)	3.9
Workers	Total drilling workers			295
	Total drilling workers / Drilling length		(w/.m)	0.45

Appendix 6 Geological Logging of the Drill Hole "MJK-1"(1/200)

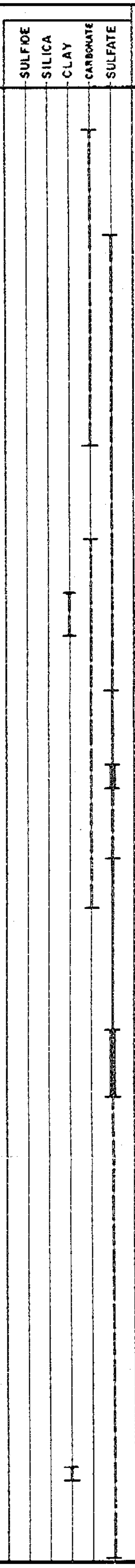
GEOLOGIC LOG (I)

MJK - I

AREA : ZHAMAN-AIBAT
INCLINATION : -90°
BEARING : —
ELEVATION : 357.04 m
FINAL DEPTH : 650.50 m



DEPTH (m)	DESCRIPTION		REMARKS	MINERALIZATION
1.20	Strongly weathered rock with fragments of red siltstone and fine-grained sandstone. Content of fragments is up to 50%.		Zhidelisai Formation (Gypsum - rich Red Siltstone)	
6.00	Reddish brown weathered siltstone with unclearly expressed bedded structure, partially fractured. Leached caverns are partly filled in by calcite. Interlayer : Grayish-red weathered, fine-grained, sandstone at the depth of 1.40 mwith thickness, equal to 30 cm.	∠5°		
17.00	Reddish brown siltstone with fine grained, partially porous sandstone. Lamination angle is equal to 5°. These rocks are strongly fractured. Fractures surface is coated by gypsum, traces of iron oxide and dendrite of manganese. Leached caverns are sizing up to 0.5x1.5 cm, filled in by calcite.	∠5°		
21.40	Light reddish brown fine grained sandstone. Interlayers of red coarse-grained sandstone (5-20 cm thick) with clayey-carbonate-ferrous cement is strongly weathered. This sandstone is sometimes porous, unclearly expresses bedded structure. Gravels included in sandstone orient at 5°.	∠5°		
39.05	Reddish brown, sandy siltstone, unclearly expressed bedded structure, strongly fractured. Traces of ferruginization, manganese dendrite and thin films of gypsum and chlorite at walls of fractures. Lamination angle is equal to 5°. Leached caverns filled in by calcite are described all through the horizon sizing up to 2.0 x 2.0 cm. Numerous layers of gypsum (selenite) are observed below 28.6 mwith thickness 0.1-1.0 cm, oriented at 5-10° (partly 35-60°).	∠5°		23.9-26.2m: weakly argillized
47.90	39.05-44.8m,46.1-47.9m: Light reddish brown fine-grained sandstone with carbonate-clayey cement. Including interlayers of kaolinized coarse-grained sandstone layers up to 6 cm, express lenticular and horizontally-bedded structure due to red fine-grained sandstone interlayers(0.2cm).There are numerous interlayers of fiber-like gypsum with thickness from 2 to 4, oriented at 5-0°. 44.8-46.1m:Granule - pebble conglomerates, composed by quartz, limestone, flintstone with fragments of red aleurosandstone from 0.5 x 2 cm to 2 x 4 cm in amount up to 30%. The gravels and flat fragments are shaped as lamination oriented at 5°.	∠5°		
66.50	Reddish brown siltstone (or fine-grained muddy sandstone) with carbonate-ferrous cement. Partially brecciated, sliding surfaces and chlorite films are observed along the fractures. Muddy sandstone has an horizontally bedded structure at 5° due to interlayers of limy sandstone. The thickness of the limy sandstone layers is from 10 to 30 cm and maximum thickness is 1.0 m. rocks fractures are filled in by fiber-like gypsum with thickness from 1 to 3 cm, oriented mostly at 5°. 65.65-65.85m:Brecciated, chloritization weak	∠5°-10° ∠5° ∠5° ∠70°		65.65-65.85m: Brecciated, chloritization weak



SAMPLE No.

