2.3.5 Final Flotation Test

(1) Bulk differential flotation

Under the best conditions obtained in the basic test of BDF, the final flotation test was carried out, with two-stage cleaning respectively in the bulk flotation section and the lead differential flotation section. The test comprised two parts, one without sulfidizer and the other using 100g/t of the same. The test flow sheet and the test results are shown in Fig.34 in Table 25, respectively.

The test findings are summarized as follows:

1) The two-stage cleaning in the bulk flotation section generally produced favorable results as far as the lead and zinc grades are concerned. However, pyrite in free particles were abundant in the zinc concentrate, indicating insufficient depression of pyrite in the bulk flotation. Further study on quantity of added lime in the bulk flotation cleaners is considered necessary.

2) Almost 85% of lead grading nearly 70% was recovered in concentrate of the first stage of the two-stage cleaning in the differential flotation. Under the microscope, however, sphalerite-chalcocite middling was abundantly distributed; therefore, two-stage cleaning is considered necessary for stabilizing flotation performance.

3) Sulfidizer was added to recover cerussite, non-sulfide lead component, which lowered the tailing grade but only insufficiently. Intentional recovery of non-sulfide component appears to be unfeasible.

(2) Straight-differential flotation

Under the best conditions obtained in the basic test of SDF, the final flotation test with two-stage cleaning in the lead flotation and in the zinc flotation sections was carried out. The test consisted of two parts: one without sufidizer and the other using 100g/t of it. The test flow sheet appears in Fig.35 and the test results in Table 26.

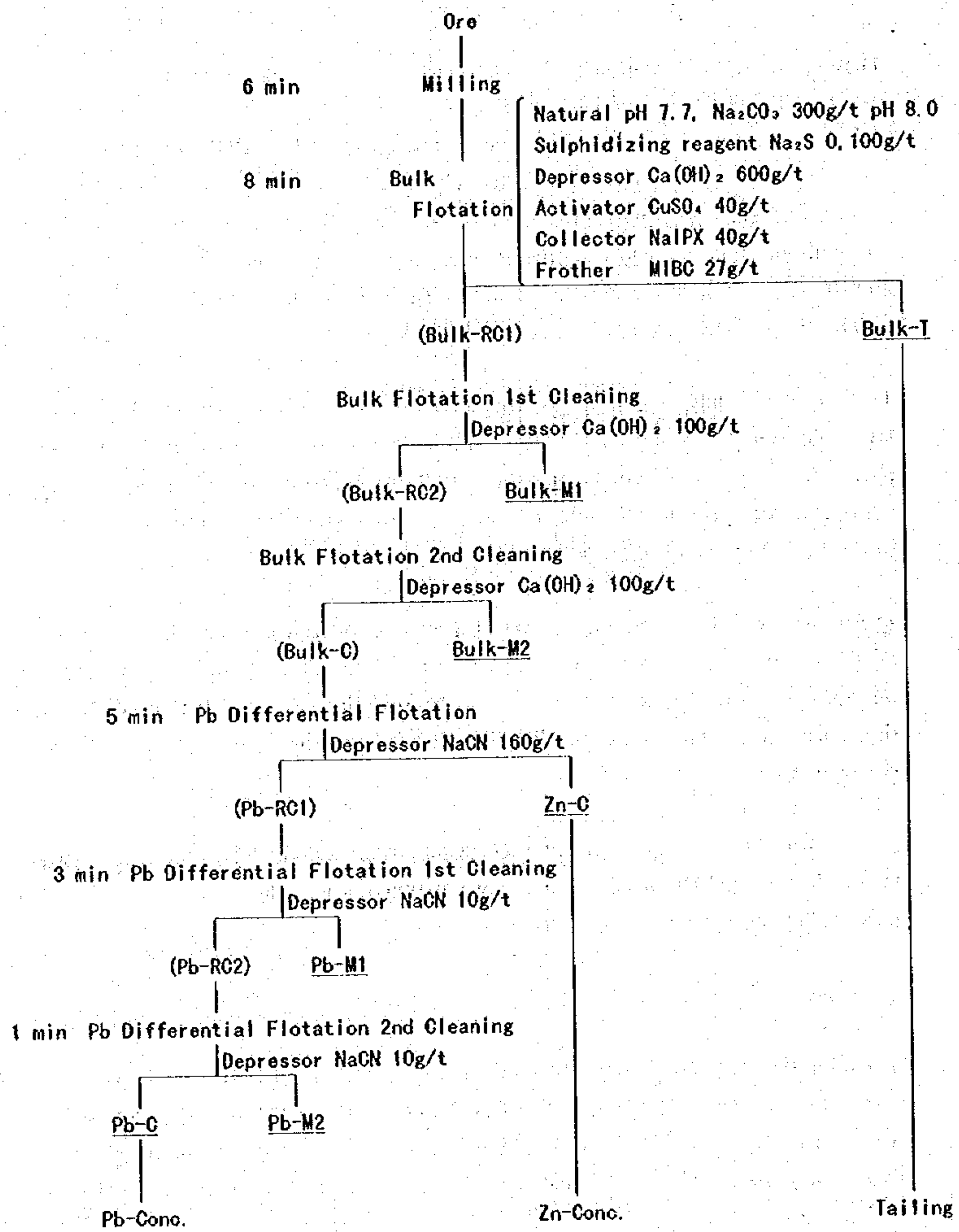


Fig. 34 Final Flotation Test Flowsheet
(Bulk Differential Flotation)

Test No.	Type	Weight (g)	Weight (%)	Weight (%)	Grade						Distribution						Distribution					
					Cu	Pb	Zn	Fe	Ag	Au	Cu	Pb	Zn	Fe	Ag	Au	Cu	Pb	Zn	Fe	Ag	Au
1	Crude Ore	497.61	100.0		0.09	7.82	3.12	4.66	116	0.7	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%						
	Bulk Rough 1	98.69	19.7		0.36	35.90	15.04	8.43	528	3.2	81.6%	91.1%	95.6%	35.8%	90.3%	88.6%						
	Bulk Tail	398.92	80.3		0.02	0.87	0.17	3.73	14	0.1	18.4%	8.9%	4.4%	64.2%	9.7%	11.4%						
	Bulk Rough 2	88.65	17.7		0.38	39.54	16.62	7.29	563	3.1	77.4%	90.1%	94.9%	27.8%	86.5%	77.7%						
	Bulk Mid 1	10.04	2.0		0.18	3.76	1.09	18.50	219	3.8	4.2%	1.0%	0.7%	8.0%	3.8%	10.9%						
	Bulk Conc	78.64	15.7	100.0	0.41	43.94	18.13	7.09	625	3.1	74.2%	88.8%	91.8%	24.0%	85.2%	69.7%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%
	Bulk Mid 2	10.01	2.0		0.14	4.97	4.76	8.86	76	2.8	3.2%	1.3%	3.1%	3.8%	1.3%	8.0%						
	Pb Rough 1	51.22	10.3	65.1	0.12	66.54	5.76	3.30	918	2.8	14.2%	87.6%	19.0%	7.3%	81.5%	40.1%	19.1%	98.6%	20.7%	30.3%	95.6%	57.5%
	Zn Conc	27.42	5.4	34.9	0.95	1.72	41.24	14.17	78	3.8	60.0%	1.2%	72.8%	16.7%	3.7%	29.6%	80.9%	1.4%	79.3%	69.7%	4.4%	42.5%
	Pb Rough 2	48.86	9.8	62.1	0.12	69.20	5.40	2.40	925	2.8	13.5%	86.9%	17.0%	5.1%	78.3%	38.8%	18.2%	97.8%	18.5%	21.0%	91.9%	55.6%
	Pb Mid 1	2.36	0.5	3.0	0.12	11.47	13.21	21.93	773	2.0	0.7%	0.7%	2.0%	2.2%	3.2%	1.3%	0.9%	0.8%	2.2%	9.3%	3.7%	1.9%
	Pb Conc	36.35	7.3	46.2	0.11	72.24	5.00	1.12	954	3.1	9.2%	67.5%	11.7%	1.8%	60.1%	32.0%	12.4%	75.9%	12.7%	7.3%	70.5%	45.9%
	Pb Mid 2	12.51	2.5	15.9	0.15	60.37	6.56	6.12	841	1.9	4.3%	19.4%	5.3%	3.3%	18.2%	6.8%	5.8%	21.9%	5.8%	13.7%	21.4%	9.7%
2	Crude Ore	496.85	100.0		0.11	7.68	3.14	4.72	115	0.6	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%						
	Bulk Rough 1	92.89	18.6		0.42	37.60	15.97	7.93	554	2.7	70.7%	91.5%	95.1%	31.4%	90.1%	86.0%						
	Bulk Tail	403.96	81.4		0.04	0.80	0.19	3.98	14	0.1	29.3%	8.5%	4.9%	68.6%	9.9%	14.0%						
	Bulk Rough 2	88.65	17.7		0.38	38.90	16.60	7.32	561	2.7	61.0%	90.3%	94.3%	27.7%	87.1%	83.5%						
	Bulk Mid 1	4.24	0.9		1.26	10.42	2.80	20.68	408	1.7	9.7%	1.2%	0.8%	3.7%	3.0%	2.5%						
	Bulk Conc	72.07	14.4	100.0	0.43	45.64	19.50	4.03	643	3.1	56.2%	86.1%	90.1%	12.4%	81.2%	76.6%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%
	Bulk Mid 2	16.58	3.3		0.16	9.60	3.99	21.62	205	1.2	4.8%	4.2%	4.2%	15.3%	5.9%	6.9%						
	Pb Rough 1	49.17	9.9	68.2	0.14	65.84	6.50	2.10	870	2.8	12.5%	84.7%	20.5%	4.4%	74.9%	47.2%	22.2%	98.4%	22.8%	35.6%	92.3%	61.7%
	Zn Conc	22.90	4.5	31.8	1.05	2.27	47.41	8.17	156	3.7	43.7%	1.4%	69.6%	8.0%	6.3%	29.4%	77.8%	1.6%	77.2%	64.4%	7.7%	38.3%
	Pb Rough 2	44.87	9.0	62.2	0.14	70.60	4.60	1.20	940	2.9	11.4%	82.9%	13.2%	2.3%	73.8%	45.6%	20.3%	96.3%	14.7%	18.6%	91.0%	59.6%
	Pb Mid 1	4.30	0.9	6.0	0.14	16.17	26.33	11.49	140	1.1	1.1%	1.8%	7.3%	2.1%	1.1%	1.6%	1.9%	2.1%	8.1%	17.0%	1.3%	2.1%
	Pb Conc	36.35	7.3	50.4	0.13	74.05	3.54	0.70	989	3.2	8.6%	70.4%	8.2%	1.1%	62.9%	40.3%	15.3%	81.8%	9.2%	8.8%	77.6%	52.7%
	Pb Mid 2	8.52	1.7	11.8	0.18	55.88	9.12	3.33	731	1.8	2.8%	12.5%	5.0%	1.2%	10.9%	5.3%	5.0%	14.5%	5.5%	9.8%	13.4%	6.9%

Table 25 Results of Final Flotation Test(Bulk Diffrential Flotation)

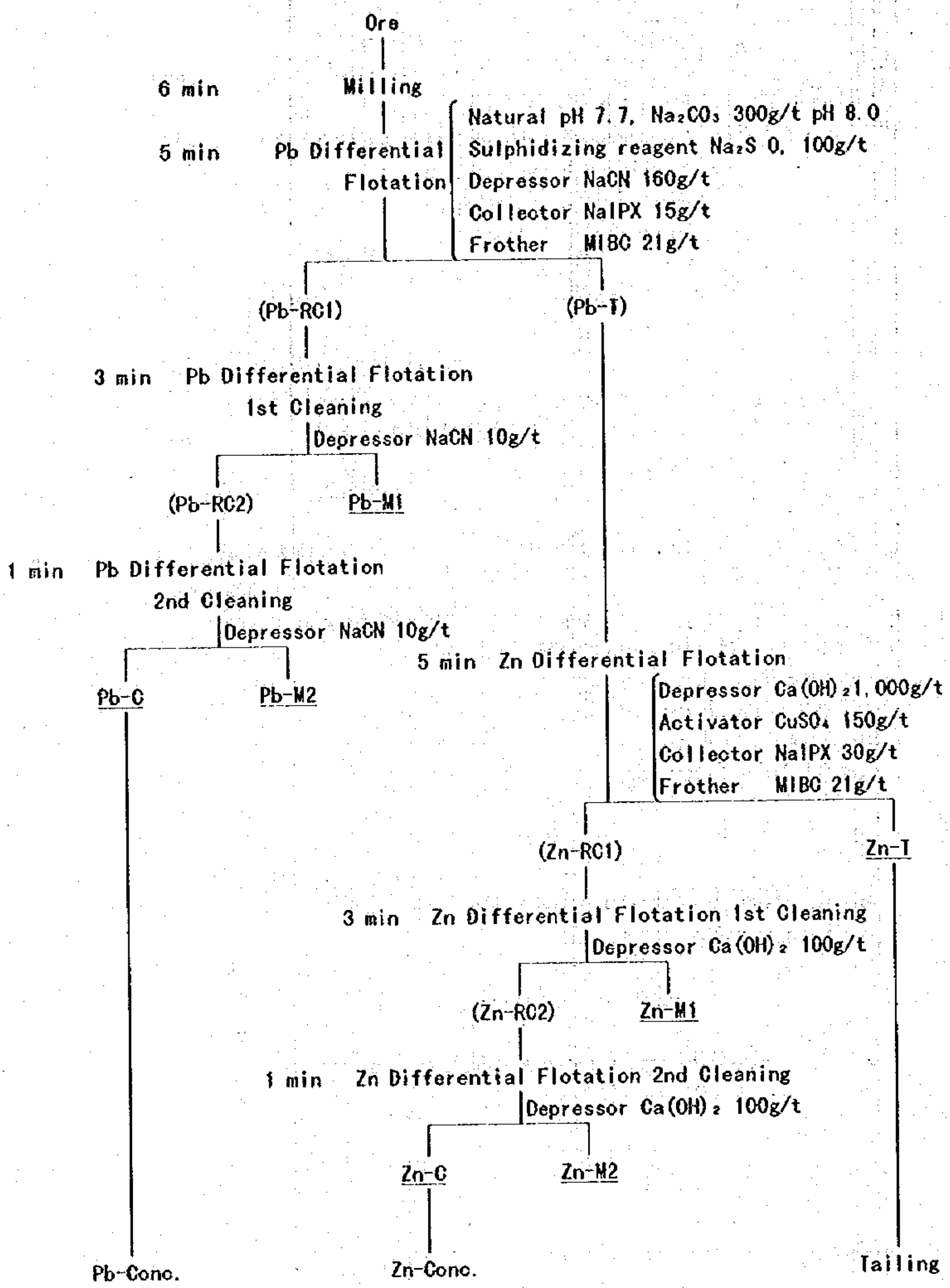


Fig. 35 Final Flotation Test Flowsheet
(Straight Differential Flotation)

Test No.	Type	Weight (g)	Weight (%)	Weight (%)	Grade Cu	Pb	Zn	Fe	Ag	Au	Distribution Cu	Pb	Zn	Fe	Ag	Au	Distribution Cu	Pb	Zn	Fe	Ag	Au
1	Crude Ore	497.76	100.0		0.13	7.12	3.00	4.70	105	0.6	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%						
	Pb Rough 1	54.91	11.1		0.67	51.96	14.27	2.11	694	3.8	55.5%	80.5%	52.5%	4.9%	73.1%	64.9%						
	Pb Tail	442.85	88.9	100.0	0.07	1.56	1.60	5.02	32	0.3	44.5%	19.5%	47.5%	95.1%	26.9%	35.1%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%
	Pb Rough 2	41.58	8.4		0.39	65.34	8.10	0.79	797	4.0	24.5%	76.7%	22.6%	1.4%	63.6%	51.7%						
	Pb Mid 1	13.33	2.7		1.54	10.22	33.52	6.23	373	3.2	31.0%	3.8%	29.9%	3.5%	9.5%	13.2%						
	Pb Conc	35.64	7.2		0.08	70.65	4.47	0.45	812	4.2	4.3%	71.1%	10.7%	0.7%	55.5%	46.5%						
	Pb Mid 2	5.94	1.2		2.25	33.48	29.88	2.83	707	2.8	20.2%	5.6%	11.9%	0.7%	8.1%	5.2%						
	Zn Rough 1	25.71	5.2	5.8	0.82	13.52	25.15	7.34	317	2.8	31.9%	9.8%	43.3%	8.0%	15.7%	22.2%	71.7%	50.4%	91.2%	8.5%	58.3%	63.1%
	Zn Tail	417.14	83.7	94.2	0.02	0.82	0.15	4.88	14	0.1	12.6%	9.7%	4.2%	87.1%	11.2%	12.9%	28.3%	49.6%	8.8%	91.5%	41.7%	36.9%
	Zn Rough 2	14.31	2.9	3.2	0.77	14.11	37.77	4.02	345	2.9	16.7%	5.7%	36.2%	2.4%	9.5%	13.0%	37.7%	29.3%	76.2%	2.6%	35.3%	36.9%
	Zn Mid 1	11.40	2.3	2.6	0.88	12.79	9.30	11.52	282	2.6	15.2%	4.1%	7.1%	5.6%	6.2%	9.2%	34.0%	21.1%	15.0%	5.9%	23.0%	26.2%
	Zn Conc	9.29	1.9	2.1	0.75	12.37	49.21	3.80	352	3.1	10.5%	3.2%	30.6%	1.5%	6.3%	8.9%	23.7%	16.7%	64.4%	1.6%	23.4%	25.4%
	Zn Mid 2	5.02	1.0	1.1	0.82	17.32	16.60	4.42	332	2.6	6.2%	2.5%	5.6%	0.9%	3.2%	4.1%	14.0%	12.6%	11.8%	1.0%	11.9%	11.5%
2	Crude Ore	497.30	100.0		0.13	7.25	3.14	4.66	102	0.5	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%						
	Pb Rough 1	60.46	12.1		0.68	51.60	13.91	2.28	652	2.6	62.0%	86.4%	53.9%	5.9%	77.6%	64.7%						
	Pb Tail	436.84	87.9	100.0	0.06	1.11	1.65	4.99	26	0.2	38.0%	13.6%	46.1%	94.1%	22.4%	35.3%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%
	Pb Rough 2	46.89	9.4		0.47	64.69	9.64	1.01	754	2.8	33.2%	84.0%	29.0%	2.0%	69.6%	54.1%						
	Pb Mid 1	13.57	2.7		1.41	6.37	28.66	6.67	300	1.9	28.8%	2.4%	24.9%	3.9%	8.0%	10.6%						
	Pb Conc	40.86	8.2		0.13	71.16	5.27	0.62	791	2.9	8.0%	80.5%	13.8%	1.1%	63.6%	48.9%						
	Pb Mid 2	6.03	1.2		2.77	20.85	39.25	3.65	503	2.1	25.2%	3.5%	15.2%	0.9%	6.0%	5.2%						
	Zn Rough 1	20.85	4.2	4.8	0.81	9.35	30.55	11.20	264	2.1	25.5%	5.5%	40.8%	10.2%	10.9%	18.1%	67.1%	40.0%	88.4%	10.8%	48.6%	51.4%
	Zn Tail	415.99	83.7	95.2	0.02	0.70	0.20	4.68	14	0.1	12.5%	3.1%	5.3%	83.9%	11.5%	17.2%	32.9%	60.0%	11.6%	89.2%	51.4%	48.6%
	Zn Rough 2	12.45	2.5	2.9	0.73	9.67	44.56	6.93	299	2.3	13.7%	3.4%	35.5%	3.8%	7.4%	11.9%	36.1%	24.7%	77.0%	4.0%	32.9%	33.7%
	Zn Mid 1	8.40	1.7	1.9	0.93	8.88	9.79	17.53	212	1.8	11.8%	2.1%	5.3%	6.4%	3.5%	6.2%	31.0%	15.3%	11.4%	6.8%	15.7%	17.7%
	Zn Conc	8.87	1.8	2.0	0.74	9.56	50.98	4.84	324	2.4	9.9%	2.4%	28.9%	1.9%	5.7%	8.8%	26.0%	17.4%	62.8%	2.0%	25.4%	24.9%
	Zn Mid 2	3.58	0.7	0.8	0.71	9.94	28.65	12.11	237	2.1	3.8%	1.0%	6.6%	1.9%	1.7%	3.1%	10.1%	7.3%	14.2%	2.0%	7.5%	8.8%

Table 26 Results of Final Flotation Test(Straight Differential Flotation)

The test findings are summarized as follows:

1) In the second stage concentrate of the lead flotation(SDF), the lead grade reached 71% and the recovery 76%, indicating the zinc depression effect in the cleaners. In the lead concentrate, however, sphalerite grains in free particles were abundantly observed, while 40% of zinc was circulating in the cleaners. There still remains a question as to whether the two-stage cleaning can sufficiently depress and separate zinc.

2) 30% of zinc grading 50% was recovered in the concentrate of the two-stage cleaning in the zinc flotation. Under the microscope, galena in free particles was abundantly distributed in the zinc concentrate, which indicates insufficient separation in the lead flotation section. Compared to BDF, higher grade zinc in zinc concentrate was more easily recoverable, whereas the question of recovery still remains.

3) In relation to the sulfidizing effect, the tailing grade tended to decrease but only insufficiently; therefore, intentional recovery of non-sulfide component appears unfeasible.

4) In order to determine the number of stages of SDF and quantity of reagents added, locked tests or continuous tests have to be conducted, thereby clarifying zinc behavior in the lead flotation.

As compared to the performance of SDF, BDF demonstrated higher stability in the lead and zinc recovery, as the latter process recovers floatable zinc together with lead.

2.3.6 Settling test of concentrates and tailing

The Tsav area being situated in an arid land, there is no large water source in the vicinity. If the mine is put into development, mineral dressing water supply would be insufficient; therefore, water recycling must be contemplated. For this purpose, the settling rates of the concentrates and tailing were measured, which constitute the basic items for planning the mineral dressing water recycling.

For the settling test samples, the undersize(−200 mesh) of sieved bulk flotation tailing was prepared, which is assumed to substitute for the tailing slime fed to the tailing thickener, whereas, in substitution for the lead and zinc concentrates fed to each concentrate thickeners, the concentrates produced from the flotation test were used without sieving because the quantity of the test products was small. The initial pulp density of respective samples, as the settling test condition, was determined at 10%. Flocculant was not used in the test.

The settling rate measurement is shown in Table 27, while Fig.36 ~ 38 indicate the relationship between the height of interface and the density of sediment by product and by measurement time.

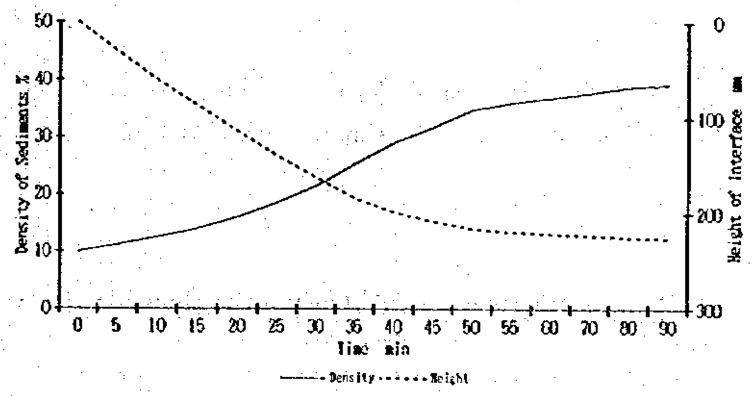


Fig. 36 Height of Interface and Density of Sediments(Tailing)

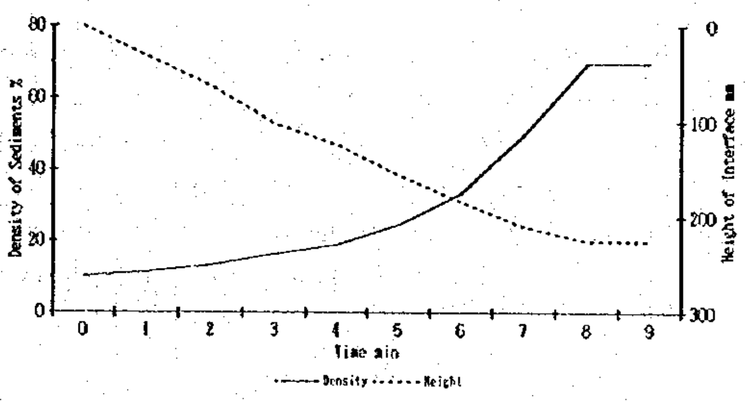


Fig. 37 Height of Interface and Density of Sediments(Pb−Conc.)

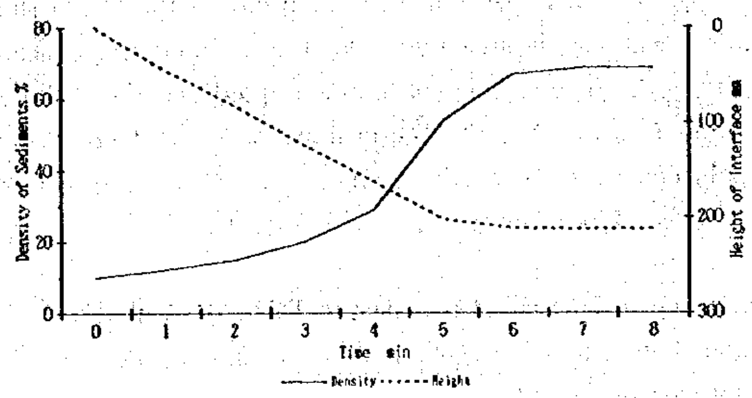


Fig. 38 Height of Interface and Density of Sediments(Zn–Conc.)