ROCK PROPERTY		
Angle of Fiss. (°)	No. of Fiss. (/m)	Core Rec. %
		100
∠10-80°	10	100
〃	10	93
〃	10	93
〃	10	91
		91
		92
		94
		94
∠40°	1	94
∠25°	1	94
		94
		94
∠5°	1	94
〃	4	94
〃	4	94
〃	4	94
〃	4	95
∠80°	4	95
∠35°	4	95
〃	4	94
〃	4	94
〃	4	93
〃	3	93
〃	2	94
∠5°	4	94
〃	4	91
〃	4	91
〃	4	91
〃	4	90
〃	4	90
〃	4	90
〃	4	90
〃	4	91
〃	4	91
〃	4	91
〃	4	91
〃	4	92
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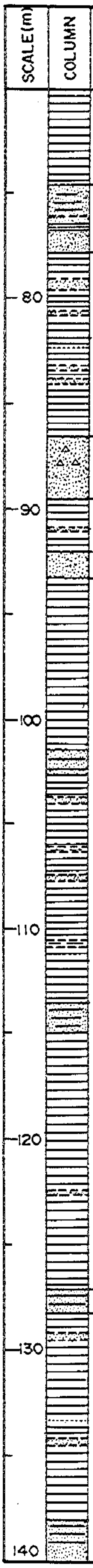
〃	4	92
〃	4	92
〃	4	92
〃	4	92
〃	4	92
〃	4	92
〃	4	93
∠5-10°	4	93
〃	4	93
〃	4	93
〃	4	94
〃	4	94
〃	4	94
〃	4	94
〃	4	94
〃	4	94
〃	4	94
〃	4	94
〃	4	94
〃	4	95
〃	4	95
∠30-80°	4	95
∠30-70°	4	95
∠10-30°	3	95
∠10°	2	95
∠5-10°	2	95
〃	2	95

GEOLOGIC LOG (2)