From the measurement in Fig.36 ~ 38, the settling rates for each sample to be thickened to certain densities of sediment are calculated as follows:

Tailing		Pb Conc		Zn Conc	
Sediment density(%)	Settling rate(mm/min)	Sediment density(%)	Settling rate(mm/min)	Sediment density(%)	Settling rate(mm/min)
20	5.50	20	31.07	20	41.33
30	4.85	30	30.87	30	40.49
35	4.26	40	30.38	40	40.45
		50	29.98	50	40.41

All the three types of the samples showed certain turbidity in the decanted water(the top water) influenced by clay minerals and fine slime contained but their settling characteristics appeared relatively good in view of the settling rates. Since the turbidity of the tailing slime was higher than that of the concentrates, it would probably be necessary, in the actual operation, to add high–molecular flocculant in the tailing thickener thereby lowering the turbidity of the decanted water.

Tailing			Pb-Conc.			Zn-Conc.		
Time min	Height of Interface mm	Density of Sediments %	Time min	Height of Interface mm	Density of Sediments %	Time min	Height of Interface mm	Density of Sediments %
0	0	10.0	0	0	10.0	0	0	10.0
1	6	10.2	1	32	11.4	1	45	12.2
2	13	10.4	2	63	13.2	2	82	14.9
3	19	10.7	3	103	16.4	3	124	20.0
5	30	11.1	4	124	18.9	4	162	28.8
10	60	12.5	5	156	24.6	5	202	53.8
15	87	14.0	6	184	33.4	6	211	66.8
20	114	16.0	7	210	49.8	7	212	68.7
25	140	18.6	8	225	69.6	8	212	68.7
30	162	21.5	9	225	69.6			
35	184	25.4						
40	199	29.1						
45	208	31.8						
50	216	34.8						
55	219	36.0						
60	221	36.9						
70	223	37.8						
80	225	38.8						
90	226	39.3						
Sample 100g			Sample 50g			Sample 50g		
Water 900cc			Water 450cc			Water 450cc		
pH 9.7			pH 7.5			pH 7.4		

Table 27 Settling Rates

In turn, water quality analysis was made on the waters that are assumed to correspond to the decanted waters in the tailing thickener and the concentrates thickeners, which would be recycled in the actual operation, to judge if the water recycled from the thickeners would have any quality problems. The results of analysis are shown in Table 28.

	Overflow of Tailing Thickener	Overflow of Pb Conc. Thickener	Overflow of Zn Conc. Thickener
pH	9.9	8.2	9.6
Cu (ppm)	2.7	0.15	1.6
Pb (ppm)	<0.02	<0.1	<0.04
Zn (ppm)	<0.02	<0.1	<0.04
T-Fe (ppm)	<0.05	<0.2	<0.1
Cd (ppm)	<0.01	<0.05	<0.02
As (ppm)	<0.02	<0.1	<0.05
CN^- (ppm)	<0.1	0.5	1.4
SO_4^{-2} (ppm)	582	25	121

Table 28 Water Quality Analysis of Decanted Water in the Assumed Thickeners

The analysis revealed that the decanted water in the assumed zinc concentrate thickener was high in cyanide concentration; therefore, the water would desirably be returned to the differential flotation circuit.

Decanted water from the other thickeners have no problem whatever if recycled within the beneficiation plant.

As water quality of the tailing pond overflow can be improved by neutralization treatment, there would be no problem either, if the tailing pond overflow is returned to the beneficiation plant.

Since an excessive portion of the beneficiation effluent would finally be drained off only from the tailing pond, it would cause no problem whatsoever to the environment.

2.4 Final Selection of the Process Flow

(1) Comparison between SDF and BDF

Based on the results of the series of beneficiation tests, a comparative study of the BDF and SDF processes as applied to the Tsav ore has been made, which is summed up in the following paragraphs.

Straight– Differential Flotation

· Advantages :

1) As iron is easily depressed by lime in the zinc flotation, it is easy to elevate grade of zinc in zinc concentrate.

· Disadvantages :

1) Due to the existence of copper which is apt to activate zinc, a large amount of NaCN has to be used to depress zinc in the lead flotation.
Nevertheless, a greater part of zinc mixes into lead floats, circulating between the cleaners. Therefore, zinc grade and distritbtion in lead concentrate are apt to rise. It requires NaCN of a quantity almost similar to that for bulk flotation.

2) For depressing zinc in lead floats, the number of cleaning stages and cells have to be increased while, for decreasing returned zinc, certain measures such as additonal treatment processes for zinc in middling in the lead flotation have to be taken.

3) The increase in NaCN addition causes to elevate NaCN concentration in effluent sent to the tailing pond.

4) The beneficiation process flow becomes complicated, which makes operation performance less stable.

Bulk– differential flotation

· Advantage :

1) Thanks to the existence of copper which is apt to activate zinc, zinc depressor($CuSO_4$) requirement in the bulk flotation is small.

2) In the bulk flotation, iron and waste alone have to be depressed ; therefore, quantity of return ore is small which can be met by fewer stages of cleaning.

3) Bulk concentrate fed to the differential flotation is in a decreased quantity, requiring fewer differential flotation cells than those of the zinc flotation in SDF.

4) Mineral dressing effluent containing NaCN is returned to the differential flotation circuit, thereby reducing NaCN requirement and, accordingly, the NaCN content in the mineral dressing effluent.

5) The mineral dressing process flow is simpler, which makes operation performance more stable.

· Disadvantages :

1) Lime used for depression of iron in the bulk flotation may cause to lower lead recovery if added in excess.
2) If depression of iron in the bulk flotation is incomplete, iron depressed by NaCN may mix into zinc concentrate in the differential flotation, lowering zinc grade.

The comparison of the advantages and disadvatages of the two processes leads to a conclusion that the BDF process is superior to the SDF process, when applied to the Tsav ore, in that BDF takes advantage of copper being apt to activate zinc, has a simpler flotation process, requires smaller plant equipment and has stabler operation performance than SDF.

(2) Comparison between production of bulk concentrate alone and separate production of lead and zinc concentrates

It is a conceivable alternative to produce bulk concentrate(of lead and zinc) alone, without separation into lead concentrate and zinc concentrate.

Operation performance in this case is estimated in the table below. In this case, metal grades turn out to be inferior to those in case of separate production of lead and zinc concentrates.

		Grade					Distribution Quantity				
		Au	Ag	Pb	Zn	Cu	Au	Ag	Pb	Zn	Cu
	(t)	g/t	g/t	%	%	%	g/t	g/t	%	%	%
Mill-F	30,000	1.22	161	6.4	2.9	0.22	100.0	100.0	100.0	100.0	100.0
Bulk-C	3,722	7.77	1,717	45.6	21.7	0.81	79.0	82.6	88.3	92.8	68.1
Tail.	26,278	0.29	32	0.9	0.2	0.08	21.0	17.4	11.7	7.2	31.9

Remarks: Mill-F;Mill feed Bulk-C;Bulk concentrate Tail.;Tailing

The project revenues and expenses in case of production of bulk concentrate are roughly estimated as follows:

Annual sales revenue of bulk concentrate:

Zinc : 3,722x0.995x(0.217-0.08)=	$558,100
Lead : 3,722x0.995x(0.456-0.03)x650=	1,025,468
Gold : 3,722x0.995x(7.77-1)x0.9x390/31.1024=	283,237
Silver : 3,722x0.995x(1071-100)x0.9x390/31.1024=	582,135
(-)T/C : (-)3,722x0.995x(190+10+(13/100x(1,100-1,000)))=	(-)788,822
Total	$1,660,118

The total annual sales revenue in case of production of bulk concentrate alone is $168,000 less than that in case of separate production of lead and zinc concentrates.

Expenses:

In case of bulk concentrate production, one series of differential flotation system and concentrate dewatering system are dispensed with; therefore, the annual depreciation expenses will decrease accordingly.

$12,250 + $8,250 = $20,500

Likewise, the variable expense items such as the flotation reagents usedin the omitted process sections, as well as repair expenses, etc., will become unnecessary, which represents $35,000 of reduction in expenses.
Altogether, the reduction in annual expenses will be $56,000.

The calculations reveals that, in case of production of bulk concentratealone, the decline in sales revenue is far greater than the reduction in expenses, which leads to a conclusion that separate production of lead and zinc concentrates is more favorable.

(3) Optimum mineral dressing process flow

The optimum mineral dressing process flow worked out on the basis of all the results of tests and studies so far undertaken is shown in Fig.39.

2.5 Esimated Mineral dressing Performance

The grades of run-of-mine ore of the Tsav deposits are estimated at Au 1.22g/t, Ag 161g/t, Pb 6.37%, Zn 2.94% and Cu 0.22% performance estimated on the basis of the grades of run-of-mine ore and the BDF test results is shown in Table 29.

	Weight	Grade					Distribution Quantity				
	(t)	Au g/t	Ag g/t	Pb %	Zn %	Cu %	Au g/t	Ag g/t	Pb %	Zn %	Cu %
Crude	30,000	1.22	161	6.4	2.9	0.22	100.0	100.0	100.0	100.0	100.0
Pb-C	2,371	11.01	1,507	69.0	5.0	0.32	71.3	74.0	85.2	13.6	11.5
Zn-C	1,351	2.09	307	4.4	51.0	2.77	7.7	8.6	3.1	79.2	56.6
Tail.	26,278	0.29	32	0.9	0.2	0.08	21.0	17.4	11.7	7.2	31.9

Remarks: Crude;Crude ore Pb-C;Pb Concentrate Zn-C;Zn Concentrate
Tail;Tailing

Table 29 Estimated operation performance

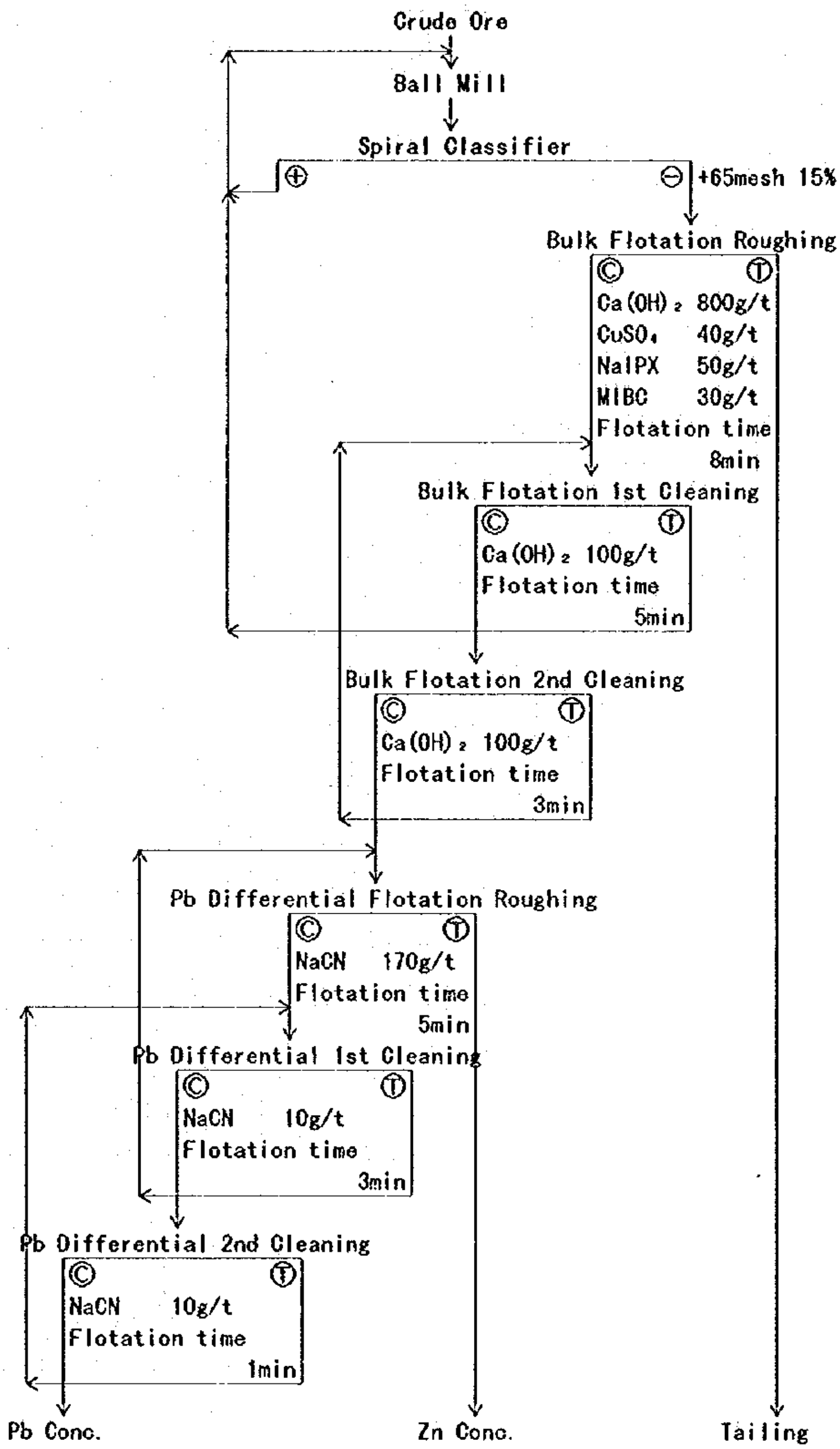


Fig. 39 Optimum Flotation Process Flow sheet

Ⅳ. Mine Development Plan

Part Ⅳ. Mine Development Plan

Chapter 1. Mining Plan

1.1 General

1.1.1 Selection of mining method

The Tsav deposits have the following characteristics:

(1) The deposits consist of high-grade, fine veins with an average width (in terms of the minable crude ore) of 1.03m.
(2) The deposits are poor in continuity.
(3) The deposits dip rather steeply ; the No.4 vein dips 65° to 70°.
(4) The hanging and foot walls are in poor rock conditions as long as it was confirmed at the −60m level.

In mining deposits of this type, the dilution constitutes a serious question. The exploration since 1992 has been carried out by the truckless mining method, and the machinery used for the work still remain at the site and are operable by the Mongolian operators.

Considering the small population of Mongolia (2.2 million), the mechanized cut and fill method combined with the truckless mining is most appropriate for the deposits, even though machinery costs are expensive in relation to labor costs in Mongolia.

1.1.2 Numerical basis for planning

The following base figures are applied for the mining plan:

Minable crude ore to be extracted (t)	332,464
Annual extraction of crude ore (t)	30,000
Number of operating days per year (days)	300
Daily extraction of crude ore (t)	100
Number of shift per day (shifts)	2

1.1.3 Underground structure (Ref. Fig.40 Development Plan of No.4 Vein)

At the No.4 vein, already existing are a vertical shaft from the surface to 630m level(PW-14) which was driven during the ex-Soviet-Mongolian joint exploration, a drift at 630m level, an inclined shaft from the surface to 750m level and a 750m level drift at foot wall, the latter two being truckless and driven by MMAJ since 1992.

A truckless inclined shaft system is considered appropriate for the deposits for the following reasons:

(1) Compared with a vertical shaft, an inclined shaft can more flexibly accommodate changes in orebodies, allowing a wider range of options for planning and its revisions. A truckless inclined shaft can be driven from a free level with a free dip and in a free direction, which represents a great advantage in terms of the time required for reaching an orebody and of the stope preparation.

(2) In case of a vertical shaft, mining cannot be commenced until all the excavation and installation are completed, whereas an inclined shaft permits mining to start as soon as an orebody is reached. At the No.4 vein, the lowest level of the object area of development is 630m, or 220m deep from the surface, while the daily rate of crude ore extraction is as small as 100t. The operation can be done with trucks(for underground use) or load haul dumps(LHDs) on hand, without an additional investment. In contrast, a vertical shaft system requires expensive equipment.

(3) The orebodies being small in size and separately located, excavation has to be carried out at each of them, for which the vertical shaft system is too costly.

It is therefore envisaged that, starting from the No.2 waste dump at the bottom of the inclined shaft driven at the No.4 vein up to 750m level in 1993, a declining shaft (hereafter called "ramp") with a dip of 6° 30′ be driven at the foot wall of the orebody so that the ramp may bore through the existing drift at 630m level. The prospecting, development and stope preparation are to be done from this ramp.

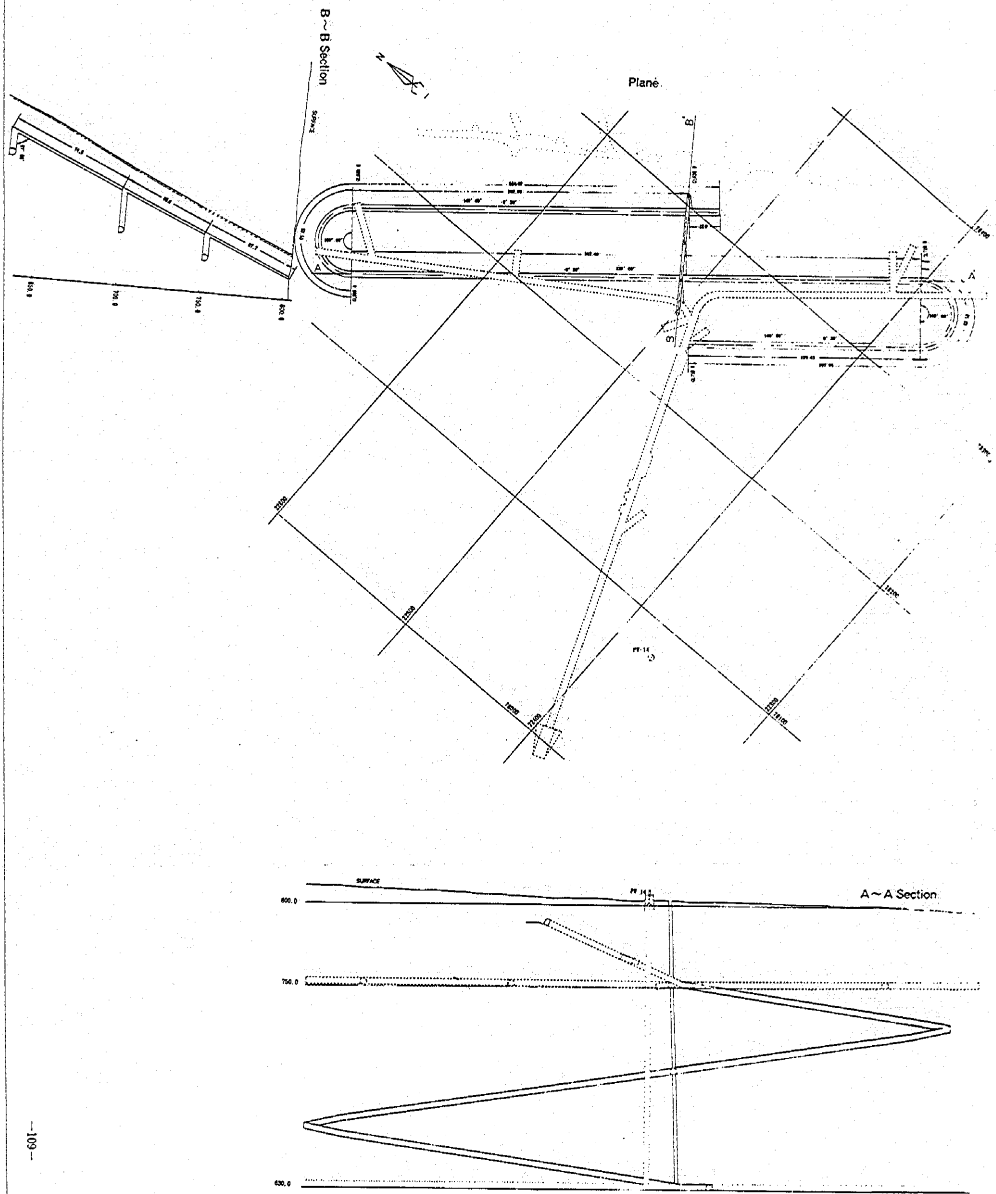


Fig.40 Development Plan of No.4 Vein

Title	Development Plan of No.4Vein		
Scale		Date	
Drawing No.			

1.1.4 Ventilation raise and filling raise(Ref. Fig.40 Development Plan No.4 Vein)

A ventilation shaft will be driven in the central part of the mentioned ramp at the foot wall side, which will be extended downward as the ramp advances. A fan for the shaft sinking will be shifted downward, accordingly. When the shaft reaches 630m level, a ventilation system is established, which comprises the ramp for intake air and the vertical shaft(PW－14) for return air.

After completion of the ventilation system, the ventilation shaft drive as the ramp advances will no longer be used for ventilation but serves as a filling raise. To bore this raise, mechanical boring with a raise borer may be considered as an alternative; in view of the investment effect, however, a stage blasting method(Ref. Fig.41 Stage Blasting Method) is preferable since this is performable with the machinery on hand.

1.2 Operation Plan

1.2.1 Prospecting and Development

Starting from the mentioned ramp, excavation of accesses to the ore deposits, drifting along the veins and prospecting northward(－60m level) are to be conducted in sequence.

1.2.2 Mining methods

(1) In principle, the mechanized cut and fill method will be employed. In practice, however, 1) Ramp in Stoping and 2) Filled Rill Stoping will desirably be applied depending on local conditions. The mechanized cut and fill method in Tsav is shown in Fig.42.

1) Ramp in stoping

Under the mechanized(truckless) cut and fill method, it will become necessary to provide new accesses from the ramp as excavation advances, which increases the quantity of excavation in relation to the quantity of ore. This is a weak point of the method when applied to a deposit like Tsav.

A solution for the problem is the ramp in stoping method, whereby mining is done diagonally in a stope and changeovers of accesses are minimized.(Ref. Fig.43 Ramp in Stoping) In the ramp in stoping method, a mining block is diagonally divided in two, and the lower half is excavated from an access which rises in the stope each time a slice is extracted. When the access reaches the upper drift, extraction of the upper half is carried out from the upper drift.

In case of Tsav, however, the average width of veins of minable ore is as narrow as 1.03m whilst an LHD's bucket is 1.6m in width; thus, the dilution comes into a question. To cope with the question, extraction and blasting are to be conducted at two stages(ore and waste). Whether waste or ore should first be blasted is determined depending on vein width, dip, rock conditions, etc.

Drilling directions may be upward or horizontal. However, rock conditions on the hanging wall is so poor that the full filling is required; consequently, horizontal drilling is opted for. Height of the ceiling and width of stoping at the time of completion of stoping are assumed to be 3.0m and 2.5m, respectively.

2) Filled rill stoping

This mining method is fit to narrow, steep veins like those of the Tsav deposits. In case the hanging and foot walls are of poor rock conditions, however, the method is not appropriate because it increases the dilution.

As shown in Fig.44 Filled Rill Stoping, the lower drift is first driven, from where blasting holes are drilled either upward or downward. After charging explosives, blasting and mucking at a face, the opened space is filled from the upper drift and, at the face of the filled portion, charging, blasting and mucking are again done. This cycle is repeated later on. The interval between drifts are tentatively set at 6m but, for final determination, vein width, dip and rock conditions must be taken into account.

In order to minimize the dilution, the drifting has to be done with two-staged blasting(for separation of waste and ore), whilst drilling and blasting between drifts should be limited to ore portions as long as it is feasible.

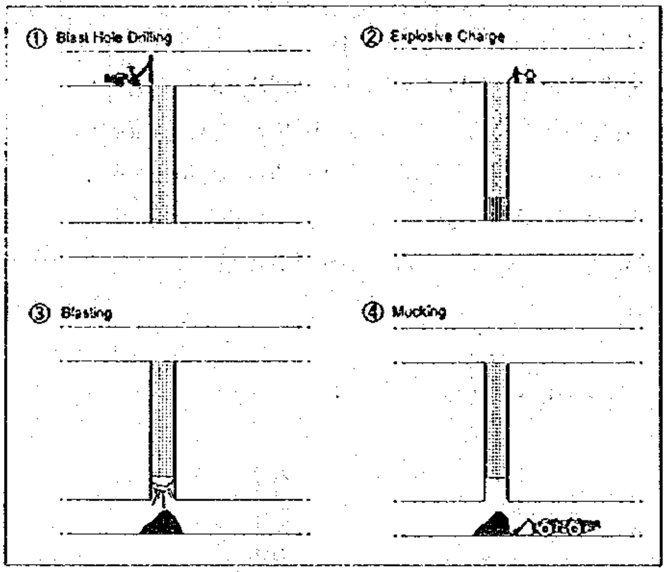


Fig.41 Stage Blasting Method

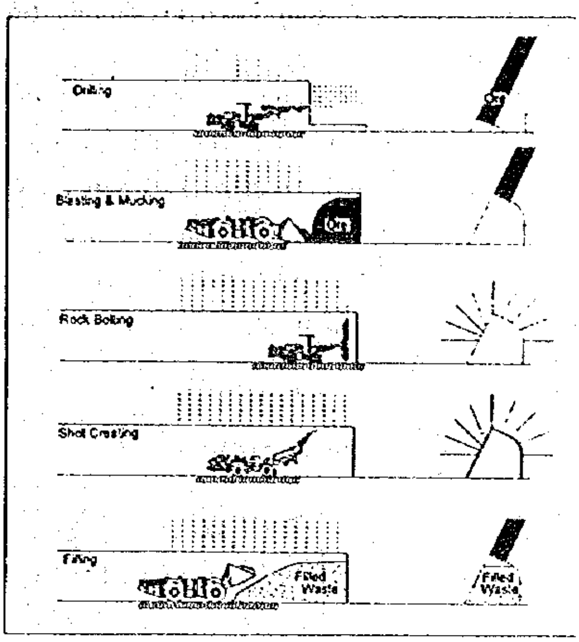


Fig.42 Mechanized Cut & Fill Method

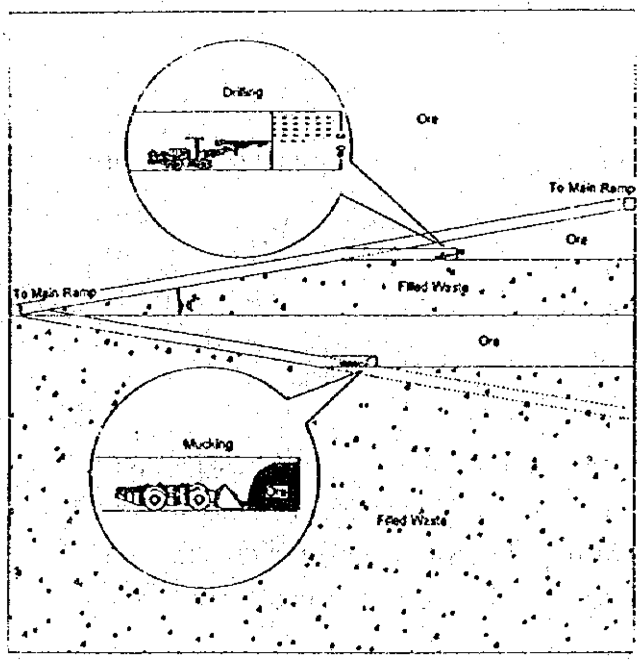


Fig.43 Ramp in Stoping

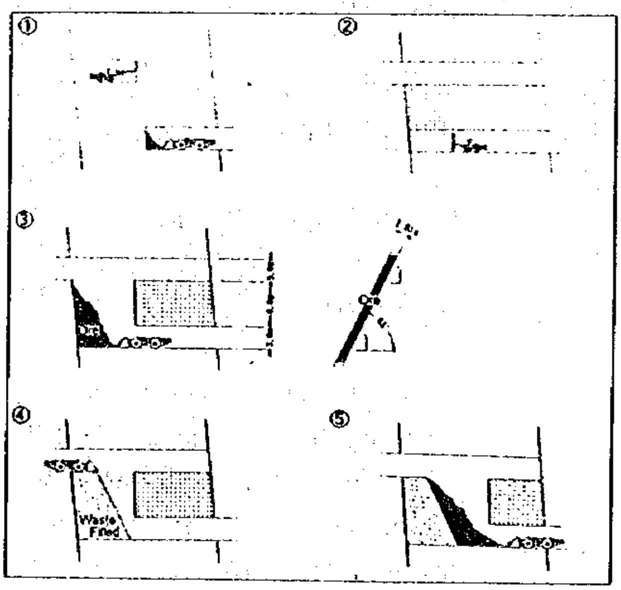


Fig.44 Filled Rill Stoping

(2) Mining Recovery and dilution

The mining recovery is determined by economic viability, vein width and continuity and reliability of ore reserve calculation, whereas vein width and mining method determine the percentage of dilution. The figures below have been empirically worked out on the basis of these considerations. The No.4 vein above 630m level and the other veins up to 100m under the surface are chosen as the mining target. At the No.4 vein, exploration has advanced considerably and the tunnels have been driven up to 750m level.

Vein width		Mining recovery	Dilution
0.7m or more	Proved ore	90%	20%
	Probable ore	80%	20%
	Possible ore	70%	20%
Less than 0.7m	Proved ore	80%	30%
	Probable ore	70%	30%
	Possible ore	60%	30%

(3) Minable crude ore

Quantity of minable crude ore calculated on the assumptions in (2) above are shown in Table. 30-1 and 30-2.

1.2.3 Drilling and blasting

(1) Drilling

For drilling operations for tunneling and drifting, the existing diesel-driven, 2-boom, hydraulic jumbo loaded with a 150kg hydraulic drifter is used. For excavation, the single boom, hydraulic jumbo loaded with a 120kg hydraulic drifter is used because of narrow drifts. Their guide cells are of the telescopic type so that they may drive rockbolts.

(2) Blasting

For blasting, AN-FO, desirable in view of easy handling and safety, is

Ore Reserves & Minable Crude Ore Estimation

ORE RESERVES ESTIMATION

Proved Ore Reserves

Vein	Level	Block	Area m2	Length m	Width m	Hight m	Volume m3	S.G.	Reserve t	Grade Au g/t	Grade Ag g/t	Grade Pb %	Grade Zn %	Grade Cu %	Quantity of metal Au g	Quantity of metal Ag kg	Quantity of metal Pb t	Quantity of metal Zn t	Quantity of metal Cu t
No. 4		1	162.0	99.3	1.6	26.5	4,293.0	3.1	13,308	1.32	92.47	6.44	1.51	0.24	17,571.06	1,230.63	856.57	201.33	31.62
		2	134.0	86.3	1.6	28.5	3,819.0	3.1	11,839	0.63	70.53	5.88	4.24	0.13	7,488.99	835.02	696.11	501.91	14.91
		3	85.5	58.0	1.3	26.5	2,265.8	3.1	7,024	2.91	102.83	8.14	0.44	0.20	20,408.94	722.29	571.50	30.98	13.70
		4	12.0	36.7	0.3	26.0	312.0	3.1	967	3.21	7.67	1.17	1.72	0.06	3,107.94	7.42	11.32	16.67	0.58
		5	70.6	85.5	0.8	20.5	1,435.0	3.1	4,449	4.76	111.83	6.60	0.23	0.11	21,181.22	497.48	293.79	10.17	5.08
		6	72.5	62.5	1.2	27.0	1,957.5	3.1	6,068	2.26	64.38	3.45	0.94	0.19	13,728.47	390.65	209.23	56.91	11.55
		7	9.5	17.6	0.5	29.5	280.3	3.1	869	4.46	135.30	7.09	4.90	0.67	3,875.65	117.54	61.56	42.53	5.83
		total	545.5	455.7	1.2	26.3	14,362.5	3.1	44,524	1.96	85.37	6.06	1.93	0.19	87,362.25	3,801.03	2,700.08	860.50	83.27

RESERVES of MINABLE CRUDE ORE

Proved Ore Reserves

Vein	Level	Block	Mining Recovery %	Dilution %	Crude Ore Reserve t	Grade of Crude Ore Au g/t	Ag g/t	Pb %	Zn %	Cu %	Quantity of metal Au g	Ag kg	Pb t	Zn t	Cu t
No. 4		1	90	20	14,373	1.10	77.06	5.36	1.26	0.20	15,813.95	1,107.57	770.91	181.20	28.46
		2	90	20	12,786	0.53	58.78	4.90	3.53	0.10	6,740.09	751.52	626.50	451.72	13.42
		3	90	20	7,586	2.42	85.70	6.78	0.37	0.16	18,368.05	650.06	514.35	27.88	12.33
		4	80	30	1,006	2.47	5.90	0.90	1.33	0.05	2,486.35	5.94	9.06	13.34	0.46
		5	90	20	4,804	3.97	93.19	5.50	0.19	0.10	19,063.10	447.73	264.41	9.15	4.57
		6	90	20	6,554	1.89	53.65	2.87	0.78	0.16	12,355.62	351.59	188.31	51.22	10.40
		7	80	30	904	3.43	104.07	5.45	3.77	0.52	3,100.52	94.03	49.25	34.02	4.66
		total	90	20	48,012	1.62	70.99	5.05	1.60	0.15	77,927.68	3,408.43	2,422.78	768.53	74.30

Probable Ore Reserves

Vein	Level	Block	Area m2	Length m	Width m	Hight m	Volume m3	S.G.	Reserve t	Grade Au g/t	Grade Ag g/t	Grade Pb %	Grade Zn %	Grade Cu %	Quantity of metal Au g	Quantity of metal Ag kg	Quantity of metal Pb t	Quantity of metal Zn t	Quantity of metal Cu t
No. 4	Trench	1	21	25.5	0.8	25.5	267.8	3.1	830	2.46	109.32	8.30	0.83	0.35	2,041.86	90.74	68.89	6.89	2.91
		2	83	54.5	1.5	28.5	1,182.8	3.1	3,667	2.30	231.40	13.29	2.90	0.55	8,433.01	848.43	487.28	106.33	20.17
		6	25	28.0	0.9	23.0	287.5	3.1	891	4.90	99.77	7.44	0.13	0.15	4,367.13	88.92	66.31	1.16	1.34
		7	11	17.0	0.6	17.0	93.5	3.1	290	3.73	261.86	10.77	0.10	0.24	1,081.14	75.90	31.22	0.29	0.70
		8	2	7.0	0.3	7.0	7.0	3.1	22	0.54	43.56	3.03	0.06	0.03	11.72	0.95	0.66	0.01	0.01
		9	7	10.0	0.7	10.0	35.0	3.1	109	1.34	37.41	1.97	0.13	0.24	145.39	4.06	2.14	0.14	0.26
		10	31	43.0	0.7	23.5	364.3	3.1	1,129	2.96	139.52	13.51	0.17	0.12	3,342.36	156.41	152.55	1.92	1.36
		11	16	23.0	0.7	23.0	184.0	3.1	570	2.02	134.89	11.87	0.23	0.14	1,152.21	76.94	67.71	1.31	0.80
		12	11	29.0	0.4	23.5	129.3	3.1	401	1.47	93.00	5.88	0.17	0.05	588.99	37.26	23.56	0.68	0.20
		13	3	8.5	0.4	8.5	12.8	3.1	40	2.75	147.86	6.08	0.14	0.10	108.69	5.84	2.40	0.06	0.04
		14	9	10.0	0.9	10.0	45.0	3.1	140	2.28	245.43	8.60	0.11	0.07	318.06	34.38	12.00	0.15	0.10
		15	1	8.0	0.5	8.0	16.0	3.1	50	1.02	56.53	2.97	0.12	0.09	50.59	2.80	1.47	0.06	0.04
		16	6	10.0	0.6	10.0	30.0	3.1	93	0.65	124.89	4.05	0.12	0.14	60.45	11.61	3.77	0.11	0.13
		17	5	12.0	0.4	12.0	30.0	3.1	93	4.21	128.44	7.68	0.11	0.11	391.53	11.94	7.14	0.10	0.10
		18	3	8.0	0.4	8.0	12.0	3.1	37	1.04	302.62	29.76	0.03	0.05	38.69	11.26	11.07	0.01	0.02
		total	237	283.5	0.8	22.8	2,696.8	3.1	8,360	2.65	174.34	11.22	1.43	0.34	22,131.81	1,457.45	938.16	119.23	28.16
	750m Upper	3	100	90.0	1.1	23.5	1,175.0	3.1	3,643	1.12	175.08	8.20	5.95	0.28	4,079.60	637.73	298.69	216.73	10.20
		4	26	33.0	0.8	23.5	305.5	3.1	947	1.03	85.15	1.49	1.21	0.14	975.45	80.64	14.11	11.45	1.33
		11	19	20.0	0.5	10.0	95.0	3.1	295	4.40	2.67	0.45	0.26	0.06	1,295.80	0.79	1.33	0.77	0.18
		12	20	50.0	0.4	10.0	100.0	3.1	310	5.61	0.10	0.45	0.10	0.02	1,739.10	0.00	1.40	0.31	0.06
		subtotal	165	193.0	0.9	20.3	1,675.5	3.1	5,194	1.55	138.45	6.07	4.41	0.23	8,089.95	719.16	315.52	229.25	11.76
	750m Lower	5	126	105.0	1.2	30.0	1,890.0	3.1	5,859	1.94	73.68	3.90	1.04	0.21	11,366.46	431.69	228.50	60.93	12.30
		7	100	90.0	1.1	30.0	1,500.0	3.1	4,650	1.12	175.08	8.20	5.95	0.28	5,208.00	814.12	381.30	276.68	13.02
		8	26	33.0	0.8	30.0	390.0	3.1	1,209	1.03	85.15	1.49	1.21	0.14	1,245.27	102.95	18.01	14.63	1.69
		9	125	71.0	1.8	20.0	1,250.0	3.1	3,875	0.83	85.44	7.71	6.79	0.19	3,216.25	331.08	298.76	263.11	7.36
		10	8	21.0	0.4	10.0	40.0	3.1	124	4.30	23.00	3.19	2.91	0.08	533.20	2.85	3.96	3.61	0.10
		13	14	25.0	0.4	10.0	70.0	3.1	217	5.24	177.23	8.46	6.58	0.86	1,137.08	38.46	18.36	14.28	1.87
		subtotal	399	345.0	1.2	25.8	5,140.0	3.1	15,934	1.43	108.02	5.96	3.97	0.23	22,706.26	1,721.15	948.89	633.24	36.34
	750m	total	564	538.0	1.0	24.2	6,815.5	3.1	21,128	1.46	115.50	5.98	4.08	0.23	30,796.22	2,440.31	1,264.41	862.50	48.11
	630m	1	7	15.5	0.5	31.0	108.5	3.1	336	0.84	34.31	3.88	8.23	0.07	283.80	11.54	13.03	27.69	0.23
		2	16	23.0	0.7	46.0	368.0	3.1	1,141	0.53	84.42	3.50	3.83	0.11	603.52	96.31	39.94	43.73	1.31
		3	41	35.5	1.2	60.0	1,230.0	3.1	3,813	1.43	319.16	9.10	4.35	0.43	5,460.53	1,216.95	347.05	165.79	16.35
		4	62	42.0	1.5	60.0	1,860.0	3.1	5,766	0.71	133.58	5.69	6.26	0.39	4,073.04	770.22	328.28	360.78	22.49
		5	11	18.0	0.6	36.0	198.0	3.1	614	0.84	194.52	6.35	6.08	0.69	514.17	119.40	38.98	37.30	4.22
		6	31	41.0	0.8	60.0	930.0	3.1	2,883	0.82	363.26	6.36	3.46	0.31	2,368.34	1,047.27	183.46	99.73	8.83
		7	63	68.0	0.9	60.0	1,890.0	3.1	5,859	3.15	171.52	3.05	3.13	0.14	18,473.89	1,004.96	178.90	183.42	8.20
		8	13	13.0	1.0	26.0	169.0	3.1	524	0.54	14.58	0.46	2.32	0.04	280.50	7.64	2.40	12.17	0.21
		9	67	69.0	1.0	60.0	2,010.0	3.1	6,231	1.95	294.79	6.87	4.57	0.23	12,131.61	1,836.85	428.14	284.80	14.11
		10	36	32.0	1.1	60.0	1,080.0	3.1	3,348	1.23	459.19	11.50	3.45	0.21	4,112.68	1,537.36	384.90	115.49	7.03
		11	39	18.0	2.2	36.0	702.0	3.1	2,176	2.02	118.08	6.83	3.82	0.16	4,386.51	256.96	148.60	83.13	3.46
		12	34	34.0	1.0	60.0	1,020.0	3.1	3,162	0.78	72.20	3.91	2.69	0.15	2,481.93	228.30	123.63	85.06	4.74
		13	16	22.5	0.7	45.0	360.0	3.1	1,116	0.33	27.49	2.87	1.48	0.08	370.90	30.68	32.05	16.52	0.93
		14	8	17.0	0.5	34.0	136.0	3.1	422	0.69	97.43	5.99	7.22	0.49	292.29	41.09	25.26	30.43	2.08
		15	115	93.5	1.2	60.0	3,450.0	3.1	10,695	1.32	211.32	7.58	7.29	0.52	14,139.53	2,260.03	810.99	779.73	55.37
		total	559	542.0	1.0	55.5	15,511.5	3.1	48,086	1.46	217.64	6.42	4.84	0.31	69,973.25	10,465.54	3,085.62	2,325.77	149.51
		total	1,360	1,373.5	1.0	36.8	25,023.8	3.1	77,574	1.58	185.16	6.82	4.26	0.29	122,901.29	14,363.29	5,288.19	3,307.50	225.84
No. 6	Trench	1	8	20.0	0.4	20.0	80.0	3.1	248	1.04	117.44	5.67	0.11	0.04	257.40	29.12	14.05	0.28	0.11
		2	113	105.5	1.1	30.0	1,695.0	3.1	5,255	0.70	177.94	11.44	0.51	0.19	3,678.27	934.99	601.15	26.81	9.97
		3	84	71.5	1.1	30.0	1,260.0	3.1	3,906	0.35	129.63	10.37	1.55	0.04	1,356.82	506.34	405.24	60.50	1.65
		4	20	29.0	0.7	29.0	290.0	3.1	899	0.79	71.08	1.76	0.81	0.19	708.15	63.90	15.79	7.29	1.69
		5	8	14.0	0.6	14.0	56.0	3.1	174	0.12	70.86	4.12	0.50	0.15	20.73	12.30	7.16	0.87	0.26
		6	6	9.5	0.6	9.5	28.5	3.1	88	0.17	63.33	7.97	0.41	0.02	14.62	5.60	7.04	0.37	0.02
		7	13	30.0	0.4	30.0	195.0	3.1	605	0.12	194.24	14.31	0.21	0.07	73.94	117.42	86.49	1.27	0.41
		total	252	282.5	0.9	28.6	3,604.5	3.1	11,174	0.55	149.43	10.17	0.87	0.13	6,109.92	1,669.68	1,136.93	97.40	14.10
No. 8	630m	1	9	13.0	0.7	26.0	117.0	3.1	363	0.07	313.07	1.03	1.38	0.02	26.25	113.55	3.75	5.01	0.06
		2	10	10.5	1.0	21.0	105.0	3.1	326	0.04	216.78	1.13	1.11	0.02	13.26	70.56	3.67	3.60	0.08
		3	13	27.0	0.5	54.0	351.0	3.1	1,088	0.13	292.97	0.26	2.00	0.03	146.43	318.78	2.80	21.78	0.28
		4	103	65.0	1.6	60.0	3,090.0	3.1	9,579	0.07	703.59	1.66	4.41	0.03	654.32	6,739.68	158.83	422.05	2.44
		total	135	115.5	1.2	54.3	3,663.0	3.1	11,355	0.07	637.61	1.49	3.99	0.03	840.27	7,242.57	169.05	452.45	2.86
		grand total	1,747	1,771.5	1.0	37.0	32,291.3	3.1	100,103	1.30	232.52	6.59	3.85	0.24	129,851.48	23,275.54	6,594.17	3,857.35	242.79

Probable Ore Reserves

Vein	Level	Block	Mining Recovery %	Dilution %	Crude Ore Reserve t	Grade of Crude Ore Au g/t	Ag g/t	Pb %	Zn %	Cu %	Quantity of metal Au g	Ag kg	Pb t	Zn t	Cu t
No. 4	Trench	1	80	20	797	2.05	91.10	6.92	0.69	0.29	1,633.49	72.59	55.11	5.51	2.33
		2	80	20	3,520	1.92	192.83	11.07	2.42	0.46	6,746.41	678.74	389.82	85.06	16.14
		6	80	20	856	4.08	83.14	6.20	0.11	0.13	3,493.70	71.14	53.05	0.93	1.07
		7	70	30	264	2.87	201.43	8.29	0.08	0.19	756.80	53.13	21.85	0.20	0.49
		8	70	30	20	0.41	33.68	2.34	0.04	0.04	8.20	0.67	0.46	0.01	0.01
		9	80	20	104	1.12	31.18	1.64	0.11	0.20	116.31	3.25	1.71	0.11	0.21
		10	80	20	1,084	2.47	115.43	11.26	0.14	0.10	2,673.89	125.13	122.04	1.54	1.09
		11	80	20	548	1.68	112.41	9.89	0.19	0.12	921.77	61.55	54.17	1.05	0.64
		12													
		13													
		14													
		15													
		16													
		17													
		18													
		total	71	20	7,132	2.27	148.26	9.71	1.31	0.31	16,350.51	1,066.80	698.22	94.41	21.97
	750m Upper	3	80	20	3,497	0.93	145.88	6.83	4.96	0.23	3,263.68	510.18	238.95	173.38	8.16
		4	80	20	909	0.86	70.96	1.24	1.01	0.12	780.37	64.51	11.29	9.17	1.06
		11	80	20	283	3.66	2.23	0.38	0.22	0.05	1,036.64	0.63	1.06	0.62	0.14
		12	70	30	282	4.32	0.01	0.35	0.08	0.01	1,217.37	0.00	0.98	0.22	0.04
		subtotal	79	21	4,972	1.27	115.72	5.07	3.69	0.19	6,298.06	575.33	252.28	183.39	9.41
	750m Lower	5	80	20	5,625	1.62	61.40	3.25	0.87	0.17	9,093.17	345.35	182.80	48.74	9.84
		7	80	20	4,464	0.93	145.90	6.83	4.96	0.23	4,166.40	651.30	305.04	221.34	10.42
		8	80	20	1,161	0.86	70.96	1.24	1.01	0.12	996.22	82.36	14.41	11.70	1.35
		9	80	20	3,720	0.69	71.20	6.42	5.66	0.16	2,573.00	264.86	239.01	210.49	5.89
		10	70	30	113	3.31	17.68	2.45	2.24	0.06	373.24	2.00	2.77	2.53	0.07
		13	70	30	197	4.03	136.33	6.51	5.06	0.66	795.96	26.92	12.85	10.00	1.31
		subtotal	80	20	15,280	1.18	89.84	4.95	3.30	0.19	17,997.98	1,372.79	756.88	504.80	28.88
	750m	total	80	20	20,251	1.20	96.20	4.98	3.40	0.19	24,296.04	1,948.12	1,009.16	688.19	38.29
	630m	1	40	20	161	0.70	28.62	3.23	6.87	0.06	113.52	4.62	5.21	11.08	0.09
		2	40	20	548	0.44	70.34	2.92	3.19	0.10	241.41	38.52	15.98	17.49	0.52
		3	40	20	1,830	1.19	265.97	7.58	3.62	0.36	2,184.21	486.78	138.82	66.32	6.54
		4	40	20	2,768	0.59	111.32	4.74	5.21	0.33	1,629.22	308.09	131.31	144.31	9.00
		5	35	30	279	0.64	149.59	4.88	4.67	0.53	179.96	41.79	13.64	13.06	1.48
		6	40	20	1,384	0.68	302.71	5.30	2.88	0.26	947.34	418.91	73.38	39.89	3.53
		7	40	20	2,812	2.63	142.94	2.54	2.61	0.12	7,389.56	401.98	71.56	73.37	3.28
		8	40	20	252	0.45	12.15	0.38	1.94	0.03	112.20	3.06	0.96	4.87	0.08
		9	40	20	2,991	1.62	245.66	5.73	3.81	0.19	4,852.64	734.74	171.26	113.92	5.64
		10	40	20	1,607	1.02	382.66	9.58	2.87	0.17	1,645.07	614.94	153.96	46.20	2.81
		11	40	20	1,044	1.68	98.41	5.69	3.18	0.13	1,754.60	102.78	59.44	33.25	1.38
		12	40	20	1,518	0.65	60.17	3.26	2.24	0.12	992.77	91.92	49.45	34.02	1.90
		13	40	20	536	0.28	22.91	2.39	1.23	0.07	148.36	12.27	12.82	6.61	0.37
		14	35	30	192	0.53	74.88	4.60	5.55	0.38	102.30	14.38	8.84	10.65	0.73
		15	40	20	5,134	1.10	176.10	6.32	6.08	0.43	5,655.81	904.01	324.40	311.89	22.15
		total	40	20	23,055	1.21	181.22	5.34	4.02	0.26	27,948.97	4,178.20	1,231.03	926.92	59.51
		total	54	20	50,498	1.36	142.43	5.82	3.39	0.24	68,595.58	7,192.51	2,938.42	1,709.52	119.76
No. 6	Trench	1													
		2	80	20	5,045	0.58	148.27	9.53	0.43	0.16	2,942.62	747.99	480.92	21.45	7.98
		3	60	20	3,750	0.29	108.03	8.65	1.29	0.04	1,085.46	405.07	324.19	48.40	1.32
		4	80	20	863	0.66	59.23	1.46	0.68	0.16	566.52	51.12	12.63	5.83	1.35
		5													
		6													
		7													
		total	72	20	9,658	0.48	124.69	8.47	0.78	0.11	4,594.59	1,204.18	817.74	75.68	10.65
No. 8	630m	1													
		2													
		3													
		4													
		total													
		grand total	50	20	60,156	1.22	139.58	6.24	2.97	0.22	73,190.17	8,396.69	3,756.16	1,785.20	130.41