MJK - I

AREA : ZHAMAN-AIBAT
INCLINATION : -90°
BEARING : —
ELEVATION : 357.04m
FINAL DEPTH : 650.50m

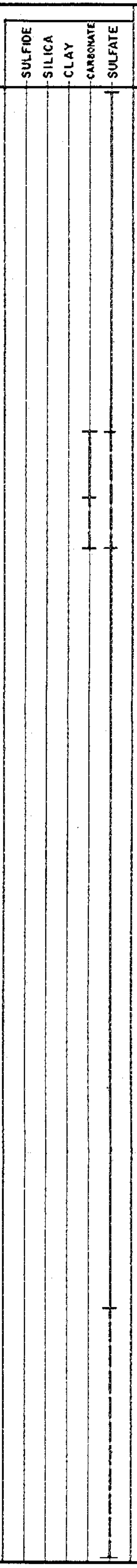


DEPTH (m)	DESCRIPTION	
74.55	Red sandstone, including light-red limy sandstone layers (intervals: 10-30cm, thickness : 2-5mm). These rocks are medium-fractured; gypsum at fractures is oriented at 5°.	∠5°
77.90	Red sandstone, fine-grained with carbonate-ferrous cement, horizontally bedded structure due to fine lamination with siltstone. The rock is medium fractured. Rare veinlets of gypsum (5°) with thickness up to 2 cm. Contact with underlying horizon is clear, at 5°. Including rare fragments of brown colored siltstone.	
86.50	77.9-79.1m: Red siltstone with spotted structure due to spots of light-red colored limy sandstone. Rare interlayers of fiber-like gypsum (selenite) with thickness up to 0.5 cm, oriented at 5°. Contact with underlying horizon is gradual. 79.1-79.8m: Red sandstone, fine-grained, bedded due to interlayers of aleurosandstone 5° with gypsum veinlets 0.2-0.4 cm thick, oriented at 10-20°. 79.8-86.5m: Red siltstone with interlayers of fine-grained (83.9-84.15 m) gypsum and single gypsum veinlets 0.3-0.4 cm thick.	
89.50 92.00	Light-red sandstone, fine grained with carbonate-ferrous cement; horizontally-bedded structure at 5°. Graded bedding and lamina structures are developed. The rock is medium fractured; veinlets of gypsum with thickness up to 1.5 cm. Carbonate-gypsum inclusions sizing up to 1.0 x 1.5 cm within the interval from 87.2 to 87.8 m. Contact with underlying horizon is clear, at 10°.	∠5° ~10°
93.25	Red siltstone with unclearly expressed bedded structure. Including rare siliceous-carbonate accumulations, veinlets of gypsum with thickness 5 mm, interlayer of fine-grained grayish-red colored sandstone with thickness 20 cm. Contact with underlying horizon is at 0°.	∠5°
	Reddish brown sandstone, fine-grained with carbonate-ferrous cement. Lenticular bedded structure at 10-15° due to interlaying with fine-grained siltstone. The rock is medium fractured with gypsum (up to 1.5 cm thick) filling in fractures. Contact with underlying horizon is at 0°.	
	Reddish brown siltstone with spotted or rarely horizontally-bedded (at 5°) structure due to interlaying with fine-grained sandstone. Interlayers of fine-grained sandstone with thickness up to 25 cm, maximum: 140cm. There is an interlayer of green colored siltstone at the depth 94.8m, 106.25m and 110.6m with thickness 10 - 30 cm. These rocks are medium fractured, fractures are filled in by gypsum with thickness from 1mm to 5 cm. Contact with underlying horizon is unclear and gradual.	∠5°
127.20 128.30	Red colored laminated sandstone, fine-grained, with carbonate-ferrous cement. Banded structure at 5-15. The rock is medium fractured, gypsum (1mm to 4 cm thick) filling in fractures. Contact with underlying horizon is at 5°.	∠5°
	Reddish brown massive siltstone. The rock is medium fractured; gypsum veinlets at rare fractures (thickness up to 5 mm). There are numerous Interlayers of fine-grained sandstone with thickness 7-10 cm. The layer is continued below.	

REMARKS

Zhidelisol Formation (Gypsum - rich Red siltstone)

MINERALIZATION



SAMPLE No.

ROCK PROPERTY		
Angle of Fiss. (°)	No. of Fiss. (/m)	Core Rec. %
∠5°	2	95
〃	2	95
〃	2	95
〃	2	95
〃	2	95
〃	2	95
〃	2	95
∠5~10°	4	95
〃	4	95
〃	4	95
〃	4	96
〃	4	96
〃	4	96
〃	4	96
〃	4	96
〃	4	96
〃	4	96
〃	4	96
〃	4	96
〃	4	96
〃	4	96
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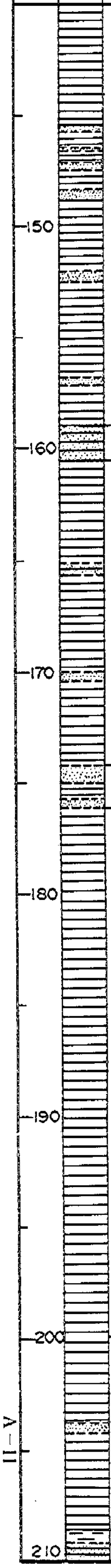
〃	4	96
〃	4	96
〃	4	96
〃	4	96
〃	4	96
〃	4	96
〃	4	96
〃	4	96
〃	4	96
〃	2	96
∠5°	4	96
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
∠10°	2	97
〃	2	97
〃	2	97
〃	2	97
〃	2	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
∠10~40°	4	97
∠5°	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97

GEOLOGIC LOG (3)