Table 30-1 Ore Reserves & Minable Crude Ore Estimation

Possible Ore Reserves

Vein	Block	No.	Area m2	Width	Safty factor %	Volume m3	S.G.	Reserve t	Grade Au g/t	Grade Ag g/t	Grade Pb %	Grade Zn %	Grade Cu %
No.1	No.1	1	1,600	0.14	45	224.0	3.1	312		64	6.30	0.02	
		2	1,600	0.44	45	704.0	3.1	982		105	6.88	1.41	
		3	800	1.00	45	800.0	3.1	1,116		85	4.63	0.06	
		4	800	2.20	45	1,760.0	3.1	2,455		54	7.46	1.74	
		5	57,880	0.58	45	33,570.4	3.1	46,831		99	2.63	10.19	
		6	54,970	0.35	45	19,239.5	3.1	26,839		421	9.90	6.95	
		7	1,600	0.19	45	304.0	3.1	424		88	3.55	3.60	
		8	800	0.45	45	360.0	3.1	502		343	16.00	1.20	
		9	2,410	0.32	45	771.2	3.1	1,076		71	1.70	3.48	
	total		122,460	0.47	45	57,733.1	3.1	80,538		206	5.37	8.39	
	No.1A	1	11,840	0.55	45	6,512.0	3.1	9,084		75	1.53	2.14	
		2	800	0.50	45	400.0	3.1	558		390	11.86	0.58	
		3	162,870	0.39	45	63,519.3	3.1	88,609		146	5.02	6.63	
	total		175,510	0.40	45	70,431.3	3.1	98,252		141	4.74	6.18	
	No.1B	1	26,400	0.20	45	5,280.0	3.1	7,366		137	7.19	3.17	
		2	1,250	0.85	45	1,062.5	3.1	1,482		360	16.22	0.11	
		3	8,200	0.44	45	3,608.0	3.1	5,033		87	1.62	3.28	
		4	1,600	0.16	45	256.0	3.1	357		152	4.26	1.40	
		5	1,050	1.20	45	1,260.0	3.1	1,758		59	3.35	2.83	
		6	6,560	0.58	45	3,804.8	3.1	5,308		41	3.80	1.68	
	total		45,060	0.34	45	15,271.3	3.1	21,303		111	5.29	2.55	
No.1 Vein Total			343,030	0.42	45	143,435.7	3.1	200,093		164	5.05	6.68	
No.2	No.2	1	121,680	0.88	45	107,078.4	3.1	149,374		274	11.06	2.90	
		2	1,600	0.30	45	480.0	3.1	670		33	0.91	3.37	
		3	1,600	0.68	45	1,088.0	3.1	1,518		32	1.16	1.87	
		4	1,860	0.65	45	1,209.0	3.1	1,687		160	5.21	0.30	
		5	57,380	1.09	45	62,544.2	3.1	87,249		267	5.18	1.53	
		6	800	1.20	45	960.0	3.1	1,339		41	5.86	0.02	
	total		184,920	0.94	45	173,359.6	3.1	241,837		267	8.78	2.37	
	No.2A	1	800	1.60	45	1,280.0	3.1	1,786		1,246	15.52	0.66	
		2	13,290	0.41	45	5,448.9	3.1	7,601		94	2.90	4.19	
		3	1,600	0.41	45	656.0	3.1	915		144	8.59	1.70	
		4	13,480	1.02	45	13,749.6	3.1	19,181		39	1.61	5.07	
		5	800	1.20	45	960.0	3.1	1,339		170	4.65	0.19	
	total		29,970	0.74	45	22,094.5	3.1	30,822		131	3.07	4.29	
	No.2B	1	26,700	0.57	45	15,219.0	3.1	21,231		108	5.67	0.86	
		2	800	0.23	45	184.0	3.1	257		44	2.14	3.60	
	total		27,500	0.56	45	15,403.0	3.1	21,487		107	5.63	0.89	
	No.2HW	1	23,660	0.92	45	21,767.2	3.1	30,365		129	6.99	4.66	
	total		23,660	0.92	45	21,767.2	3.1	30,365		129	6.99	4.66	
No.2 Vein Total			266,050	0.87	45	232,624.3	3.1	324,511		231	7.86	2.67	
No.4	630m upper	1	16,180	0.87	65	14,076.6	3.1	28,364		196	5.83	6.80	
		2	7,750	0.84	65	6,510.0	3.1	13,118		89	4.37	1.78	
		3	3,925	1.12	65	4,396.0	3.1	8,858		448	14.17	4.46	
		4	4,650	0.63	65	2,929.5	3.1	5,903		232	14.38	9.94	
		5	4,910	0.94	65	4,615.4	3.1	9,300		205	13.40	7.49	
		6	4,480	0.31	65	1,388.8	3.1	2,798		55	6.86	0.77	
		7	13,690	0.42	65	5,749.8	3.1	11,586		66	5.42	4.97	
	subtotal		55,585	0.71	65	39,666.1	3.1	79,927		186	8.00	5.55	
	630m lower	8	12,950	0.72	45	9,324.0	3.1	13,007		311	7.55	8.07	
		9	8,030	1.24	45	9,957.2	3.1	13,890		374	5.34	8.39	
		11	30,440	1.80	45	54,792.0	3.1	76,435		91	4.95	3.67	
		12	5,250	0.42	45	2,205.0	3.1	3,076		66	5.42	4.97	
	subtotal		56,670	1.35	45	76,278.2	3.1	106,408		154	5.33	4.86	
	north	13	14,730	0.18	45	2,651.4	3.1	3,699		434	7.45	3.09	
		14	61,030	0.61	45	37,228.3	3.1	51,933		51	4.31	1.87	
		15	8,600	0.34	45	2,924.0	3.1	40,079		51	6.21	1.34	
	subtotal		84,360	0.51	45	42,803.7	3.1	59,711		75	4.63	1.91	
	total		196,615	0.81	50	158,748.0	3.1	246,046		145	6.03	4.37	
	No.4A	1	222,460	0.41	45	91,208.6	3.1	127,236		327	9.57	4.09	
		2	23,050	0.29	45	6,684.5	3.1	9,325		103	7.24	3.24	
		3	19,940	0.46	45	9,172.4	3.1	12,795		126	6.72	3.26	
		4	400	0.34	45	136.0	3.1	190		52	3.16	0.82	
	total		265,850	0.40	45	107,201.5	3.1	149,546		295	9.17	3.96	
No.4 vein total			462,465	0.58	48	265,949.5	3.1	395,593		202	7.22	4.22	
No.6	No.6	1	44,800	0.83	45	37,184.0	3.1	51,872		510	6.70	3.47	
		2	63,850	0.56	45	35,761.6	3.1	49,897		67	4.26	2.96	
		3	1,600	1.73	45	2,768.0	3.1	3,861		69	2.37	1.54	
		4	68,640	1.28	45	87,859.2	3.1	122,564		140	9.68	4.68	
No.6 vein Total			178,900	0.91	45	163,572.8	3.1	228,184		207	7.69	3.98	
No.8	No.8	1	51,840	0.40	45	20,736.0	3.1	28,927		353	9.81	2.31	
		2	15,120	1.00	45	15,120.0	3.1	21,092		884	7.18	4.99	
		3	14,130	1.52	65	21,477.6	3.1	43,277		739	1.68	4.60	
		4	1,600	0.53	45	848.0	3.1	1,183		367	17.50	7.72	
		5	1,600	1.23	45	1,968.0	3.1	2,745		58	0.82	3.66	
		6	3,190	0.45	65	1,435.5	3.1	2,893		2,653	13.10	4.73	
		7	48,320	1.42	45	68,614.4	3.1	95,717		263	4.57	2.99	
		8	2,190	1.25	65	2,737.5	3.1	5,516		1,085	7.38	6.94	
		9	13,610	0.72	45	9,799.2	3.1	13,670		337	0.95	3.91	
	subtotal		151,600	0.94	48	142,736.2	3.1	215,020		488	4.93	3.64	
	No.8A	1	3,530	0.15	45	529.5	3.1	739		63	3.20	1.69	
		2	6,170	0.14	45	863.8	3.1	1,205		75	4.50	6.13	
		3	1,600	0.63	45	1,008.0	3.1	1,406		172	16.00	6.80	
		4	1,600	0.59	45	944.0	3.1	1,317		59	4.26	2.69	
		5	1,690	0.32	45	540.8	3.1	754		254	7.51	1.20	
		6	2,880	0.34	65	979.2	3.1	1,973		760	1.09	3.88	
		7	1,600	0.39	45	624.0	3.1	870		418	1.36	2.22	
		8	6,390	0.66	45	4,217.4	3.1	5,883		538	8.86	1.50	
		9	1,600	0.25	45	400.0	3.1	558		309	3.59	4.11	
	subtotal		27,060	0.37	47	10,105.7	3.1	14,706		398	6.73	2.94	
	No.8FW	1	26,440	0.35	45	9,254.0	3.1	12,909		1,708	18.51	6.64	
	subtotal		26,440	0.35	45	9,254.0	3.1	12,909		1,708	18.51	6.64	
No.8 vein total			205,100	0.79	48	162,096.3	3.1	242,636		547	5.76	3.75	
No.10		1	15,553	0.23	45	3,578.5	3.1	4,983		904	2.63	3.18	
		2	7,169	0.40	45	2,864.0	3.1	3,995		1,808	7.54	2.73	
No.10 vein total			22,710	0.28	45	6,442.5	3.1	8,984		1,306	4.81	2.98	
grand total			1,478,255	0.66	46	974,119.7	3.1	1,400,000		271	6.87	4.08	
GRAND TOTAL			1,490,548	0.69	49	1,020,773.5	3.1	1,544,627		263	6.84	4.01	

Vein	Block	No.	Quantity of metal Au g	Ag kg	Pb t	Zn t	Cu t	Mining Recovery	Dilution %	Crude Ore Reserve t	Grade of Crude Ore Au g/t	Ag g/t	Pb %	Zn %	Cu %	Quantity of metal Au g	Ag kg	Pb t	Zn t	Cu t
No.1	No.1	1		20.00	19.69	0.06														
		2		103.12	67.57	13.85														
		3		94.86	51.67	0.67														
		4		132.57	183.16	42.72														
		5		4,636.24	1,231.65	4,772.05														
		6		11,299.26	2,657.07	1,865.32														
		7		37.32	15.05	15.27														
		8		172.25	80.35	6.03														
		9		76.38	18.29	37.44														
	total			16,572.02	4,324.50	6,753.40														
	No.1A	1		681.32	138.09	194.40														
		2		217.62	66.18	3.24														
		3		12,936.98	4,448.19	5,874.80														
	total			13,835.91	4,652.45	6,072.44														
	No.1B	1		1,009.09	529.59	233.49														
		2		533.59	240.41	1.63														
		3		437.88	81.54	165.09														
		4		54.28	15.21	5.00														
		5		103.70	58.89	49.74														
		6		217.62	201.69	89.17														
	total			2,356.16	1,127.32	544.18														
No.1 Vein Total				32,764.09	10,104.27	13,369.96														
No.2	No.2	1		40,928.58	16,520.81	4,331.86		30	20	54,446		228.33	9.22	2.42			12,431.96	5,019.16	1,315.79	
		2		22.10	6.09	22.57														
		3		48.57	17.61	28.38														
		4		269.85	87.87	5.06		60	30	1,316		123.05	4.01	0.23			161.91	52.77	3.04	
		5		23,295.53	4,519.51	1,334.91		16	20	16,349		222.50	4.32	1.28			3,637.64	705.73	208.45	
		6		54.91	78.48	0.27		60	30	1,044		31.54	4.51	0.02			32.93	47.09	0.16	
	total			64,619.53	21,230.36	5,723.04		25	20	73,156		222.33	7.96	2.09			16,264.45	5,823.70	1,527.44	
	No.2A	1		2,224.85	277.13	11.78		70	20	1,500		1,038.10	12.93	0.55			1,557.49	193.99	8.25	
		2		714.51	220.44	318.49		47	30	4,640		72.31	2.23	3.22			335.48	103.50	149.54	
		3		131.78	78.61	15.56		48	30	571		110.79	6.61	1.31			63.25	37.73	7.47	
		4		748.05	308.81	972.46		40	20	9,180		32.50	1.34	4.22			298.33	123.16	387.83	
		5		227.66	62.27	2.54		70	20	1,125		141.63	3.88	0.16			159.36	43.59	1.78	
	total			4,046.85	947.25	1,320.81		45	23	17,015		141.86	2.95	3.26			2,413.83	501.97	554.86	
	No.2B	1		2,292.89	1,203.77	183.58		19	30	5,247		83.07	4.36	0.66			435.91	228.85	34.71	
		2		11.29	5.49	9.24		60	30	200		33.79	1.64	2.77			6.77	3.29	5.54	
	total			2,304.19	1,209.26	191.82		20	30	5,448		81.26	4.26	0.74			442.68	232.15	40.25	
	No.2HW	1		3,917.12	2,122.53	1,415.02		27	20	9,653		107.50	5.83	3.88			1,038.38	562.66	375.11	
	total			3,917.12	2,122.53	1,415.02		27	20	9,653		107.50	5.83	3.88			1,038.38	562.66	375.11	
No.2 Vein Total				74,887.69	25,509.40	8,650.73		27	21	105,278		191.49	6.76	2.37			20,159.35	7,120.47	2,497.66	
No.4	630m upper	1		5,559.41	1,653.64	1,928.78		70	20	23,826		163.34	4.86	5.67			3,891.59	1,157.55	1,350.15	
		2		1,167.47	573.24	233.49		70	20	11,019		74.16	3.64	1.48			817.23	401.27	163.44	
		3		3,968.36	1,255.17	395.06		70	20	7,441		373.33	11.81	3.72			2,777.85	878.62	276.54	
		4		1,369.49	848.84	586.75		60	30	4,604		178.45	11.06	7.65			821.69	509.30	352.05	
		5		1,906.51	1,246.20	696.57		70	20	7,812		170.83	11.17	6.24			1,334.56	872.34	487.60	
		6		153.91	191.97	21.55														
		7		764.67	627.95	575.82														
	subtotal			14,889.81	6,397.03	4,438.02		51	21	54,702		176.28	6.98	4.81			9,642.91	3,819.08	2,629.78	
	630m lower	8		4,045.17	982.03	1,049.68														
		9		5,194.97	741.74	1,165.49														
		11		6,955.57	3,783.52	2,805.16														
		12		203.01	166.72	152.88														
	subtotal			16,398.73	5,674.01	5,173.03														
	north	13		1,605.24	275.55	114.29														
		14		2,648.61	2,238.33	971.15														
		15		208.03	253.30	54.66														
	subtotal			4,461.87	2,767.18	1,140.10														
	total			35,750.41	14,838.23	10,751.22		18	21	54,702		176.28	6.98	4.81			9,642.91	3,819.08	2,629.78	
	No.4A	1		41,606.17	12,176.49	5,203.95														
		2		960.46	675.12	302.13														
		3		1,612.23	859.86	417.13														
		4		9.87	6.00	1.56														
	total			44,188.73	13,717.46	5,924.77														
No.4 vein total				79,939.14	28,555.69	16,675.99		11	21	54,702		176.28	6.98	4.81			9,642.91	3,819.08	2,629.78	
No.6	No.6	1		26,454.56	3,475.40	1,799.95		29	20	17,896		425.00	5.58	2.89			7,605.63	999.18	517.49	
		2		3,342.46	2,125.20	1,476.67		22	30	14,502		51.54	3.28	2.28			747.42	475.22	330.20	
		3		266.43	91.51	59.46														
		4		17,158.90	11,864.15	5,735.98		22	20	31,918		116.67	8.07	3.90			3,723.75	2,574.71	1,244.60	
No.6 vein Total				47,222.35	17,556.28	9,072.06		23	22	64,316		187.77	6.30	3.25			12,076.87	4,049.12	2,092.49	
No.8	No.8	1		10,211.13	2,837.31	668.21														
		2		18,645.68	1,514.43	1,052.51														
		3		31,981.97	727.06	1,990.76														
		4		434.15	207.02	91.32														
		5		159.23	22.51	100.48														
		6		7,673.89	378.92	136.82														
		7		25,173.59	4,374.27	2,861.94														
		8		5,984.93	407.09	382.81														
		9		4,606.75	129.86	534.49														
	subtotal			104,871.32	10,598.88	7,819.35														
	No.8A	1		46.54	23.64	12.48														
		2		90.38	54.23	73.87														
		3		241.86	224.99	95.62														
		4		76.38	56.10	34.63														
		5		191.62	56.65	9.05														
		6		1,499.55	21.51	76.56														
		7		363.86	11.84	19.32														
		8		3,165.20	521.26	88.25														
		9		172.42	20.03	22.93														
	subtotal			5,847.80	990.24	432.72														
	No.8FW	1		22,049.14	2,389.52	857.18														
	subtotal			22,049.14	2,389.52	857.18														
No.8 vein total				132,768.26	13,978.63	9,109.21														
No.10		1		4,510.25	131.22	158.65														
		2		7,223.47	301.24	109.07														
No.10 vein total				11,733.72	432.45	267.73														
grand total				379,315.25	96,136.73	57,145.71		13	21	224,295		186.71	6.68	3.22			41,879.13	14,988.67	7,219.93	
GRAND TOTAL				406,391.82	105,430.98	61,863.56		18	21	332,464		161.17	6.37	2.94			53,584.26	21,167.81	9,773.66	

Table 30-2 Ore Reserves & Minable Crude Ore Estimation

unavailable in Mongolia; instead, "Ammonita," a kind of ammonia explosive currently used in Mongolia, will be used. For ignition, detonators are used.

1.2.4 Haulage

Waste and ore extracted at faces and drifts are transported by the existing load haul dumps(LHDs) with $3.8m^3$ and $2.0m^3$ cap. buckets. Ores conveyed from faces are transported by the LHD with a $3.8m^3$ cap. bucket to the surface via the inclined shaft. In case haulage distance exceeds 500m, ores are temporarily placed at the waste dumps driven in the inclined shaft during the exploration and, in turn, transported to the surface, thereby avoiding excessive load on the LHD engines. Compared to another alternative of utilizing 15 or 20−ton underground dump trucks, haulage by LHDs is more advantageous in respect of the investment effect.

Waste coming out of faces and drifts is stored at unused tunnels to the maximum possible extent for future use as filler, and only excess waste is hauled to the surface.

(2) Personnel, Machinery and supplies

For underground transportation of personnel, machinery and supplies, the existing light truck is used. For heavy machinery, the LHDs may be used.

1.2.5 Filling and timbering

(1) Filling

Wastes built up on the surface during the development phase and extracted during the operation phase are utilized as filling material.

Waste on the surface is cast into the vertical ventilation shaft sunk to the central part of the foot wall of No.4 vein when driving the ramp. The cast waste is extracted by an LHD at a level needing waste and transported via the ramp to fill stopes.

(2) Timbering

In principle, timbering is done by rock bolts alone. When rocks are self-supported, concrete spray is applied in thickness of 3 to 5cm to cracks even though such cracks are abundant. In case rocks are not self-supported(faults, etc.), steel supports such as H-type steel frames are put into use.

The spray concrete is prepared at a batch plant(0.5m^3/batch) to be installed either on surface or underground, transported to stopes by a 1.8m^3 truck mixer and sprayed to wall rocks with a spraying machine. Considering temperature control of sand and water, the batch plant will preferably be built underground.

1.3 Equipment Plan

1.3.1 Compressed air

The concrete spray machine (and AN-FO chargers in case AN-FO is available) being the only machinery using compressed air, the existing portable compressor(discharge 21m^3/min) can cater for the requirement. Another alternative is to mount a baby compressor on the spraying machine(or the AN-FO charger).

1.3.2 Ventilation

(1) Ventilation system

Since truckless mining method is applied, the ventilation system has to be capable of treating exhaust gas of diesel engines and noxious gases including blasting fume and dust particles.

During the ramp sinking phase, a ventilation system comprising fans fixed at the chute under the foot wall of ore deposits, which serves for intake air, air ducts through which air is sent to stopes and the ramp itself for return air.

At the operating phase(after completion of the ramp), the ramp will serve for intake air, whilst return air is exhausted through the vertical shaft(PW-14) by a fan to be installed at the shaft mouth. Ventilation at stopes is done by local fans and air ducts. When the drifts along veins are completed, a chute will be sunk from the upper drift near to the face(by the stage blasting method), thereby securing the ventilation and improving the work environment without using local fans.

(2) Fans

One of the existing equipment uses two fans(gas volume 1,000m ^{3}/min, wind pressure 150mmAq, 37.5kW) in series and the other uses two fans(gas volume 500m ^{3}/min, wind pressure 100mmAq, 18.5kW) in series.

If underground airflow requirement is assumed to be 3m ^{3}/min per 1kW of engine power, and in case two LHDs(with diesel engines 185ps and 87ps, respectively) are in use, the required gas volume comes to 600m ^{3} per minute.

$$(185+87)\text{ps} \times 0.7355\text{kW/ps} \times 3\text{m}^3/\text{kW} = 600\text{m}^3/\text{min}$$

Therefore, a fan of gas volume of 1,000m ^{3}/min or two fans of 500m ^{3}/min installed in parallel will be sufficient. Local ventilation will be done by a local fan of 500m ^{3}/min.

1.3.3 Water supply and drainage

(1) Water supply

Water requirement for drilling is 65 ℓ/min per rock drill. On the assumption that three rock drills are used, the total requirement comes to 195 ℓ/min, which can be met by estimated underground drainage of 200 ℓ/min. Underground drainage is distributed to each stope via 2" pipes laid through the ramp.

(2) Underground drainage

While the ramp is being driven, drainage is done by a pump station to be installed in the ramp near the ventilation chute and by 4" pipes laid in the chute. After completion of the ramp, a pump station will be installed at 630m level, and underground water will be drained through the pipes laid in the vertical shaft(PW-14). Drained water is sent to a thickener at the mineral dressing plant and utilized for mineral dressing.

1.3.4 Ancillary facilities

(1) Explosives storage and handling station

The existing magazine and handling station are utilized.

(2) Oil and lubricant storage

For gas oil, the tank(25,000 ℓ cap.) buried near the shaft mouth will be utilized, while a lubricant storage will be newly built in the vicinity of the workshop.

(3) Office and accommodations

An office building and staff accommodations will be built at the site of ex-lodging house for the Japanese team. For workers' accommodations, the existing facilities for the Mongolian personnel will be utilized.

1.4 Main Machinery and Equipment

Type	Quantity	Specifications
Drill Jumbo(Drift)	1	Hydraulic 2 Boom, 150Kg, Telscopic boom Type
Drill Jumbo(Mining)	1	Hydraulic 1 Boom, 120Kg, Telscopic boom Type
Concrete spray Equ.	1	$10m^3$/Hrs class
Mixer Truck	1	$1.8m^3$ class
Mortar Charger	1	1t truck, Squeeze type charger
Concrete Batch Plant	1	$0.5m^3$/batch
Load Haul Dump	2	$3.8m^3$ class
Load Haul Dump	1	$2.0m^3$ class
Compressor	1	$21m^3$/min
Truck	1	2t attached with crane
Truck	2	Underground Service(For Man, Materials)
Wagon Car	2	Underground Patrol
Turbine Pump	2	$1.0m^3$/min, 50Kw
Water Pump	3	5.2Kw
Water Pump	2	2.2Kw
Ventilation Fan	1	1,000mmϕ, 300mmAq, 75Kw
Ventilation Fan	1	900mmϕ, 100mmAq, 18.5Kw
Boring Machine	1	150m class(For Underground)
Boring Machine	1	565m class(For Surface)

Chapter 2. Mineral Dressing Plan

2.1 Mineral Dressing Plant

2.1.1 General

The mineral dressing plant is assumed to treat crude ore at a rate of 100t/d and operate for 300 days a year; therefore, annual treatment of crude ore comes to 30,000t.

For mineral dressing, the bulk differential flotation process is employed to produce Pb and Zn concentrates.

To receive crude ore, a 600t stockyard will be provided near a crushing section, thereby avoiding troubles with ore quantities which may take place when receiving ore, due to the difference between the mining and mineral dressing divisions in respect to numbers of operating hours and days.

Crushing operation is carried out in two shifts a day while grinding, flotation and dewatering of concentrates are in three shifts.

As regards location of the mineral dressing plant, a site 500m up from the shaft mouth is selected, considering the locations of the shaft mouth and the ore deposits, topography, ground condition and location of the tailing dam. As the plant site is an almost level ground with slight inclination, the plant will be a flat type.

2.1.2. Design criteria

The mineral dressing tests have revealed that zinc ore from Tsav deposits is partially accompanied by oxidized copper ore. Therefore, zinc is apt to be activated and is hard to depress when separating lead from zinc. To depress zinc, sodium cyanide consumption increases. Lead and zinc contained in Tsav ore are easily liberated by the conventional flotation process, without any special treatment. However, a problem lies in the separation of lead and zinc because zinc tends to float. Consequently, the bulk flotation process is more favorable than the straight flotation process, which is the reason for the bulk differential flotation process being selected.

With a view to minimizing the capital cost and the operating cost, and also to make the mineral dressing plant operable, the plant is designed on the following principles:

(1) A simple process is to be pursued.

(2) Instrumentation is to remain at a conventional level; high-tech instrumentation involving automation will not be considered.

(3) A washing plant is installed since there are argillized portions near the veins.

(4) All the plant equipment are to be installed indoors. For equipment layout, due consideration is to be paid to maintenance and control.

(5) For selection of machinery and equipment, due consideration is to be paid to their operability and maintainability.

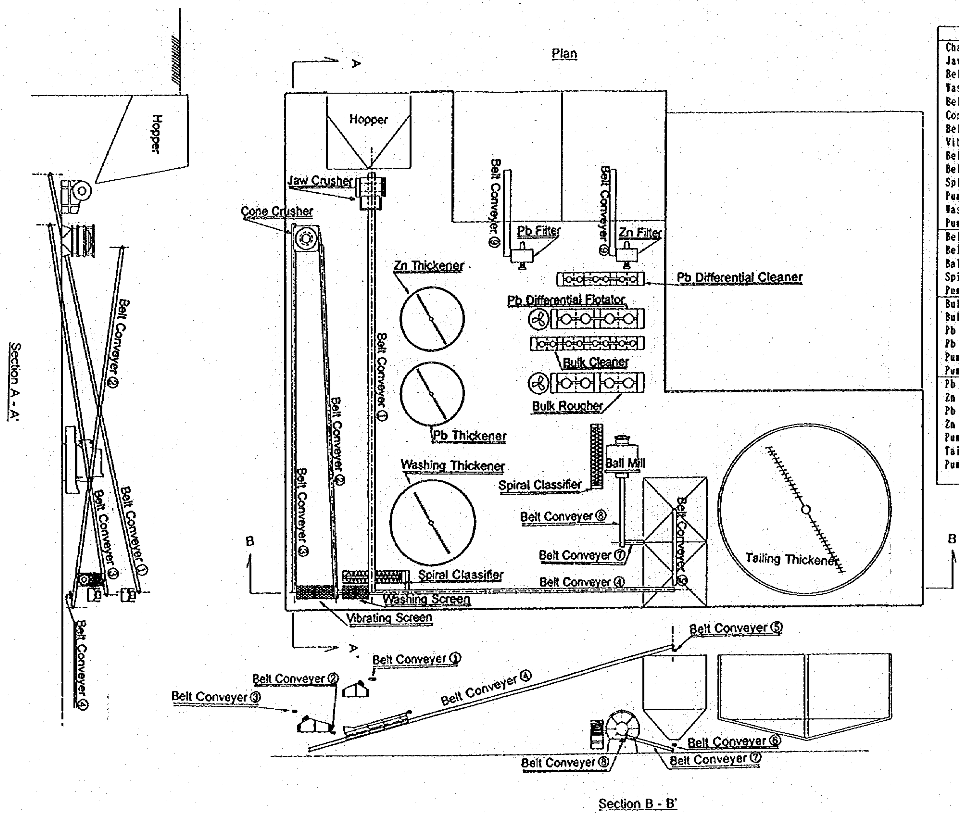
The mineral dressing plant layout is exhibited in Fig.45. The design basis follows:

(1) Operating conditions

Operating Days per year		300
Amount of treatment ore per day (t)		(Average)100
Shifts per day	Crushing	2
	Milling ~ Concentrate	3
	Tailing treatment	3
Working Hours per shift		8

(2) Mill feed

Crude ore's average grade	Pb(%)	6.4
	Zn(%)	2.9
Absolute Specific gravity		3.1
Apparent Specific gravity		1.9
Moisture(%)		5.0
Grinding work index Wi(kWh/t)		11.65



	Pcs	
Chain Feeder	1	500mm^W, 1.25kW
Jaw Crusher	1	510mm^W x 230mm^{Opening}, 19kW
Belt Conveyer①	1	400mm^W x 20m^L, 3.75kW
Washing Screen	1	600mm^W x 1800mm^L, 3.7kW
Belt Conveyer②	1	300mm^W x 17m^L, 3.75kW
Cone Crusher	1	D 600mmϕ, 30kW
Belt Conveyer③	1	300mm^W x 18m,^{L} 3.75kW
Vibrating Screen	1	600mm^W x 1800mm^L, 3.7kW
Belt Conveyer④	1	300mm^W x 18m^L, 3.75kW
Belt Conveyer⑤	1	300mm^W x 4m^L, 3.75kW
Spiral Classifier	1	D 500mmϕ x 4000mm^L, Single-Line, 1.5kW
Pump	1	0.15m^3/min, 2.2kW
Washing Thickener	1	D 4mϕ, 2.2kW
Pump	2	0.15m^3/min, 7.7kW
Belt Conveyer⑧	1	300mmW x 3m^L, 1.5kW
Belt Conveyer⑨	2	300mmW x 4m^L, 3kW
Ball Mill	1	D 1500mmϕ x 1200mm^L, 37kW
Spiral Classifier	1	D 600mmϕ x 4000mm^L, Double-Line, 2.5kW
Pump	1	0.2m^3/min, 2.2kW
Bulk Rougher	4	0.6m^3, 15kW
Bulk Cleaner	8	0.3m^3, 17.6kW
Pb Differential Flotator	4	0.6m^3, 15kW
Pb Differential Cleaner	6	0.3m^3, 13.2kW
Pump	1	0.3m^3/min, 11kW
Pump	5	0.2m^3/min, 11kW
Pb Thickener	1	D 3mϕ, 2.2kW
Zn Thickener	1	D 3mϕ, 2.2kW
Pb Filter	1	D 900mmϕ x 300mm^L, 3.8kW
Zn Filter	1	D 900mmϕ x 300mm^L, 3.8kW
Pump	4	0.2m^3/min, 8.8kW
Tailing Thickener	1	D 8mϕ, 3.7kW
Pump	2	0.5m^3/min, 22kW

Fig.45 Layout of Mineral Dressing Plant

(3) Crushing section

Crushing method Tow stage grinding(Closed Circuit in Second)

	Primary	Jaw Crusher
	Secondary	Cone Crusher

Treatment ore per 1 hour (t)	(Average)7.2
Max feed ore size (mm)	180
Stockyard Volume (t)	600
Hopper Volume (t)	10

(4) Grinding section

Grinding method Single Ball mill (Closed circuit)

Treatment ore per 1 hour (t)	(Average)4.2
Feed size at 80% (mm)	3.5
Final grinding size at 80% (μ)	185
Classifier recycle load (%)	300
Mill stock yard volume (t)	100

(5) Flotation section

Flotation method Pb·Zn Bulk differential flotation

(5)－1 Pb·Zn bulk flotation

Cleaning method Two stage

Density and flotation time

	Density (%)	Time (min)
Conditions	35	3
Roughing	35	10
Primary cleaning		4
Secondary Cleaning		4
Roughing water (pH)		10

(5)－2 Pb Differential flotation

Cleaning method Two stage

Density and flotation time

	Density (%)	Time (min)
Conditions	20	3
Roughing	20	10
Primary cleaning		3
Secondary Cleaning		6
Roughing water (pH)		9

(5)－3 Mineral dressing test result

	Pb concentrate	Zn concentrate
Concentrate grade (%)	69.0	51.0
Recovery (%)	85.2	79.2

(6) Concentrate section

Dewatering method   vaccumed filteration dewater

(6)－1 Pb concentrate

Thickener underflow density (%)	40
Concentrate size (%) －100mesh	85
Concentrate moisture (%)	7
Concentrate specific gravity	6.9

(6)－2 Zn concentrate

Thickener underflow density (%)	40
Concentrate size (%) －100mesh	70
Concentrate moisture (%)	9
Concentrate specific gravity	4.1

(7) Tailings section

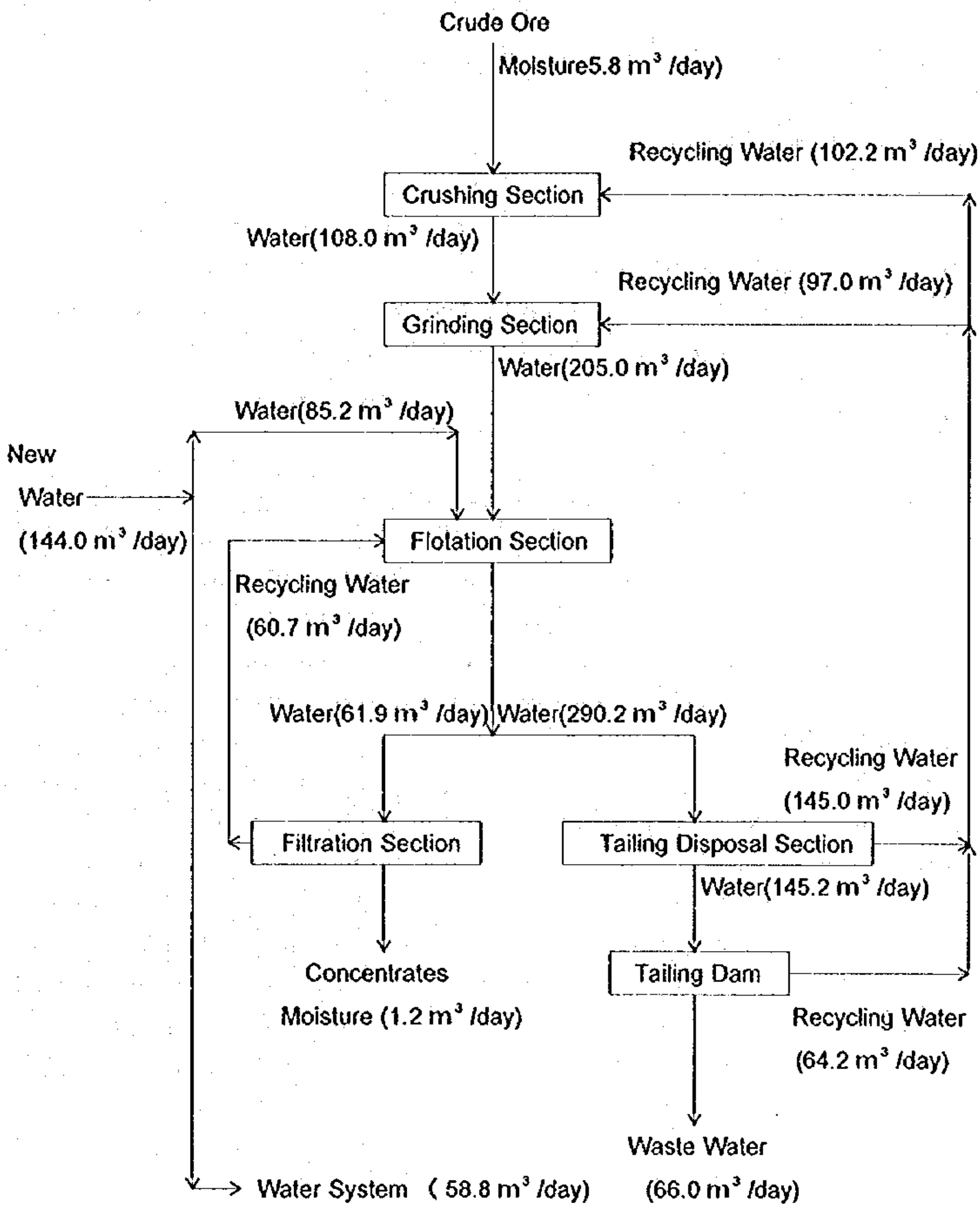
Concentration method   Cyclone and Thickener

Density of flotation tailing (%)	20
Transportation tailing density (%)	40
Cyclone overflow's sedimentation velocity (cm/hr)	25.56

(8) Water consumption

Water consumption volume (m^3/day)	404
New water supply volume (m^3/day)	144
Recycle rate (%)	64

The water balance of the mineral dressing is shown in Fig.46.



*) Crusing Section 2 Shifts / day、 After Grinding Section 3 Shifts / day

Fig.46 Water Balance of Mineral Dressing Plant

2.1.3 Mineral dressing process

The process flow sheet of mineral dressing plant is shown in Fig.47.

(1) Receiving of crude ore

Crude ore is hauled from underground to the surface by LHDs. A 600t stockyard is constructed near the crushing section of the mineral dressing plant for buffering ore feed and also for blending purpose. From the stockyard, ore is conveyed and fed by payloader into a receiving hopper of the crushing section.

(2) Crushing

The crushing process comprises a single series of two-stage crushing; the first stage forms an open circuit while the closed circuit is formed at the second stage. Main equipment are one each of primary and secondary crusher, a vibrating screen which, together with the secondary crusher, forms the closed circuit, another vibrating screen for washing purpose and a spiral classifier.

A chain feeder(500mmW) extracts ore from the receiving hopper and feed it to a jaw crusher(510mm x 230mm). Crushed ore is fed by a conveyer to a washing screen(600mmW x 1,800mmL, single deck, opening size 10mm) for slime removal.

The oversize of the washing screen is fed by a conveyer to a cone crusher(600mmW) while the undersize enters in a spiral classifier. Ore crushed by the cone crusher is , in turn, fed by another conveyer to a vibrating screen(600mmW x 1,800mL, single deck, opening size 10mm). The oversize of this screen is returned to the cone crusher while the undersize is the final crushed product. On the other hand, the undersize of the washing screen fed to the spiral classifier is separated into slime and raking sand. The classifier slime is stored in a washing thickener(4m dia) while the raking sand joins the undersize of the vibrating screen and these are transported to a mill bin as the final crushed product.

Overflow of the washing thickener is recycled to the washing screen while underflow is fed to a ball mill.

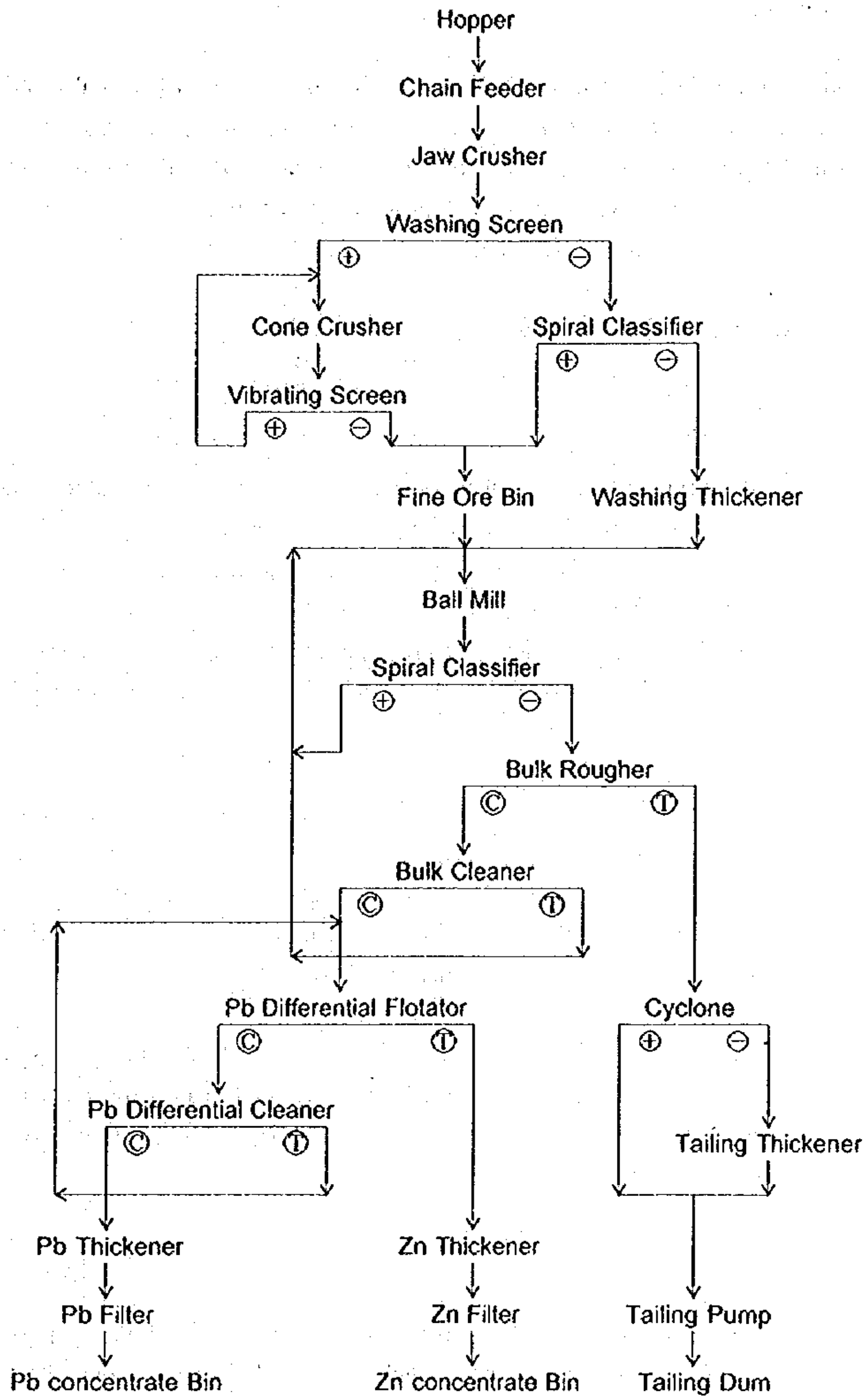


Fig.47 Process Flowsheet

(3) Grinding

Ore from the grinding section's bin is fed by conveyor to a ball mill (1500mm ϕ x 1200mmL) for grinding. Ground ore, in turn, enters the spiral classifier(600mm ϕ) and is separated into flotation feed and return sand. The return sand from the classifier is returned to the ball mill. The flotation feed is sent to the flotation section by a pump.

The grinding mill bin has a storage capacity of 100t.

(4) Flotation

The flotation process is composed of a single series of bulk flotation circuit(4 roughers of $0.6m^3$ and 8 cleaners of $0.28m^3$) and a single series of Pb differential flotation(4 roughers of $0.6m^3$ and 6 cleaners of $0.28m^3$). Flotation feed is subjected to bulk flotation roughing so that lead and zinc may be recovered as float, whilst the roughing tailing becomes the final tailing. The roughing float is further upgraded through two-stage cleaners and sent to the differential flotation. Cleaner tailings is returned to the grinding circuit.

Float in the bulk flotation is subjected to differential flotation whereby lead may be recovered as float while zinc becomes tailing. Differential flotation roughing float is upgraded to lead concentrate by the two-stage cleaners. Differential flotation cleaner tailing is returned to differential flotation feed. Zinc as differential flotation tailing is stored in the zinc concentrate thickener.

In the bulk flotation roughing and differential flotation, neither regrinding of cleaner tailing, etc. nor intermediate thickening within the flotation circuit is done because of easy liberation. Details of the flotation circuit has to be determined on a basis of pilot test results.

(5) Dewatering of concentrates

Lead concentrate and zinc concentrate, which are the final flotation products, enter their respective concentrate thickeners. Thickened concentrates are dewatered by the drum filters(900mm ϕ x 300mmL), respectively. Overflow of the both thickeners of concentrate are recycled to the flotation section.

Dewatered concentrates are sent directly to the concentrate stockyard by conveyors and stored there. When necessary during wintertime, they are dried by drum-type dryers before storage.

Concentrates at the stockyard are loaded on to trucks by a payloader and weighed by a truck scale before shipment.

2.1.4 Ancillary facilities

(1) Instrumentation

The instrumentation is limited to a minimum necessary level and installation of automation equipment is not considered. For instrumentation, pH meters are installed at the places where bulk flotation feed and differential flotation feed enter. A control center is not envisaged.

(2) Flotation reagents, etc.

Flotation reagents to be used are sodium cyanide, copper sulfate, sodium sulfide, calcium hydroxide, isopropyl xanthate and MIBC.

Reagents are dissolved at the reagent preparation room, sent to storage tanks at the flotation section and added by constant feeders where necessary.

Following are the assumed requirements of reagents, etc. per ton of crude ore:

Sodium cyanide (g)	200
Copper sulfate (g)	70
Sodium sulfide (g)	50
Calcium hydroxide (g)	600
Isopropyl xanthate (g)	50
MIBC (g)	35
Balls for the ball mill (g)	675
Electric power (kWh)	43.2

(3) Dust collection

Dust covers are installed on the cone crushers and the vibrating screens of the crushing section, to draft dust particles, which are treated by a wet-type dust collector.

(4) Sampling

To check operation performance of the mineral dressing plant, one each of samplers is installed at the four points for sampling of the bulk flotation feed, bulk flotation tailing, final concentrate of differential flotation(Pb conc.) and differential flotation tailing(Zn conc.).

(5) Others

Power : Power for the mineral processing plant equipment are distributed at each of the three blocks, ie., 1) crushing block, 2) grinding-flotation-tailing pipage block, and 3) concentrate dewatering block.

Electric hoist : One each of 5t electric hoists is installed at the crushing and grinding sections while a 2t hoist at the flotation section, as these sections have heavy equipment.

Storage space for supplies : Appropriate spaces are provided at the crushing section, flotation section and workshop for storing supplies.

Workshop : The workshop built by the Japanese team near the shaft mouth is utilized.

Assay and laboratory : Rooms for assay, testing and sample preparation are built adjacent to the mineral dressing plant, to conduct the minimum necessary testing for the purpose of process control.

Office: A site office is provided within the mineral dressing plant.

2.2 Facilities for Tailing Disposal

2.2.1 Thickening

Flotation tailing with a concentration of around 25% is thickened by cyclones to around 45%. Cyclone overflow is sent to a tailing thickener (8m ϕ).

Thickened cyclone underflow and thickener underflow are sent together to the tailing dam by two sand pumps. Waste thickener overflow and tailing dam overflow are recycled to the crushing, grinding and flotation processes.

2.2.2 Tailing disposal

The tailing disposal has a capacity of 11–year service and is designed to allow further expansion. The dam site is selected at a location nearest possible to the mineral dressing plant and out of the area of ore deposits.

As the site is too gently inclined for tailing to be transported by the natural gradient, tailing will be transported by pipage pressurized by Wahman pumps. However, a steep gradient is unnecessary for transportation of tailing dam overflow, which will be sent back by the natural gradient to the mineral dressing plant from the tailing dam to be sited in the upper side of the plant.

The tailing dam is an earth dam of a pond type with an embankment at the lower part to be made from soil excavated in the upper part of the site, taking advantage of the gentle gradient.

With a view to minimize the initial investment, the dam will be partitioned so it may be constructed in four separate phases, balancing the cutting volume with the banking volume.

In the upper side of the dam, an unlined channel is cut along the dam, whereby rain water is drained into a dry stream bed in the lower part.

For intake of the overflow water, a particular portion of the embankment is used for height adjustment, so that overflow can be controlled in relation to the ups and downs of the dam water level.

A plan, a location map, and a longitudinal section and a hydrographic map of the tailing dam are exhibited in Fig.48, and Apx.7, 8 and 9, respectively.

Design of various channels is based on the following assumptions:

- 60 minute rainfall depth probability : 7mm/hour
- Tailing transport volume : 300m ^{3} /8 hours
- Overflow water transport volume : 0.01m ^{3} /second

Main specifications of the tailing dam are shown in the following Table:

Descriptions	Quantity	Remarks
Volume	170,000m^3	
Area	43,000m^2	
Earth dam height	max. 5m	
Crest total length	1,300m	
Crest width	5m	
Slope gradient		
Inside wall	1 : 1.8	
Outside wall	1 : 2.0	
Cutting face of slope	1 : 1.5	
Tailing pipeline	1,350m	
Channel of overflow drainage cycle	640m	
Drainage for rainfall	1,560m	

Table 31 Main Specifications of the Tailing Dam

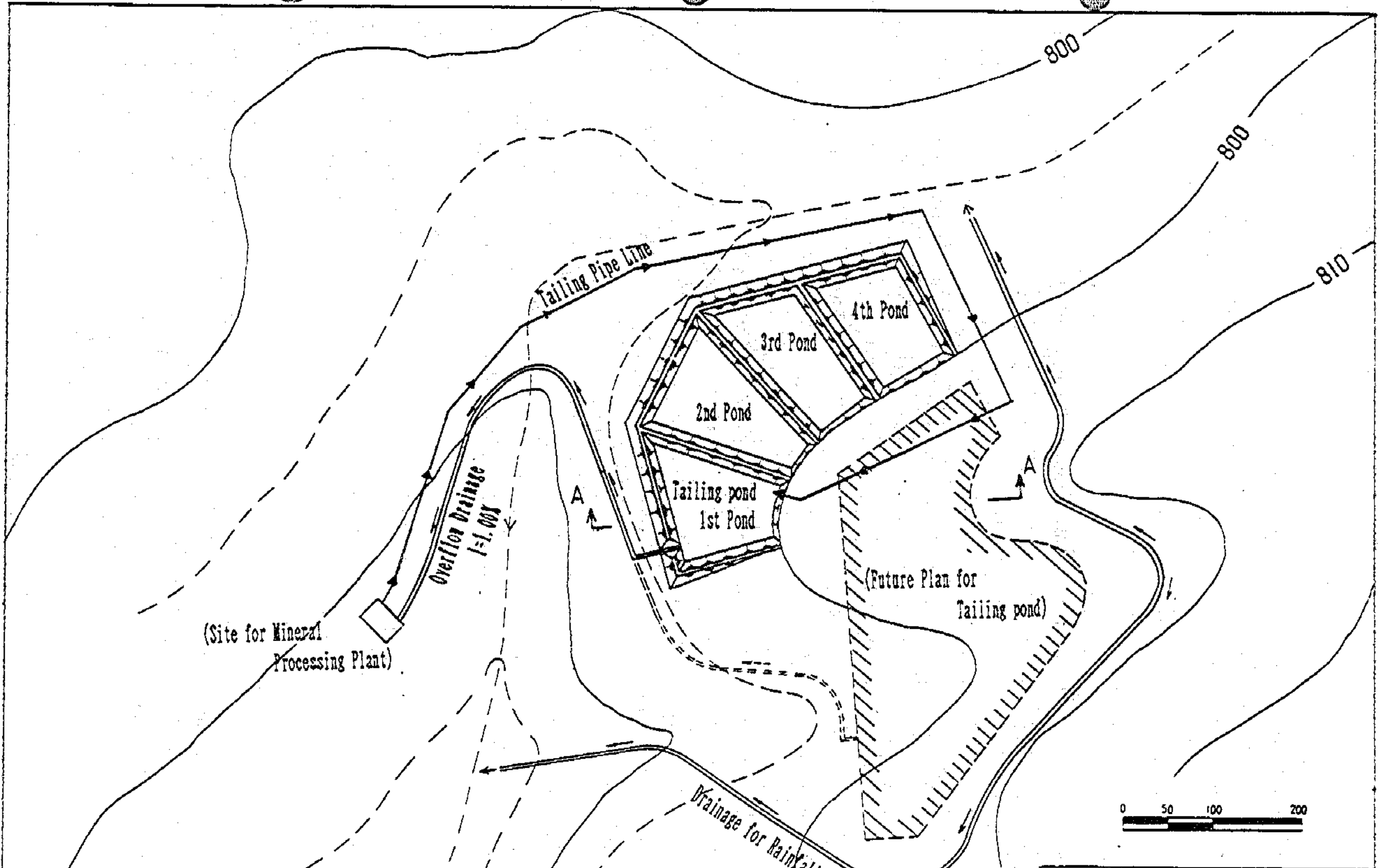


Fig.48 Tailing Pond Plan

Scale		Date	
Drawing No.			