MJK - I

AREA : ZHAMAN-AIBAT INCLINATION : -90° BEARING : — ELEVATION : 357.04 m FINAL DEPTH : 650.50 m



DEPTH (m)	DESCRIPTION	
159.00	Reddish brown colored siltstone with banded and spotted structure due to the interlayers of limy sandstone. Partings with thickness from 1 to 3 cm and interlayers of fine-grained sandstone are described at intervals 145.40-145.70 m, 147.3-147.6, 148.2-148.6, 156.9-157.20 m. Graded bedding structure (inclination: 10-15°) is observed in the above intervals. The rock is medium fractured, fractures are filled in by gypsum and are oriented at 5°, 45° and 15°; thickness of gypsum layers varying from1 mm to 2 cm. Contact with underlying horizon is gradual.	∠10°~15°
160.50	Brown sandstone, fine grained, with carbonate-ferrous cement. Structure is horizontally bedded at 5° due to interlaying with red colored siltstone. The rock is medium fractured; fractures are filled in by gypsum. Contact with underlying horizon is gradual.	∠5°
174.20	Reddish brown siltstone. Structure is basically horizontally-bedded at 5° due to partings of fine-grained sandstone with thickness up to 1 mm. Interlayers of red fine-grained sandstone with thickness 10-15 cm occur all through the layer. Interlayer of medium-grained sandstone with thickness 30 cm is described at the depth 165.0m and 169.9m	∠5°
176.10	Red sandstone, fine grained, with carbonate-ferrous cement. Structure is horizontally bedded at 5° due to the graded bedding structure. Interlayer of siltstone with thickness 33 cm. The rock is medium fractured; fractures are filled in by gypsum with thickness up to 2 cm. Contact with underlying horizon is at 0°.	∠5° ∠0°
200.80	Brown massive siltstone. Unclearly expressed bedded structure. Rare partings of fine-grained sandstone with thickness up to 5 cm and interlayers up to 25 cm. The rock is medium fractured; fractures are filled in by gypsum with thickness up to 3 cm, oriented at 10° and 30°. The layer is continued below.	∠10°~30°
208.70	200.08-208.65m: Brown colored massive siltstone. Unclearly expressed bedded structure. The rock is medium fractured. Frequent veinlets of gypsum with thickness up to 1 cm. Fractures are oriented at 45°, 15° and 5°. Gypsum veinlets at the interval from to 207.5 to 208.65 m are oriented by two groups (at 15° and 5° and along core axis). Contact with underlying horizon is at 5°.	∠5°
209.15	208.70-209.15m: Grayish-green siltstone. Horizontally-bedded structure resulted from partings of red siltstone with thickness up to 1 cm. The rock is medium fractured; fractures are filled in by gypsum with thickness up to 2 cm. Contact with underlying horizon is at 10°.	∠10°
	209.15-210.0m: Brown sandstone, fine-grained, with carbonate-ferrous cement. Structure is sometimes bedded at 10°, 5° due to fine lamination with siltstone. The rock is medium fractured; fractures are filled in by gypsum and are oriented at 30°, 10°; their thickness varies from 1 mm to 2 cm. Contact with underlying horizon is clear, at the angle 0°.	∠0°

REMARKS: Zhideilsai Formation (Gypsum — rich Red Siltstone)

MINERALIZATION: 156.50-156.55m: weakly argillized

SULFIDE · SILICA · CLAY · CARBONATE · SULFATE

SAMPLE No.

ROCK PROPERTY

Angle of Fiss. (°)	No. of Fiss. (/m)	Core Rec. %
∠5~10°	4	97
〃	4	97
〃	4	97
〃	4	97
∠20°	4	97
∠5~10°	4	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
∠10-30°	3	97
∠5~10°	3	97
〃	3	97
〃	3	93
〃	3	93
〃	3	93
〃	3	97
〃	3	97
〃	3	97
〃	3	97
∠5~20°	2	97
〃	2	97
〃	2	97
〃	2	97
〃	2	97
〃	2	97
〃	2	97
〃	2	97
〃	2	97
〃	2	97
〃	2	97
〃	2	97
〃	2	97
〃	2	97
〃	2	97
∠5~10°	3	97
〃	4	97
〃	4	97
〃	2	97
〃	1	97
〃	2	97
〃	3	97
〃	4	97
〃	6	97
〃	2	97
〃	4	97
〃	3	97
〃	1	97
〃	0	97
〃	1	97
〃	0	97
〃	2	97
〃	4	97
〃	3	97
〃	4	97
〃	7	97
〃	7	97
〃	4	97
〃	2	97
〃	2	97
〃	2	97
〃	1	97
〃	1	97
〃	4	97
〃	4	97
〃	4	97
〃	4	97
〃	3	97
〃	2	97
〃	2	97