Chapter 3. Auxiliary Facilities Plan

3.1 Power Receiving and Distribution

3.1.1 General

For electric power supply, two alternatives were considered: (1) in-house power generation using the existing 750KVA generator which was used by the Japanese survey team; and (2) receiving power from a thermal power station in Choybalsan.

In case of the in-house power generation, light oil supply is uncertain and the price is expensive. The state-run power station in Choybalsan, cap. 36,000kW, is operated with lignite mined from Adun Choulung open-pit colliery and is supplying power to the city of Choybalsan and also to the three nearby villages of Helren, Baitomung and Toblagan.

Power is transmitted at 35KV from Choybalsan to Helren(approx. 50km) and at 10KV from Helren to Habilka(approx. 70km). From Habilka to Tsav, no transmission line has been laid. The Choibalsan power station is said to have a surplus supply capacity of 2,000 to 3,000kW in summertime and about 1,000kW in wintertime.

In order for the Tsav mine to receive power supply from the Choybalsan station, a 35KV transmission line would have to be laid by the power station from the village of Habilka to Tsav(approx. 80km). 50% of the cost is to be borne by the user.

In this project, the alternative of receiving power from Choybalsan is chosen, since the long-term stable power supply at low cost is essential. The alternative would be beneficial to the future development of the surrounding area, as well.

3.1.2 Power demand

The maximum power demand of the Tsav mine's production units and other facilities is estimated at about 600KW, whilst the annual power demand is about 2,300MWh, which is broken up as follows:

Section	Maximum Power Demand(KWh)	Annual Power Demand(MWh)
Mining	400	960
Mineral Dressing	200	1,296
Total	600	2,256

3.1.3 Power receiving and distribution facilities

In order for the Tsav mine to receive power from Choybalsan, to be added to the existing facilities are ; (1) an outdoor cubicle for receiving power at 35KV, (2) a 750KVA transformer, and (3)a 3.3KV outdoor cubicle.

(1) Outdoor cubicle for power receiving at 35KV

The cubicle contains a disconnecting switch(DS) board and a vacuum circuit breaker(VCS) panel, to which a protective relay for power receiving has to be added.

(2) 750KVA transformer

The transformer is to step down from the receiving tension of 35KV to 3.3KV, the distribution tension within the mine. The required capacity is put at 750KV, considering the mentioned power demand.

(3) 3.3KV outdoor cubicle

This cubicle is to send the power stepped down by the transformer from 35KV to 3.3KV to the electric systems existing at the mine. It has no synchronizing device as parallel feed with the diesel generator is not considered.

3.1.4 Power cost comparison

Estimated power cost in case power is supplied by Choybalsan

Transmission line laying cost	US$ 267,000.–
Power receiving facilities	335,000.–
Power charges (for mine life)	*[1] 1,985,000.–
Total	US$ 2,587,000.–

Estimated power cost in case of in–house generation

Light oil and lubricant	*[2] US$4,417,000.–

Remarks: *[1] (2,256,000kWh x 11 years x 0.08$/kWh)
*[2] ($1,100/day x 365days x 11 years)
The cost per day is based on the actual cost at Tsav.

In case of the in–house generation, labor cost for generator operators and generator repair costs have to be considered, in addition to the oil and lubricant costs. This indicates that receiving power from Choybalsan is far more economical than in–house generation.

3.2 Communication equipment

At present, communication to and from the mine site is done by wireless equipment provided respectively in Ulaan Baartar, Choybalsan and Tsav. No additional investment for communication such as telephone/facsimile via INMARSAT is contemplated in this project.

3.3 Water Supply Facilities

Currently, the drinking water is brought to Tsav by a water wagon from a well dug by an ex–USSR team 6km away from Tsav, stored in a 50m^3 tank, and distributed to each section. This project follows the current system and contemplates no extra investment.

As for industrial water, 100 ℓ/min out of some 200 ℓ/min of the estimated total underground drainage will be sent to a water storage tank of the mineral dressing plant. The fresh water will be used mainly for the cleaning process whilst, for the other processes, overflow of the tailing dam and thickeners will be recycled, in an effort to minimize water consumption.

3.4 Workshop

Repair of the underground heavy machinery and mineral dressing equipment is of great importance. In this project, utilization of the existing workshop(24m x 13m) built by the Japanese survey team near the shaft mouth of the No.4 vein is contemplated so that an additional investment may be done without.

3.5 Ancillary Facilities and Vehicles in Common Use

3.5.1 Office and Stuff accommodations

A 660m ^{2} building which houses an office and staff accommodations is planned to be built at the site where accommodation for the Japanese survey team was located. The site is conveniently located in respect of water supply and drainage, sewage systems and electric power facilities.

For workers' accommodation, the existing building which was used as the Mongolians' accommodation during the survey period is utilized.

3.5.2 Vehicles in common use

A microbus for transporting personnel between Choybalsan and Tsav and a 11t dump truck for transporting concentrates to Habilka station is included in this project, as vehicles in common use. Facilities such as the ambulance and hospital are not considered in this project.

3.6 Transportation of Concentrates

3.6.1 Stockyard

Lead concentrate and zinc concentrate are stored at a stockyard near the mineral dressing plant until shipment. Storage capacity of the yard is planned to meet the monthly output, ie., 200t of lead concentrate and 110t of zinc concentrate.

3.6.2 Concentrate transportation route

Feasibility of concentrate shipment to Russia has been studied. Under the current economic conditions of the nation, however, the Russian smelters–refineries at Zabaykarsk and Sherlovaya Cora have curtailed or suspended operation and their future trends are hardly predictable. Therefore, it is assumed in this project that the concentrates are to be shipped to coastal smelters–refineries in Japan despite the problems of shipping lot and shipping cost.

For this alternative, the following two routes are considered:

(1) Tsav → Borzya → Karymskoe → (the Siberian Railways) → Vladivostok → (ocean transportation) → Japan

(2) Tsav → Borzya → Zabaykarsk → (the Chinese Railways) → Dalian → (ocean transportation) → Japan

The transportation costs come to US$ 80/t and $ 70/t, respectively. Apparently, the second alternative route via China is less expensive. However, in case of shipment via China, bulk shipment of concentrates is not allowed, which involves extra bagging costs; furthermore, troubles may arise from the difference between the rail gauge of China and that of Mongolia and Russia.

Taking all these factors into consideration, the first alternative route via Russia is chosen for this project. For the transportation of concentrates to the nearby Habilka station, an 11t dump truck will be used.

Chapter 4. Production Plan and Personnel

4.1 Production Plan

The production plan is based on the following assumptions:

Ore reserves and grade: 1,544,627t

Cu	Pb	Zn	Au	Ag
0.24%*	6.84%	4.01%	1.3g/t*	263g/t

Average mining recovery: 18%
Average dilution: 21%
Minable crude ore and grade: 332,464t

Cu	Pb	Zn	Au	Ag
0.22%*	6.37%	2.94%	1.22g/t*	161g/t

Annual production of crude ore: 30,000t
Arithmetical mine life: 11 years
Mineral dressing recovery:

Pb	Zn	Au	Ag
88.3%	92.8%	79.0%	82.6%

Annual production of concentrates: Pb conc 2,731t
Zn conc 1,351t

Grade of concentrates:

	Cu(%)	Pb(%)	Zn(%)	Au(g/t)	Ag(g/t)
Pb conc	0.32	69.0	5.0	11.10	1,507
Zn conc	2.77	4.4	51.0	2.09	307

Remarks: * indicates an average grade of the proved and probable ore reserves. (For the possible ore reserve, no assay has been done of Cu and Au.)

The mineral dressing performance shown in Table below is used for projection of mine revenues.

	Quantity (t)	Grade					Distribution				
		Au g/t	Ag g/t	Pb %	Zn %	Cu %	Au %	Ag %	Pb %	Zn %	Cu %
Crude ore	30,000	1.22	161	6.4	2.9	0.22	100.0	100.0	100.0	100.0	100.0
Pb con	2,371	11.01	1,507	69.0	5.0	0.32	71.3	74.0	85.2	13.6	11.5
Zn con	1,351	2.09	307	4.4	51.0	2.77	7.7	8.6	3.1	79.2	56.6
Tail.	26,278	0.29	32	0.9	0.2	0.08	21.0	17.4	11.7	7.2	31.9

Table 32 Mineral dressing performance

4.2 Personnel Plan

The organization for operation and personnel arrangement by section and class are shown in Fig. 49.

Organization of Operation

- General Director(1)
  - Head Office in Ulaan Baataar — General Mnager (1)
    - Personal (1)
    - General Affairs (1)
    - Accounting (1)
    - Purchase (1)
  - Office in Choybalsan — Manager(1)* (*:Additional post)
    - Clerk (2)
  - Tsav Mine — Superintendent of mine(1)
    - Geological Division — Manager(1) — Cheif Geologist (1)
      - Drilling man (4)
      - Surveyor (2)
      - Geologist (1)
      - Clerk (1)
    - Mining Division — Manager(1) — Chief Mining Engineer (2)
      - Miner (18)
      - Transport Man (4)
      - Clerk (1)
    - Mineral Dressing Division — Manager(1) — Chief Metallurgical Engineer(3)
      - Worker of dressing(17)
      - Clerk (1)
    - Engineering Division — Manager(1) — Chief Mechanical Engineer (1) / Chief Electrical Engineer (1)
      - Mechanic (5)
      - Civil Worker (2)
      - Electrician (3)
    - Administration Division — Manager(1)*
      - Accouting (1)
      - Purchase (1)
      - Driver (5)
      - Guard (4)
      - Lodge Keeper (2)
      - Kichen Worker (4)

		Stuff	Worker		Total
			Surface	Underground	
		Persons	Persons	Persons	Persons
Head Office Ulaan Baataar		6			6
Office in Choybalsan		(1*)	2		2
Tsav Mine	Mining	3	1	22	26
	Dressing	4	18		22
	Others	7*	31	4	42
Total		20	52	26	98

*:Additional post

Fig.49 Organization of Operation

Chapter 5. Environmental Survey

5.1 General

The Tsav area is situated at a gentle hill in the steppe without trees. There is no streams with constant water flow. Only during a limited period of a year, rain water occasionally form puddles at hollows on the surface but such surface water evaporates due to the region's dry climate and puddles gradually disappear. In some cases, the underground water table appears on the surface at sunken places, forming springs, which rarely last through a year, though. The local nomads draw up and use the ground water for their living.

In August when a survey was conducted, some puddles with rain water were seen here and there in the area; however, there was no ponds or swamps where surface water stays permanently, in the downstream side of the water systems near the planned sites for the mineral dressing plant and the tailing dam and near the water system, into which the underground mine water is currently drained. The drained water, while flowing a short distance, say 1km, on the surface, sinks into the ground.

In the light of these observations, it is considered that, after the mine has been developed, water pollution by the drainage from mine and mineral dressing plant is unlikely to widely spread over the downstream side of the local water system but likely to be confined within a limited area around the mine and plant. For this reason, the survey on soil and ground water was limited only to this area.

5.2 Soil

In the vicinity of Tsav deposits, surface soil samples were collected at four points at a dry stream into which the mine and plant drainage is expected to flow. The four points are: the upper side of the drainage point, near the drainage point and two points in the lower side of the drainage point. For comparison purpose, another soil sample was taken at a place away and independent from the mentioned dry stream. These samples were brought back to Japan for assay of the five heavy metal contents(Cu, Pb, Zn, Fe and Cd) as well as arsenic content.

The assay results in Table 33 indicate a certain difference in Pb and Zn concentration between the upper sampling point and the lower sampling points while no difference in concentration of the other elements is indicated. Cu, Ad and As are in low grades whilst Fe is considered to be abundant over an extensive area, independently of the ore deposits. The area, in which the difference in Pb and Zn concentration is recognized, is not so wide. It is anticipated, therefore, that water pollution possibly caused by the mine development would be confined to a limited area in the lower side of the mineral dressing plant.

Assay of sample soil taken from the point chosen for comparison purpose indicated Cu 50ppm, Pb 340ppm, Zn 260ppm, Fe 2.5%, Cd <2.5ppm and As 31ppm, which serve as the background values before the mine development is commenced. The assay also indicates the arsenic content being somewhat high, which characterizes this area.

Sampling points	Upper side of shaft mouth upstream	Near shaft mouth immediately downstream	Lower side of shaft mouth mid-stream	Lower side of shaft mouth downstream	Comparative point (uninfluenced by ore deposits
Samples taken on : 11 August					
Cu (ppm)	60	40	48	45	52
Pb (ppm)	1000	470	420	260	340
Zn (ppm)	480	290	320	190	250
T-Fe (%)	2.6	2.3	2.5	2.7	2.5
Cd (ppm)	<2.5	<2.5	<2.5	<2.5	<2.5
As (ppm)	53	35	31	29	30

Table 33 Environmental Survey around Tsav Deposits(Assay of soil)

5.3 Underground Water

In order to check the present state of pollution of the underground mine water and well water in Tsav area and also to obtain background values, water quality measurement was conducted three times on different days of sample water collected at two points underground (water used for drilling and underground spring water at a separate location)and also of sample water taken from a well that is used for drinking, the latter serving for comparison with the former.

Three samples collected on the final day of measurement and one sample taken from another well(different from the abovementioned well) were brought back to Japan for assay of the five heavy metal contents and arsenic content, similarly to the case of soil. The results of the assay and water quality measurement are shown in Table 34 and 35, respectively.

Based upon these results, suitability of the Tsav underground water for mineral dressing use was also studied.

The items of water quality measurement are pH, electric conductivity, turbidity, dissolved oxygen, water temperature, Cu, Fe^{2+}, total Fe, Zn, Ni, Mn, Cr^{6+}, COD, NO_3 −N and NH^{4+} −N, of which pH, electric conductivity, turbidity, dissolved oxygen and water temperature were measured by instruments while the other items were checked by summary measurement with test papers. Underground water and well water were measured once every week over three weeks, but no significant difference was found.

Underground water taken at two points showed high Zn and total Fe among the metal contents while, among the organic components, COD, NH^{4+} −N and NO_3 −N were high. In the underground water, a minute amount of zinc was detected, which indicates that zinc contained in the ore deposits dissolves into underground water. The total Fe contained in the drilling water suggests influence of the iron piping. The recognized COD, NO_3 −N and NH^{4+} −N are presumed to derive from the pastured animals' droppings which flowed with rain water into the vertical shaft and decomposed, and/or to be influenced by components of the explosives used underground.

In the water samples taken from two different wells outside Tsav area, organic pollutants such as COD and NO_3 −N of minute quantities were detected, which are also presumed to derive from the feces of pastured horses and cows. Except this, no other questions were encountered.

On the basis of the survey findings, suitability of the underground water for the mineral dressing process water was studied, as well. Although all the water surveyed is hard water with somewhat high electric conductivity, it may be concluded that there would be no problem whatsoever if the underground water is used for the mineral dressing of ore.

As for drinking water, it is desirable to use well water located in an area where Zn and Fe contents, as well as organic pollutant contents, are low, although it is not impossible to use the underground water after certain treatment.

Sampling points	Well water	Underground water	Underground drilling water	Well water near magazine
Samples taken on : 14 August				
Cu (ppm)	<0.01	<0.01	<0.01	<0.01
Pb (ppm)	<0.05	<0.05	<0.05	<0.05
Zn (ppm)	0.07	0.31	1.5	0.01
T-Fe (ppm)	0.55	0.22	1.5	0.07
Cd (ppm)	<0.005	<0.005	<0.005	<0.005
As (ppm)	<0.01	<0.01	<0.01	<0.01

Table 34 Environmental Survey around Tsav Deposits
(Water Quality Analysis)

Sampling Points	Well water	Underground water	Underground water drilling	Well water	Underground water	Underground water drill-water	Well water	Underground water	Underground water drilling	Well water near Magazine
Date	taken on 2 August			taken on 8 August			taken on 14 August			
pH	8.2	8.2	7.4	8.5	8.2	7.2	8.4	8.1	7.3	7.7
Electric conductivity (mS/cm)	1.3	1.1	1.3	1.3	1.0	1.3	1.3	1.0	1.3	1.1
Turbidity (NTU)	0	10	100	2	6	240	2	12	111	4
Dissolved oxygen (ppm)	7.3	11.2	6.5	8.7	10.8	7.5	9.9	10.6	8.5	11.2
Water temperature (℃)	14.3	8.9	15.4	13.4	13.0	18.8	10.4	10.6	15.2	11.5
Mesurement with test paper										
Cu (ppm)	<0.5	<0.5	<0.5	<0.5	<0.5	0.5	<0.5	<0.5	<0.5	<0.5
Fe^{2+} (ppm)	<0.2	<0.2	<0.2	<0.2	<0.2	2	<0.2	<0.2	≦0.2	<0.2
T-Fe (ppm)	≦0.2	<0.2	0.4	≦0.2	<0.2	7	<0.2	≦0.2	1	≦0.2
Zn (ppm)	0	0	1	0	0	2	0	0	2	0
Ni (ppm)	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Mn (ppm)	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	0.5	<0.5
Cr^{6+} (ppm)	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
COD (ppm)	2	15	18	10	20	40	3	8	20	5
NO_3-N (ppm)	<0.006	0.03	<0.006	<0.006	0.04	<0.006	<0.006	0.03	<0.006	<0.006
NH^{4+}-N (ppm)	0.6	0.6	0.8	0.4	0.4	0.6	0.4	0.6	1.2	0.4

Table 35 Environmental Survey around Tsav deposits
(Water Quality Mesurement)

Chapter 6. Revenue Plan

6.1 Concentrate Prices

6.1.1 Assumptions

(1) Quotation of metals: Pb \$650/t, Zn \$1,100/t, Au \$390/tr oz, Ag \$5.30/tr oz.

(2) Payable metals: Pb concentrate – Pb : 95% (Minimum deduction 3%)
Au : 1g less; recovery 90%
Ag : 95% (Minimum deduction 100g)
Zn concentrate – Zn : 85% (Minimum deduction 8%)
Au : 1g less; recovery 65%
Ag : 4 tr oz less; recovery 60%

(3) Treatment charges(T/C) and refining charges(R/C):
Pb concentrate – Pb : T/C \$160/DMT; scale + 14 ¢ /\$ (Base=\$600)
Ag : R/C 30 ¢ /tr oz
Zn concentrate – Zn : T/C \$190/DMT

Zinc quotation	\$1,250 ~	\$1,250 ~ 1,000	~ \$1,000
T/C scale	+15 ¢ /\$	+13 ¢ /\$	−8 ¢ /\$

(4) Penalty: Not applied as impurities are yet to be assayed.

(5) Concentrate transportation to Japan via Siberian route: \$80/WMT, incl. transportation loss(0.5%) and moisture(8%)

(6) Transportation insurance: 0.077% x (concentrate price – T/C – R/C)

6.2 Revenues

Revenues calculated on these assumptions are shown in Table 36.

Production 、Revenue Plan

Quantity of mining crude ore:100t/day x 300days/year=30,000t/year
Quantity of treated ore:100t/day x 300days/year = 30,000t/year

Pb·Zn-Concentrate→Export to Japan

		unit	-1	1	2	3	4	5	6	7	8	9	10	11	12	Total
Crude ore production	30 '000t/y	'000t/y		30	30	30	30	30	30	30	30	30	30	30		330
Grade of Pb	6.40 %	%		6.40	6.40	6.40	6.40	6.40	6.40	6.40	6.40	6.40	6.40	6.40		6.40
Minable Zn	2.90 %	%		2.90	2.90	2.90	2.90	2.90	2.90	2.90	2.90	2.90	2.90	2.90		2.90
Au	1.22 g/t	g/t		1.22	1.22	1.22	1.22	1.22	1.22	1.22	1.22	1.22	1.22	1.22		1.22
Ag	161 g/t	g/t		161	161	161	161	161	161	161	161	161	161	161		161
Pb- Quantity		DMT/y		2,371	2,371	2,371	2,371	2,371	2,371	2,371	2,371	2,371	2,371	2,371		26,079
Concentrate Grade of Pb 69.0%	Recovery 85.20%	%		69.00	69.00	69.00	69.00	69.00	69.00	69.00	69.00	69.00	69.00	69.00		69.00
Grade of Au	Recovery 71.30%	g/t		11.00	11.00	11.00	11.00	11.00	11.00	11.00	11.00	11.00	11.00	11.00		11.00
Grade of Ag	Recovery 74.00%	g/t		1,365	1,365	1,365	1,365	1,365	1,365	1,365	1,365	1,365	1,365	1,365		1,365
Zn- Quantity		DMT/y		1,351	1,351	1,351	1,351	1,351	1,351	1,351	1,351	1,351	1,351	1,351		14,862
Concentrate Grade of Zn 51.0%	Recovery 79.20%	%		51.00	51.00	51.00	51.00	51.00	51.00	51.00	51.00	51.00	51.00	51.00		51.00
Grade of Au	Recovery 7.70%	g/t		2.10	2.10	2.10	2.10	2.10	2.10	2.10	2.10	2.10	2.10	2.10		2.10
Grade of Ag	Recovery 8.60%	g/t		579	579	579	579	579	579	579	579	579	579	579		579
Payable Metal Pb-C Pb Transportation	95 %	t		1,546	1,546	1,546	1,546	1,546	1,546	1,546	1,546	1,546	1,546	1,546		17,009
Au loss 0.5%	90 %	toz		683	683	683	683	683	683	683	683	683	683	683		7,508
Unit Discount	-1 g															
Ag	95.0 %	toz		98,347	98,347	98,347	98,347	98,347	98,347	98,347	98,347	98,347	98,347	98,347		1,081,819
Zn-C Zn Transportation	85 %	t		583	583	583	583	583	583	583	583	583	583	583		6,410
Au loss 0.5%	65.0 %	toz		31	31	31	31	31	31	31	31	31	31	31		340
Unit Discount	-1 g															0
Ag	60.0 %	toz		11,788	11,788	11,788	11,788	11,788	11,788	11,788	11,788	11,788	11,788	11,788		129,673
Unit Discount	-4 toz															0
Conc Price Pb-C Pb Price	650 $/t	'000US$		1,005	1,005	1,005	1,005	1,005	1,005	1,005	1,005	1,005	1,005	1,005		11,056
Au Price	390 $/toz	'000US$		266	266	266	266	266	266	266	266	266	266	266		2,928
Ag Price	5.3 $/toz	'000US$		521	521	521	521	521	521	521	521	521	521	521		5,734
Zn-C Zn Price	1,100 $/t	'000US$		641	641	641	641	641	641	641	641	641	641	641		7,051
Au Price	390 $/toz	'000US$		12	12	12	12	12	12	12	12	12	12	12		133
Ag Price	5.3 $/toz	'000US$		62	62	62	62	62	62	62	62	62	62	62		687
Pb T/C	160 $/DMT	'000US$		394	394	394	394	394	394	394	394	394	394	394		4,333
Scale at 600$ +14c/$																
Ag R/C	30 c/toz	'000US$		30	30	30	30	30	30	30	30	30	30	30		325
Zn T/C	190 $/DMT	'000US$		255	255	255	255	255	255	255	255	255	255	255		2,810
Scale at 1,250$< +15c/$,1,250$> >1,000$	+13c/$、1,000$> -8c/$															0
Penalty	$/DMT/%	'000US$														0
	$/DMT/%	'000US$														0
Insurance																0
(Pb-C Price-T/C-R/C)x0.077%	0.077 %	'000US$		1.054	1.054	1.054	1.054	1.054	1.054	1.054	1.054	1.054	1.054	1.054		12
(Zn-C Price-T/C-R/C)x0.077%	0.077 %	'000US$		0.354	0.354	0.354	0.354	0.354	0.354	0.354	0.354	0.354	0.354	0.354		4
Freight																0
Pb-C Moisuture 8%	80.0 $/WMT	'000US$		204	204	204	204	204	204	204	204	204	204	204		2,242
Zn-C Moisuture 8%	80.0 $/WMT	'000US$		116	116	116	116	116	116	116	116	116	116	116		1,278
Revenue Pb-C	Price-T/C,R/C-insu.-Frei.-Pn	'000US$		1,164	1,164	1,164	1,164	1,164	1,164	1,164	1,164	1,164	1,164	1,164		12,806
Zn-C	Price-T/C,R/C-insu.-Frei.-Pn	'000US$		344	344	344	344	344	344	344	344	344	344	344		3,780
Total Revenue		'000US$		1,508	1,508	1,508	1,508	1,508	1,508	1,508	1,508	1,508	1,508	1,508	553	17,139

Table 36 Production、Revenue Plan

Chapter 7. Initial, Additional and Replacement Investments and Operating Expenses

7.1 Estimated Initial Investment

7.1.1 Basis for assumption

As regards applicable Mongolian laws and regulations, and for estimation of salaries and wages, machinery prices, construction costs and general price level, the data and information as of September 1995 when the survey was conducted are used as the basis for assumption.

(1) Development and construction period : One year

(2) Infrastructure : The power transmission line from Helren to Tsav(80km) is laid by the state-run Choybalsan Electric Power Company. 50% of construction cost is to be borne by the Tsav mine.

(3) Procurement of equipment and supplies : Those which are available in Mongolia are reinforcing steel bars and cement. All the other equipment, machinery and supplies have to be imported from Russia, China or Japan.

(4) Customs duty : 7.5% of the FOR prices of imported equipment and supplies is included in the operating expenses.

(5) Exchange rates:
Mongolian Togrog(Tg) per US Dollar – 450Tg/$
Japanese Yen(¥) per US Dollar – ¥100/$

(6) Escalation : Not considered.

(7) Rate of interest during construction : 6% p.a.

7.1.2 Initial Investment Summary

An annual projection of estimated divisional initial investment (up to project start-up) is indicated in Table.37, which may be summarized as follows:

	US$'000
(1) Mining division Excavation machinery, batch plant,trucks for mining operation	1,142
(2) Mineral dressing division Construction of mineral dressing plant and equipment	1,386
(3) Auxiliary(common cost) division Power receiving facilities, transmission line cost sharing, office and accommodation construction	937
(4) Tailing disposal (Initial-phase construction expenses are included in the operating cost)	341
(5) Development preparation Head office overheads, customs duties	138
(6) Interest during construction	240
(7) Working capital(10% of divisional initial investment)	347
Initial Investment - Total	4,531

7.2 Additional and Replacement Investments

US$530,000 is appropriated for the replacement of two LHDs in the sixth year after start-up. In reality, the amount is insufficient for proper replacement capital since the other mining machinery and vehicles including the jumbo should also be replaced, which is burdensome to the project cash flow, though. Therefore, replacement investment has to be limited to the minimum necessary.

Additional investment in phased construction of the tailing dam is included in the repair expense under the mineral dressing division's operating expenses, as in the case of the initial investment, for the second, fifth and eighth years, respectively, after start-up of operation.

7.3 Estimated Operating Expenses

7.3.1 Basis for Estimation

(1) Exchange rate : Same as the initial investment

(2) Salary and wage : For estimation, personnel are classified into three categories: staff, surface and underground. (Further classification by type of job is not made.) To the estimated salaries and wages, social insurance premiums of 3% and 1.5%, and labor accident insurance premium of 1.0% are added.

(3) Consumables and machine parts : All the consumable items and machine parts, excepting cement and reinforcing steel bars, are assumed to be imports, which include explosives, detonators, mineral dressing reagents and fuels. The surveyed local prices in Mongolia of these goods are supposed to include customs duties.
{Customs duties: 15% on general goods and 7.5%(reduced rate) on mining machinery }

(4) Electric power rate :
At Choybalsan, the rate is 8 ¢ /kWh (= 36Tg/450Tg/US$)
Ref. 3.5 ¢ /kWh in Ulaan Baartaar

(5) Royalty : 10% of the revenues

(6) Agent's commission : US$5.00 per ton of concentrate

(7) Corporate income tax :

Profit before tax (PBT)	Tax rate
\$1 ~ 1,000	0%
\$1,001 ~ 33,300	15%
\$33,301 ~ 66,600	\$500 + (PBT – \$33,300) x 25%
\$66,601 ~ 100,000	\$1,300 + (PBT – 66,600) x 35%
\$100,001 ~	\$2,500 + (PBT – 100,000) x 40%

7.3.2 Operating Expenses Summary

Annual projection of the direct (divisional) operating expenses (from mining to concentrate production) and the indirect operating expenses are shown in Table 38 and 39, which may be summarized on an annual average basis, as follows:

	Annual average (US$'000)	per ton of crude ore (US$/t)
Mining	699	23.3
Mineral dressing	348	11.6
Head office	35	1.2
Direct operating exp – Total	1,082	36.1
Depreciation exp	317	10.6
Royalty	151	5.0
Agent's commission	19	0.6
Operating Expenses – Total	1,569	52.3

Chapter 8. Financial Valuation

8.1 Internal Financial Rate of Return

To estimate internal financial rate of return(IFRR), a leading index in the financial valuation, it is vital to make a financial projection based on a well-defined fund sources and application. In turn, cash inflow and outflow items are sorted out and a cash flow chart is drawn up, wherein the net inflow, or difference between cash inflow and outflow, is calculated so that a rate of return may be determined.

8.1.1 Sources and application of funds

Funds required for realization of the project are composed of the initial investment, additional investment, interest during construction and working capital. Raising of the funds is planned as follows:

(1) The project capital structure at the project start-up is assumed to comprize equity capital(30%) and borrowings(70%), which meets a normal equity-borrowings ratio in international projects. Funds required at the project start-up are those for the initial investment(in construction and equipment) and working capital. The working capital is estimated at 10% of the initial investment.

(2) For the borrowing, low interest loans from an international financing agency such as the World Bank is contemplated, as the interest rates in Mongolia is very high in terms of international standards. The borrowing at the project start-up is a long-term loan while shortages of funds during operation is to be funded by short-term loans. Interest rate for these borrowings is assumed to be 6% p.a.

(3) The 6% interest is to be paid against the outstanding loan balance as of the preceding year's end. The long-term loan principal is to be repaid in equal annual installments from the fourth to the eleventh year. The short-term loan principals are to be repaid whenever surplus cash flow is available.

(4) Based on the assumptions in (1) thru (3) above, the total project capital consisting of equity capital and borrowings is assumed to be $3,811,000, equal to the invested funds plus the working capital required for the operation phase. Therefore, the equity capital comes to $1,143,000 while the borrowings to $2,668,000.

8.1.2 Cash flow chart and IFRR

The project cash flow chart(profit and loss projection and fund sources and application) and IFRR are exhibited in Table 39.

The cash inflow side of the chart includes revenue from product sale and salvage value of equipment whilst investment expenditure, operating expenses, taxes and other expenses are included in the cash outflow side.

The corporate income tax is calculated in the following formula:

Corporate income tax = (Revenue − operating expenses − royalty − agent's commission − depreciation expense − interest) x applicable tax rate(ref. 5.3.1)

The straight−line depreciation which is generally used in Mongolia is applied to the project.

In the case of this project, the IFRR, a discount rate at which the present value of the total net cash inflow comes to zero, turns out to be (−)3.0%.

In this study, the income tax is calculated of the difference obtained by deducting all the expense(including interest) from the cash inflow (revenue) and, in turn, the tax is entered in the cash outflow; therefore, it is significant to evaluate the project's financial feasibility by comparing the IFRR and the interest rate for borrowing. The financial analysis for the project is considered unfavorable since the IFRR is far below the assumed interest rate of 6%.

Investment & Depreciation of each division

Unit:US$

Article	Div No	Investment	Invest Time	Life	Residual Book Value	Invest Rate	−1st year Book Value	−1st year Depreciation	1st year Book Value	1st year Depreciation	2nd year Book Value	2nd year Depreciation	3rd year Book Value	3rd year Depreciation	4th year Book Value	4th year Depreciation	5th year Book Value	5th year Depreciation	6th year Book Value	6th year Depreciation	7th year Book Value	7th year Depreciation	8th year Book Value	8th year Depreciation	9th year Book Value	9th year Depreciation	10th year Book Value	10th year Depreciation	11th year Book Value	11th year Depreciation	Note
Mining division																															
Jumbo for Mining 1 set	1	310,000		5	31,000	0.180			310,000	55,800	254,200	55,800	198,400	55,800	142,600	55,800	86,600	55,800	31,000		31,000		31,000		31,000		31,000		31,000		
Batcher Plant 1 set	1	350,000		5	35,000	0.180			350,000	63,000	287,000	63,000	224,000	63,000	161,000	63,000	98,000	63,000	35,000		35,000		35,000		35,000		35,000		35,000		
Equipment of Shotcreting 1 set	1	350,000		5	35,000	0.180			350,000	63,000	287,000	63,000	224,000	63,000	161,000	63,000	98,000	63,000	35,000		35,000		35,000		35,000		35,000		35,000		
Mixer Truck(1.6m3) 1 set	1	40,000		5	4,000	0.180			40,000	7,200	32,800	7,200	25,600	7,200	18,400	7,200	11,200	7,200	4,000		4,000		4,000		4,000		4,000		4,000		
Load Hauld Dump(3.8m3) 1 set	1	350,000		5	35,000	0.180													350,000	63,000	287,000	63,000	224,000	63,000	161,000	63,000	98,000	63,000	35,000		
Load Hauld Dump(1.7m3) 1 set	1	180,000		5	18,000	0.180													180,000	32,400	147,600	32,400	115,200	32,400	82,800	32,400	50,400	32,400	18,000		
Track(1t) x 2 pieces 1 set	1	32,000		5	3,200	0.180			32,000	5,760	26,240	5,760	20,480	5,760	14,720	5,760	8,960	5,760	3,200		3,200		3,200		3,200		3,200		3,200		
Wagon(350kg) x 2 pieces 1 set	1	60,000		5	6,000	0.180			60,000	10,800	49,200	10,800	38,400	10,800	27,600	10,800	16,800	10,800	6,000		6,000		6,000		6,000		6,000		6,000		
Drainage Pump 1 set	1	40,000		8	4,000	0.113					40,000	4,500	35,500	4,500	31,000	4,500	26,500	4,500	22,000	4,500	17,500	4,500	13,000	4,500	8,500	4,500	4,000		4,000		
Total of Mining division		1,712,000			171,200				1,142,000	205,560	976,440	210,060	766,380	210,060	556,320	210,060	346,260	210,060	666,200	99,900	566,300	99,900	466,400	99,900	366,500	99,900	266,600	95,400	171,200		
Mineral Dressing division																															
Crushing (Facilities.Thickner.Ore Strage)	2	334,000		9	33,400	0.100			334,000	33,400	300,600	33,400	267,200	33,400	233,800	33,400	200,400	33,400	167,000	33,400	133,600	33,400	100,200	33,400	66,800	33,400	33,400		33,400		
Milling	2	260,000		9	26,000	0.100			260,000	26,000	234,000	26,000	208,000	26,000	182,000	26,000	156,000	26,000	130,000	26,000	104,000	26,000	78,000	26,000	52,000	26,000	26,000		26,000		
Floatation	2	245,000		9	24,500	0.100			245,000	24,500	220,500	24,500	196,000	24,500	171,500	24,500	147,000	24,500	122,500	24,500	98,000	24,500	73,500	24,500	49,000	24,500	24,500		24,500		
Concentrate (Pb,Zn Thickner.Filter.Drier)	2	165,000		9	16,500	0.100			165,000	16,500	148,500	16,500	132,000	16,500	115,500	16,500	99,000	16,500	82,500	16,500	66,000	16,500	49,500	16,500	33,000	16,500	16,500		16,500		
Tailing (Thickner.Facilities)	2	110,000		9	11,000	0.100			110,000	11,000	99,000	11,000	88,000	11,000	77,000	11,000	66,000	11,000	55,000	11,000	44,000	11,000	33,000	11,000	22,000	11,000	11,000		11,000		
Pumping Facility (Recycle Drainage)	2	12,000		9	1,200	0.100			12,000	1,200	10,800	1,200	9,600	1,200	8,400	1,200	7,200	1,200	6,000	1,200	4,800	1,200	3,600	1,200	2,400	1,200	1,200		1,200		
Concentrate Loading Place	2	18,000		9	1,800	0.100			18,000	1,800	16,200	1,800	14,400	1,800	12,600	1,800	10,800	1,800	9,000	1,800	7,200	1,800	5,400	1,800	3,600	1,800	1,800		1,800		
Laboratory	2	61,000		9	6,100	0.100			61,000	6,100	54,900	6,100	48,800	6,100	42,700	6,100	36,600	6,100	30,500	6,100	24,400	6,100	18,300	6,100	12,200	6,100	6,100		6,100		
Building·Office (221 m²) 1set	2	181,000		9	18,100	0.100			181,000	18,100	162,900	18,100	144,800	18,100	126,700	18,100	108,600	18,100	90,500	18,100	72,400	18,100	54,300	18,100	36,200	18,100	18,100		18,100		
Total of Mineral Dressing		1,386,000			138,600				1,386,000	138,600	1,247,400	138,600	1,108,800	138,600	970,200	138,600	831,600	138,600	693,000	138,600	554,400	138,600	415,800	138,600	277,200	138,600	138,600		138,600		
Common division																															
Burden of Construction Cost (Electric Power Line)		267,000		15	26,700	0.060			267,000	16,020	250,980	16,020	234,960	16,020	218,940	16,020	202,920	16,020	186,900	16,020	170,880	16,020	154,860	16,020	138,840	16,020	122,820	16,020	106,800	16,020	
Office·Dormitory (660 m²) 1 set	3	165,000		9	16,500	0.100			165,000	16,500	148,500	16,500	132,000	16,500	115,500	16,500	99,000	16,500	82,500	16,500	66,000	16,500	49,500	16,500	33,000	16,500	16,500		16,500		
Microbus	3	60,000		5	6,000	0.180			60,000	10,800	49,200	10,800	38,400	10,800	27,600	10,800	16,800	10,800	6,000		6,000		6,000		6,000		6,000		6,000		
Truck for Transportation of Concentrate(11t)	3	90,000		5	9,000	0.180			90,000	16,200	73,800	16,200	57,600	16,200	41,400	16,200	25,200	16,200	9,000		9,000		9,000		9,000		9,000		9,000		
Receiving Facility of Electlic Power	3	355,000		15	35,500	0.060			355,000	21,300	333,700	21,300	312,400	21,300	291,100	21,300	269,800	21,300	248,500	21,300	227,200	21,300	205,900	21,300	164,600	21,300	163,300	21,300	142,000	21,300	
Total of Common division		937,000			224,000				937,000	80,820	856,180	80,820	775,360	80,820	694,540	80,820	613,720	80,820	532,900	53,820	479,080	53,820	425,260	53,820	371,440	53,820	317,620	37,320	280,300	37,320	
	G-Total	4,035,000			309,800				3,465,000	424,980	3,080,020	429,480	2,650,540	429,480	2,221,060	429,480	1,791,580	429,480	1,892,100	292,320	1,599,780	292,320	1,307,460	292,320	1,015,140	292,320	722,820	132,720	590,100	37,320	Depreciation Balance
per ton of Crude Ore										14.17		14.32		14.32		14.32		14.32		9.74		9.74		9.74		9.74		4.42	T-Depreciat	3,432,220	558,780

Table 37 Investment & Depreciation of each division

Direct Operating Expense

Unit :US$

Article	Unit	Price	-1st year Quantity	-1st year Amount	1st year Quantity	1st year Amount	2nd year Quantity	2nd year Amount	3rd year Quantity	3rd year Amount	4th year Quantity	4th year Amount	5th year Quantity	5th year Amount	6th year Quantity	6th year Amount	7th year Quantity	7th year Amount	8th year Quantity	8th year Amount	9th year Quantity	9th year Amount	10th year Quantity	10th year Amount	11th year Quantity	11th year Amount	Note
Mining division																											
Personal Expense																											
Staff	$/m-m	1,200.00			10	12,000	10	12,000	10	12,000	10	12,000	10	12,000	10	12,000	10	12,000	10	12,000	10	12,000	10	12,000	10	12,000	
Exterior	$/m-m	933.33			34	31,733	34	31,733	34	31,733	34	31,733	34	31,733	34	31,733	34	31,733	34	31,733	34	31,733	34	31,733	34	31,733	
Interior	$/m-m	1,120.00			26	29,120	26	29,120	26	29,120	26	29,120	26	29,120	26	29,120	26	29,120	26	29,120	26	29,120	26	29,120	26	29,120	
Total					70	72,853		72,853	70	72,853		72,853	70	72,853		72,853	70	72,853		72,853		72,853		72,853		72,853	
Exploitation Expense																											
Bits・Rods	$/t	3.37			30,000	101,100	30,000	101,100	30,000	101,100	30,000	101,100	30,000	101,100	30,000	101,100	30,000	101,100	30,000	101,100	30,000	101,100	30,000	101,100	30,000	101,100	
Explosives	$/t	0.53			30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	
Detonaters	$/t	0.09			30,000	2,700	30,000	2,700	30,000	2,700	30,000	2,700	30,000	2,700	30,000	2,700	30,000	2,700	30,000	2,700	30,000	2,700	30,000	2,700	30,000	2,700	
Fuels	$/t	0.15			30,000	4,500	30,000	4,500	30,000	4,500	30,000	4,500	30,000	4,500	30,000	4,500	30,000	4,500	30,000	4,500	30,000	4,500	30,000	4,500	30,000	4,500	
Machinary・Electric Parts	$/t	9.00			30,000	270,000	30,000	270,000	30,000	270,000	30,000	270,000	30,000	270,000	30,000	270,000	30,000	270,000	30,000	270,000	30,000	270,000	30,000	270,000	30,000	270,000	
Others(20% of abovementioned)	%	20				24,840		24,840		24,840		24,840		24,840		24,840		24,840		24,840		24,840		24,840		24,840	
Total						419,040		419,040		419,040		419,040		419,040		419,040		419,040		419,040		419,040		419,040		419,040	
Development Expense																											
Bits・Rods	$/m	138.30			240	33,192	240	33,192	240	33,192	240	33,192	240	33,192	240	33,192	240	33,192	240	33,192	240	33,192	240	33,192	240	33,192	
Explosives	$/m	21.60			240	5,184	240	5,184	240	5,184	240	5,184	240	5,184	240	5,184	240	5,184	240	5,184	240	5,184	240	5,184	240	5,184	
Detonaters	$/m	3.70			240	888	240	888	240	888	240	888	240	888	240	888	240	888	240	888	240	888	240	888	240	888	
Fuels	$/m	6.30			240	1,512	240	1,512	240	1,512	240	1,512	240	1,512	240	1,512	240	1,512	240	1,512	240	1,512	240	1,512	240	1,512	
Machinary・Electric Parts	$/m	338.00			240	81,120	240	81,120	240	81,120	240	81,120	240	81,120	240	81,120	240	81,120	240	81,120	240	81,120	240	81,120	240	81,120	
Others(20% of abovementioned)	%	20				8,155		8,155		8,155		8,155		8,155		8,155		8,155		8,155		8,155		8,155		8,155	
Total						130,051		130,051		130,051		130,051		130,051		130,051		130,051		130,051		130,051		130,051		130,051	
Electric Fee	$/kwh	0.08			960,000	76,800	960,000	76,800	960,000	76,800	960,000	76,800	960,000	76,800	960,000	76,800	960,000	76,800	960,000	76,800	960,000	76,800	960,000	76,800	960,000	76,800	
Mining direct Expense Total	$					698,745		698,745		698,745		698,745		698,745		698,745		698,745		698,745		698,745		698,745		698,745	
per ton of Mining direct Expense	$/t					23.29		23.29		23.29		23.29		23.29		23.29		23.29		23.29		23.29		23.29		23.29	
Mineral Dressing Division																											
Personal Expense																											
Staff	$/m-m	1,200.00			4	4,800	4	4,800	4	4,800	4	4,800	4	4,800	4	4,800	4	4,800	4	4,800	4	4,800	4	4,800	4	4,800	
Exterior	$/m-m	933.33			18	16,800	18	16,800	18	16,800	18	16,800	18	16,800	18	16,800	18	16,800	18	16,800	18	16,800	18	16,800	18	16,800	
Total						21,600		21,600		21,600		21,600		21,600		21,600		21,600		21,600		21,600		21,600		21,600	
Material Expense and others																											
Chemical Reagent	$/t	1.52			30,000	45,600	30,000	45,600	30,000	45,600	30,000	45,600	30,000	45,600	30,000	45,600	30,000	45,600	30,000	45,600	30,000	45,600	30,000	45,600	30,000	45,600	
Balls・Mill Liners	$/t	0.14			30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	
Bets for B. C. and Filters	$/t	0.12			30,000	3,600	30,000	3,600	30,000	3,600	30,000	3,600	30,000	3,600	30,000	3,600	30,000	3,600	30,000	3,600	30,000	3,600	30,000	3,600	30,000	3,600	
Fuels	$/t	0.53			30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	30,000	15,900	
Machinary・Electric Parts	$/t	1.56			30,000	46,800	30,000	46,800	30,000	46,800	30,000	46,800	30,000	46,800	30,000	46,800	30,000	46,800	30,000	46,800	30,000	46,800	30,000	46,800	30,000	46,800	
Repair Expense	$/t	0.34			30,000	10,200	30,000	10,200	30,000	10,200	30,000	10,200	30,000	10,200	30,000	10,200	30,000	10,200	30,000	10,200	30,000	10,200	30,000	10,200	30,000	10,200	
Constraction Expense of Tailing Dam				341,000				206,000						206,000						206,000							
Analysis and others	$/t	0.14			30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	30,000	4,200	
Electric Fee	$/kwh	0.08			1,296,000	103,680	1,296,000	103,680	1,296,000	103,680	1,296,000	103,680	1,296,000	103,680	1,296,000	103,680	1,296,000	103,680	1,296,000	103,680	1,296,000	103,680	1,296,000	103,680	1,296,000	103,680	
Service water Expense	$/t	0.18			30,000	5,400	30,000	5,400	30,000	5,400	30,000	5,400	30,000	5,400	30,000	5,400	30,000	5,400	30,000	5,400	30,000	5,400	30,000	5,400	30,000	5,400	
Total				341,000		239,580		445,580		239,580		239,580		445,580		239,580		239,580		445,580		239,580		239,580		239,580	
Dressing Direct Expense Total	$			341,000		261,180		467,180		261,180		261,180		467,180		261,180		261,180		467,180		261,180		261,180		261,180	
per ton of Dressing direct Expense	$/t			11.37		8.71		15.57		8.71		8.71		15.57		8.71		8.71		15.57		8.71		8.71		8.71	
Head Office																											
Personal Expense																											
Stuff	$/m-m	1,200.00	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	
Overheads				1,000		1,000		1,000		1,000		1,000		1,000		1,000		1,000		1,000		1,000		1,000		1,000	
Transportation Expense				3,000		43,000		3,000		3,000		3,000		3,000		43,000		3,000		3,000		3,000		3,000		3,000	
Customes on Machinary(only investment)				126,950												42,975											
Total				138,150		51,200		11,200		11,200		11,200		11,200		94,175		11,200		11,200		11,200		11,200		11,200	
Grand Total	$			479,150		1,011,125		1,177,125		971,125		971,125		1,177,125		1,054,100		971,125		1,177,125		971,125		971,125		971,125	
per ton of Direct Expense	$/t					33.70		39.24		32.37		32.37		39.24		35.14		32.37		39.24		32.37		32.37		32.37	

Table 38 Direct Operating Expense

Profit & Loss Statement·Funds-Sources and Application·Internal Financial Rate of Return

Profit & Loss Statement

			Unit	-1	1	2	3	4	5	6	7	8	9	10	11		Total
Renenue			'000US$		1,508	1,508	1,508	1,508	1,508	1,508	1,508	1,508	1,508	1,508	1,508	553	17,139
Expense																	
Direct Operat-	Mining		'000US$	0	699	699	699	699	699	699	699	699	699	699	699		7,686
ting Expense	Dressing		'000US$	341	261	467	261	261	467	261	261	467	261	261	261		3,832
	Head Office		'000US$	138	51	11	11	11	11	94	11	11	11	11	11		384
	Total		'000US$	479	1,011	1,177	971	971	1,177	1,054	971	1,177	971	971	971		11,902
	Royality Revenuex10%	10 %	'000US$	0	151	151	151	151	151	151	151	151	151	151	151		1,659
	Agent Comis.Conc Qualityx5$	5 $	'000US$	0	19	19	19	19	19	19	19	19	19	19	19		214
	Depreciation		'000US$	0	425	429	429	429	429	292	292	292	292	133	37		3,482
	Interest	6 %	'000US$		178	190	191	181	170	170	195	185	186	176	165	136	2,123
	Total		'000US$	479	1,784	1,966	1,762	1,752	1,947	1,687	1,629	1,825	1,620	1,450	1,344	136	19,381
Profit before Tax			'000US$	-479	-276	-459	-254	-244	-439	-179	-121	-317	-112	58	164	416	-2,242
Tax	Corporate Profit<0	0%	'000US$	0	0	0	0	0	0	0	0	0	0	22	64	165	251
	Income Tax Profit<1,500/450*000$	15%															0
	Profit<3,000/450*000$	25%															0
	Profit<4,500/450*000$	35%															0
	Profit>4,500/450*000$	40%															0
	Total			0	0	0	0	0	0	0	0	0	0	22	64	165	251
Profit after Tax			'000US$	-479	-276	-459	-254	-244	-439	-179	-121	-317	-112	36	100	251	-2,492

Funds-Sources and Application

			Unit		1	2	3	4	5	6	7	8	9	10	11		Total
Renenue	Profit after Tax		'000US$	-479	-276	-459	-254	-244	-439	-179	-121	-317	-112	36	100	251	-2,492
	Depreciation		'000US$	0	425	429	429	429	429	292	292	292	292	133	37		3,482
	Allowance for Reinvestment		'000US$														0
	Equity of Capital		'000US$	1,143													1,143
	Long-term Borrowings		'000US$	2,668													2,668
	Short-term Borrowings		'000US$	293	198	29		148	343	750	162	358	153	164			2,598
	Total		'000US$	3,625	347	-0	175	334	334	863	333	334	333	333	137	251	7,399
Expence	Investment		'000US$	3,465						530							3,995
	Interest during construction		'000US$	160													160
	Working Capital		'000US$		347										-347		0
	Repayment of long-term Borrowings		'000US$					334	334	334	334	334	334	334	334		2,668
	Repayment of short-term Borrowings		'000US$				175								150	2,273	2,598
	Total		'000US$	3,625	347	0	175	334	334	864	334	334	334	334	137	2,273	9,421
Cash flow surplus			'000US$	0	0	-0	0	0	0	-0	-0	0	-0	-0	0	-2,022	-2,022
Total Cash flow surplus			'000US$	0	1	0	1	1	1	1	1	1	0	-0	-0	-2,022	-2,016
Cashflow			'000US$	-3,944	-20	160	366	366	160	-247	366	160	366	345	649	388	-882

Cashflow=Cash flow surplus-Equity Capital-Borrowings+Repayment+Interest

DCFIRR= -3.0% NPV(-4%)= 358

NPV(-3%)= -15

Table 39 Profit & Loss Statement·Funds—Sources and Application·Internal Financial Rate of Return

Chapter 9. Economic Valuation

9.1 Shift from Financial Cost to Economic Cost

The financial analysis attempts to appraise an investment project from the private investor's standpoint. It lays emphasis on whether the project will generate a due reward and seeks for conditions for the profit maximization. On the other hand, the economic analysis is to evaluate project feasibility from the viewpoint of national economy, in search of the most efficient utilization of public resources as the evaluation criterion.

The economic analysis, therefore, requires adjustment of transfer costs employed in the financial analysis, calculation of opportunity cost, modification of exchange rates to be applied, adjustment of cost of public goods, etc., as detailed in the following paragraphs.

9.1.1 Adjustment of transfer cost

For computation of project economic cost, transfer costs such as taxes included in financial cost have to be adjusted or deducted. In this study, the customs duty imposed on imported goods, the commodity tax included in local commodity prices, the corporate income tax imposed on operating profit, and the royalty levied on sales revenue are deducted as transfer items. The other transfer costs are left unadjusted as they are considered negligible.

For the customs duty adjustment, 15% on general goods and 7.5% on mining machinery, etc. are deducted from the financial costs of imports. As for local goods, the 10% commodity tax is deducted from their financial costs.

In the actual calculation, 95% of material cost included in the (financial) operating cost is assumed to be imports while the remaining 5% is assumed to represent local procurement, from which 15% or 10% are deducted as the tax portions, respectively. 10% is also deducted from the local transportation cost.

As regards the initial investment cost, 30% of the cost of those equipment that requires erection work is assumed to be local costs(for erection and local transportation), from which 10%(commodity tax) is deducted, while 7.5%(customs duty) is deducted from the remaining 70%(imported mining machinery).

9.1.2 Estimation of opportunity cost of labor

This study assumes that, as for demand and supply of skilled labor(staffs) in Mongolia, the market mechanism is relatively well functioning under nearly perfect competition and, therefore, labor cost is determined at a level reflecting the actual situation of the Mongolian economy, free from market distortions. Based on the assumption, opportunity cost of skilled labor is determined by the actual market rate, which is same as the labor cost in the financial analysis.

As for opportunity cost of unskilled labor, the question is not as simple as skilled labor. In Mongolia, as well as in Choybalsan, the unemployment rate is very high. The labor market is in a state of excess labor economy. This implies that opportunity cost of unskilled labor stays at a rate considerably inferior to the actual market rate of wages applied as the financial cost.

According to the public employment security office at Choybalsan, about 3,900 out of the city's total population of 40,000 are unemployed and 70% of the unemployed is youths under 32. In this study, therefore, opportunity cost(economic cost) of unskilled labor(workers other than staffs) is assumed to be a half of the financial labor costs.

9.1.3 Modification of exchange rate

It is a general practice to observe the national trade balance, changes in the wholesale price level, etc. to find out a real(shadow) exchange rate to be applied to the economic valuation of a project. In the relationship between the past Mongolian trade/price data and the official exchange rates, however, regularities are not necessarily recognizable, probably because the nation is still in the process of transition to a market economy since 1989.

Owing to the dear interest policy persued in Mongolia to depress the hyper-inflation, the Togrog seems to have been over-estimated. But inflation rate has declined from 325.5% in 1992 to 66.3% in 1994 and recently, it has settled down to a level of 2% per month. Besides, most of the project equipment and machinery which constitutes a major part of the project costs are imports which are settled in US Dollars. Considering these factors, no modification is made in this study on the exchange rate used for the financial analysis.

9.1.4 Cost of public goods

Since the cost of laying the power transmission line is the public expenditure, only 50% of the total cost, which must be borne by a user, is included in the financial costs of the project. For economic analysis, however, the full cost must be counted in.

9.2 Internal Economic Rate of Return(IERR)

Unlike the financial analysis, the economic analysis focuses on a difference between cost and benefit in terms of economic price, after eliminating the transfer cost such as tax.

All the goods and services related to the project are divided into the foreign currency portion(traded goods) and the local currency portion(non-traded goods), the former's value being measured with a shadow exchange rate. (In this study, however, the same official exchange rate is applied, as explained.)

In accordance with these procedures, profit and loss projection, fund sources and application and IERR exhibited in Table 40 have been calculated, on the basis of the operating expenses, divisional initial investment cost and depreciation expenses exhibited in Table 41 and 42. The discount rate at which the present value of net benefit(benefit less cost) of this project comes to zero is 8.3%. In order to judge the project desirability from the national viewpoint, IERR is used as a criterion. The World Bank is said to consider an investment project feasible when IERR is not less than 12%, whilst 8% is the lower limit for the USAID and 10% for the Asian Development Bank. Whether or not this project is feasible from the viewpoint of national economy is a delicate question to answer.

Profit & Loss Statement·Funds-Sources and Application·Internal Economical Rate of Return

Profit & Loss Statement

		Unit	-1	1	2	3	4	5	6	7	8	9	10	11		Total
Renenue		'000US$		1,508	1,508	1,508	1,508	1,508	1,508	1,508	1,508	1,508	1,508	1,508	622	17,208
Expense																
Direct Operat- Mining		'000US$	0	588	588	588	588	588	588	588	588	588	588	588		6,472
ting Expense Dressing		'000US$	310	226	413	226	226	413	226	226	413	226	226	226		3,359
Head Office		'000US$	11	51	11	11	11	11	51	11	11	11	11	11		210
Total		'000US$	321	865	1,013	825	825	1,013	865	825	1,013	825	825	825		10,041
Royality Revenuex10%	0 %	'000US$	0	0	0	0	0	0	0	0	0	0	0	0		0
Agent Comis.Conc Qualityx5$	5 $	'000US$	0	19	19	19	19	19	19	19	19	19	19	19		214
Depreciation		'000US$	0	432	437	437	437	437	301	301	301	301	146	50		3,580
Interest	6 %	'000US$		175	170	168	168	147	126	105	84	63	42	21	-0	1,268
Total		'000US$	321	1,492	1,639	1,449	1,449	1,616	1,312	1,251	1,417	1,209	1,033	916	-0	15,104
Profit before Tax		'000US$	-321	16	-131	59	59	-108	196	257	91	299	475	592	622	2,105
Tax Corporate Profit<0	0%	'000US$	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Income Tax Profit<1,500/450*000$	15%															0
Profit<3,000/450*000$	25%															0
Profit<4,500/450*000$	35%															0
Profit>4,500/450*000$	40%															0
Total			0	0	0	0	0	0	0	0	0	0	0	0	0	0
Profit after Tax		'000US$	-321	16	-131	59	59	-108	196	257	91	299	475	592	622	2,105

Funds-Sources and Application

		Unit		1	2	3	4	5	6	7	8	9	10	11		Total
Renenue	Profit after Tax	'000US$	-321	16	-131	59	59	-108	196	257	91	299	475	592	622	2,105
	Depreciation	'000US$	0	432	437	437	437	437	301	301	301	301	146	50		3,580
	Allowance for Reinvestment	'000US$														0
	Equity of Capital	'000US$	1,198													1,198
	Long-term Borrowings	'000US$	2,796													2,796
	Short-term Borrowings	'000US$	125													125
	Total	'000US$	3,799	448	306	495	495	329	497	558	392	600	621	642	622	9,804
Expence	Investment	'000US$	3,632						530							4,162
	Interest during construction	'000US$	168													168
	Working Capital	'000US$		363										-363		0
	Repayment of long-term Borrowings	'000US$					350	350	350	350	350	350	350	350		2,796
	Repayment of short-term Borrowings	'000US$		85	40											125
	Total	'000US$	3,799	448	40	0	350	350	880	350	350	350	350	-14	0	7,251
Cash flow surplus		'000US$	-0	-0	266	495	146	-21	-382	209	42	251	272	656	622	2,553
Total Cash flow surplus		'000US$	-0	-1	265	760	906	885	503	711	754	1,004	1,276	1,932	2,553	11,547
Cashflow		'000US$	-3,952	260	476	663	663	476	93	663	476	663	663	1,026	622	2,791

Cashflow=Cash flow surplus-Equity Capital-Borrowings+Repayment+Interest

DCFIRR= 8.3% NPV(8%)= 57

NPV(9%)= -148

Table 40 Profit & Loss Statement·Funds–Sources and Application·Internal Economical Rate of Return

Direct Operating Expense(Economical)

Unit :US$

Article	Unit	Price	-1st year Quantity	-1st year Amount	1st year Quantity	1st year Amount	2nd year Quantity	2nd year Amount	3rd year Quantity	3rd year Amount	4th year Quantity	4th year Amount	5th year Quantity	5th year Amount	6th year Quantity	6th year Amount	7th year Quantity	7th year Amount	8th year Quantity	8th year Amount	9th year Quantity	9th year Amount	10th year Quantity	10th year Amount	11th year Quantity	11th year Amount	Note
Mining division																											
Personal Expense																											
Staff	$/m·a	1,200.00			10	12,000	10	12,000	10	12,000	10	12,000	10	12,000	10	12,000	10	12,000	10	12,000	10	12,000	10	12,000	10	12,000	
Exterior	$/m·a	466.67			34	15,867	34	15,867	34	15,867	34	15,867	34	15,867	34	15,867	34	15,867	34	15,867	34	15,867	34	15,867	34	15,867	
Interior	$/m·a	560.00			26	14,560	26	14,560	26	14,560	26	14,560	26	14,560	26	14,560	26	14,560	26	14,560	26	14,560	26	14,560	26	14,560	
Total					70	42,427		42,427	70	42,427		42,427	70	42,427		42,427	70	42,427		42,427		42,427		42,427		42,427	
Exploitation Expense																											
Bits·Rods	$/t	2.94			30,000	88,113	30,000	88,113	30,000	88,113	30,000	88,113	30,000	88,113	30,000	88,113	30,000	88,113	30,000	88,113	30,000	88,113	30,000	88,113	30,000	88,113	
Explosives	$/t	0.46			30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	
Detonaters	$/t	0.01			30,000	346	30,000	346	30,000	346	30,000	346	30,000	346	30,000	346	30,000	346	30,000	346	30,000	346	30,000	346	30,000	346	
Fuels	$/t	0.13			30,000	3,922	30,000	3,922	30,000	3,922	30,000	3,922	30,000	3,922	30,000	3,922	30,000	3,922	30,000	3,922	30,000	3,922	30,000	3,922	30,000	3,922	
Machinary·Electric Parts	$/t	7.84			30,000	235,316	30,000	235,316	30,000	235,316	30,000	235,316	30,000	235,316	30,000	235,316	30,000	235,316	30,000	235,316	30,000	235,316	30,000	235,316	30,000	235,316	
Others(20% of abovementioned)	%	20				21,248		21,248		21,248		21,248		21,248		21,248		21,248		21,248		21,248		21,248		21,248	
Total						362,802		362,802		362,802		362,802		362,802		362,802		362,802		362,802		362,802		362,802		362,802	
Development Expense																											
Bits·Rods	$/m	120.53			240	28,928	240	28,928	240	28,928	240	28,928	240	28,928	240	28,928	240	28,928	240	28,928	240	28,928	240	28,928	240	28,928	
Explosives	$/m	18.83			240	4,518	240	4,518	240	4,518	240	4,518	240	4,518	240	4,518	240	4,518	240	4,518	240	4,518	240	4,518	240	4,518	
Detonaters	$/m	3.22			240	774	240	774	240	774	240	774	240	774	240	774	240	774	240	774	240	774	240	774	240	774	
Fuels	$/m	5.49			240	1,318	240	1,318	240	1,318	240	1,318	240	1,318	240	1,318	240	1,318	240	1,318	240	1,318	240	1,318	240	1,318	
Machinary·Electric Parts	$/m	294.58			240	70,699	240	70,699	240	70,699	240	70,699	240	70,699	240	70,699	240	70,699	240	70,699	240	70,699	240	70,699	240	70,699	
Others(20% of abovementioned)	%	20				7,108		7,108		7,108		7,108		7,108		7,108		7,108		7,108		7,108		7,108		7,108	
Total						113,345		113,345		113,345		113,345		113,345		113,345		113,345		113,345		113,345		113,345		113,345	
Electric Fee	$/kwh	0.07			960,000	69,818	960,000	69,818	960,000	69,818	960,000	69,818	960,000	69,818	960,000	69,818	960,000	69,818	960,000	69,818	960,000	69,818	960,000	69,818	960,000	69,818	
Mining direct Expense Total	$					588,392		588,392		588,392		588,392		588,392		588,392		588,392		588,392		588,392		588,392		588,392	
per ton of Mining direct Expense	$/t					19.61		19.61		19.61		19.61		19.61		19.61		19.61		19.61		19.61		19.61		19.61	
Mineral Dressing Division																											
Personal Expense																											
Staff	$/m·a	1,200.00			4	4,800	4	4,800	4	4,800	4	4,800	4	4,800	4	4,800	4	4,800	4	4,800	4	4,800	4	4,800	4	4,800	
Exterior	$/m·a	466.67			18	8,400	18	8,400	18	8,400	18	8,400	18	8,400	18	8,400	18	8,400	18	8,400	18	8,400	18	8,400	18	8,400	
Total						13,200		13,200		13,200		13,200		13,200		13,200		13,200		13,200		13,200		13,200		13,200	
Material Expense and others																											
Chemical Reagent	$/t	1.32			30,000	39,742	30,000	39,742	30,000	39,742	30,000	39,742	30,000	39,742	30,000	39,742	30,000	39,742	30,000	39,742	30,000	39,742	30,000	39,742	30,000	39,742	
Balls·Mill Liners	$/t	0.12			30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	
Nets for B.C. and Filters	$/t	0.10			30,000	3,138	30,000	3,138	30,000	3,138	30,000	3,138	30,000	3,138	30,000	3,138	30,000	3,138	30,000	3,138	30,000	3,138	30,000	3,139	30,000	3,138	
Fuels	$/t	0.46			30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	30,000	13,858	
Machinary·Electric Parts	$/t	1.36			30,000	40,788	30,000	40,788	30,000	40,788	30,000	40,788	30,000	40,788	30,000	40,788	30,000	40,788	30,000	40,788	30,000	40,788	30,000	40,788	30,000	40,788	
Repair Expense	$/t	0.30			30,000	8,890	30,000	8,890	30,000	8,890	30,000	8,890	30,000	8,890	30,000	8,890	30,000	8,890	30,000	8,890	30,000	8,890	30,000	8,890	30,000	8,890	
Construction Expense of Tailing Dam				310,000				187,273						187,273						187,273							
Analysis and others	$/t	0.12			30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	30,000	3,660	
Electric Fee	$/kwh	0.07			1,296,000	94,255	1,296,000	94,255	1,296,000	94,255	1,296,000	94,255	1,296,000	94,255	1,296,000	94,255	1,296,000	94,255	1,296,000	94,255	1,296,000	94,255	1,296,000	94,255	1,296,000	94,255	
Service water Expense	$/t	0.16			30,000	4,909	30,000	4,909	30,000	4,909	30,000	4,909	30,000	4,909	30,000	4,909	30,000	4,909	30,000	4,909	30,000	4,909	30,000	4,909	30,000	4,909	
Total				310,000		212,900		400,173		212,900		212,900		400,173		212,900		212,900		400,173		212,900		212,900		212,900	
Dressing Direct Expense Total	$			310,000		226,100		413,373		226,100		226,100		413,373		226,100		226,100		413,373		226,100		226,100		226,100	
per ton of Dressing direct Expense	$/t			10.33		7.54		13.78		7.54		7.54		13.78		7.54		7.54		13.78		7.54		7.54		7.54	
Head Office																											
Personal Expense																											
Staff	$/m·a	1,200.00	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	6	7,200	
Overheads				909		909		909		909		909		909		909		909		909		909		909		909	
Transportation Expense				2,727		42,727		2,727		2,727		2,727		2,727		42,727		2,727		2,727		2,727		2,727		2,727	
Customs on Machinary(only investment)																											
Total				10,836		50,836		10,836		10,836		10,836		10,836		50,836		10,836		10,836		10,836		10,836		10,836	
Grand Total	$			320,836		865,328		1,012,601		825,328		825,328		1,012,601		865,328		825,328		1,012,601		825,328		825,328		825,328	
per ton of Direct Expense	$/t					28.84		33.75		27.51		27.51		33.75		28.84		27.51		33.75		27.51		27.51		27.51	

Table 41 Direct Operating Expense(Economical)

Investment & Depreciation of each division(Economical)

Unit: US$

Article	Div No	Investment	Invest time	Life	Residual Book Value	Invest Rate	-1st year Book Value	-1st year Depreciation	1st year Book Value	1st year Depreciation	2nd year Book Value	2nd year Depreciation	3rd year Book Value	3rd year Depreciation	4th year Book Value	4th year Depreciation	5th year Book Value	5th year Depreciation	6th year Book Value	6th year Depreciation	7th year Book Value	7th year Depreciation	8th year Book Value	8th year Depreciation	9th year Book Value	9th year Depreciation	10th year Book Value	10th year Depreciation	11th year Book Value	11th year Depreciation	Note
Mining division																															
Jumbo for Mining 1 set	1	310,000		5	31,000	0.180			310,000	55,800	254,200	55,800	198,400	55,800	142,600	55,800	86,800	55,800	31,000		31,000		31,000		31,000		31,000		31,000		
Batcher Plant 1 set	1	340,455		5	34,045	0.180			340,455	61,282	279,173	61,282	217,891	61,282	156,609	61,282	95,327	61,282	34,045		34,045		34,045		34,045		34,045		34,045		
Equipment of Shotcreting 1 set	1	350,000		5	35,000	0.180			350,000	63,000	287,000	63,000	224,000	63,000	161,000	63,000	98,000	63,000	35,000		35,000		35,000		35,000		35,000		35,000		
Mixer Truck(1.8m3) 1 set	1	40,000		5	4,000	0.180			40,000	7,200	32,800	7,200	25,600	7,200	18,400	7,200	11,200	7,200	4,000		4,000		4,000		4,000		4,000		4,000		
Load Hauld Dump(3.8m3) 1 set	1	350,000		5	35,000	0.180													350,000	63,000	287,000	63,000	224,000	63,000	161,000	63,000	98,000	63,000	35,000		
Load Hauld Dump(1.7m3) 1 set	1	180,000		5	18,000	0.180													180,000	32,400	147,600	32,400	115,200	32,400	82,800	32,400	50,400	32,400	18,000		
Track(1t) x 2 pieces 1 set	1	32,000		5	3,200	0.180			32,000	5,760	26,240	5,760	20,480	5,760	14,720	5,760	8,960	5,760	3,200		3,200		3,200		3,200		3,200		3,200		
Wagon(350kg) x 2 pieces 1 set	1	60,000		5	6,000	0.180			60,000	10,800	49,200	10,800	38,400	10,800	27,600	10,800	16,800	10,800	6,000		6,000		6,000		6,000		6,000		6,000		
Drainage Pump 1 set	1	40,000		8	4,000	0.113					40,000	4,500	35,500	4,500	31,000	4,500	26,500	4,500	22,000	4,500	17,500	4,500	13,000	4,500	8,500	4,500	4,000		4,000		
Total of Mining division		1,702,455			170,245				1,132,455	203,842	968,613	208,342	760,271	208,342	551,929	208,342	343,587	208,342	665,245	99,900	565,345	99,900	465,445	99,900	365,545	99,900	265,645	95,400	170,245		
Mineral Dressing division																															
Crushing (Facilities Thickner.Ore Strage)	2	324,891		9	32,489	0.100			324,891	32,489	292,402	32,489	259,913	32,489	227,424	32,489	194,935	32,489	162,445	32,489	129,956	32,489	97,467	32,489	64,978	32,489	32,489		32,489		
Milling	2	252,909		9	25,291	0.100			252,909	25,291	227,618	25,291	202,327	25,291	177,036	25,291	151,745	25,291	126,455	25,291	101,164	25,291	75,873	25,291	50,582	25,291	25,291		25,291		
Floatation	2	238,318		9	23,832	0.100			238,318	23,832	214,486	23,832	190,655	23,832	166,823	23,832	142,991	23,832	119,159	23,832	95,327	23,832	71,496	23,832	47,664	23,832	23,832		23,832		
Concentrate (Pb.Zn Thickner.Filter.Drier)	2	160,500		9	16,050	0.100			160,500	16,050	144,450	16,050	128,400	16,050	112,350	16,050	96,300	16,050	80,250	16,050	64,200	16,050	48,150	16,050	32,100	16,050	16,050		16,050		
Tailing (Thickner.Facilities)	2	107,000		9	10,700	0.100			107,000	10,700	96,300	10,700	85,600	10,700	74,900	10,700	64,200	10,700	53,500	10,700	42,800	10,700	32,100	10,700	21,400	10,700	10,700		10,700		
Pumping Facility (Recycle Dragenage)	2	11,673		9	1,167	0.100			11,673	1,167	10,505	1,167	9,338	1,167	8,171	1,167	7,004	1,167	5,836	1,167	4,669	1,167	3,502	1,167	2,334	1,167	1,167		1,167		
Concentrate Loading Place	2	17,509		9	1,751	0.100			17,509	1,751	15,758	1,751	14,007	1,751	12,256	1,751	10,505	1,751	8,755	1,751	7,004	1,751	5,253	1,751	3,502	1,751	1,751		1,751		
Laboratory	2	59,336		9	5,934	0.100			59,336	5,934	53,403	5,934	47,469	5,934	41,536	5,934	35,602	5,934	29,668	5,934	23,735	5,934	17,801	5,934	11,868	5,934	5,934		5,934		
Building・Office (721 m²) 1set	2	176,064		9	17,606	0.100			176,064	17,606	158,457	17,606	140,851	17,606	123,244	17,606	105,638	17,606	88,032	17,606	70,425	17,606	52,819	17,606	35,212	17,606	17,606		17,606		
Total of Mineral Dressing		1,348,200			134,820				1,348,200	134,820	1,213,380	134,820	1,078,560	134,820	943,740	134,820	808,920	134,820	674,100	134,820	539,280	134,820	404,460	134,820	269,640	134,820	134,820		134,820		
Common division																															
Burden of Construction Cost (Electric Power Line)	3	485,455		15	48,545	0.060			485,455	29,127	456,327	29,127	427,200	29,127	398,073	29,127	368,945	29,127	339,818	29,127	310,691	29,127	281,563	29,127	252,436	29,127	223,309	29,127	194,182	29,127	
Office・Dormitory (680 m²) 1 set	3	160,500		9	16,050	0.100			160,500	16,050	144,450	16,050	128,400	16,050	112,350	16,050	96,300	16,050	80,250	16,050	64,200	16,050	48,150	16,050	32,100	16,050	16,050		16,050		
Microbus	3	60,000		5	6,000	0.180			60,000	10,800	49,200	10,800	38,400	10,800	27,600	10,800	16,800	10,800	6,000		6,000		6,000		6,000		6,000		6,000		
Truck for Transportation of Concentrate(11t)	3	90,000		5	9,000	0.180			90,000	16,200	73,800	16,200	57,600	16,200	41,400	16,200	25,200	16,200	9,000		9,000		9,000		9,000		9,000		9,000		
Receiving Facility of Electric Power	3	355,000		15	35,500	0.060			355,000	21,300	333,700	21,300	312,400	21,300	291,100	21,300	269,800	21,300	248,500	21,300	227,200	21,300	205,900	21,300	184,600	21,300	163,300	21,300	142,000	21,300	
Total of Common division		1,150,955			140,873				1,150,955	93,477	1,057,477	93,477	964,000	93,477	870,523	93,477	777,045	93,477	683,568	66,477	617,091	66,477	550,613	66,477	484,136	66,477	417,659	50,427	367,232	50,427	
	0	4,201,609			305,065				3,631,609	432,139	3,239,470	436,639	2,802,831	436,639	2,366,192	436,639	1,929,552	436,639	2,022,913	301,197	1,721,716	301,197	1,420,518	301,197	1,119,321	301,197	818,124	145,827	672,237	50,427	Depreciation Balance
per ton of Crude Ore										14.40		14.55		14.55		14.55		14.55		10.04		10.04		10.04		10.04		4.86	T-Depreciat	3,579,749	621,859

Table 42 Investment & Depreciation of each division(Economical)