# REPORT ON
# THE PRE-FEASIBILITY STUDY FOR THE DEVELOPMENT
# IN
# TSAV AREA, MONGOLIA

## FINAL

MARCH, 1996



JAPAN INTERNATIONAL COOPERATION AGENCY
METAL MINING AGENCY OF JAPAN

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Preface:

In response to the Mongolian Government's request, the Japanese Government decided to execute geological survey for the purpose of local area development in Tsav, located in the eastern part of Dolnod Prefecture, Mongolia, and entrusted the task to Japan International Cooperation Agency (JICA). JICA, in turn, commissioned the Metal Mining Agency of Japan (MMAJ) to undertake the survey, as it pertains to geology and mineral exploration.

In 1995, the forth year of the survey (which started in 1992), MMAJ sent to the survey area a team of fifteen members from July 18 to September 16.

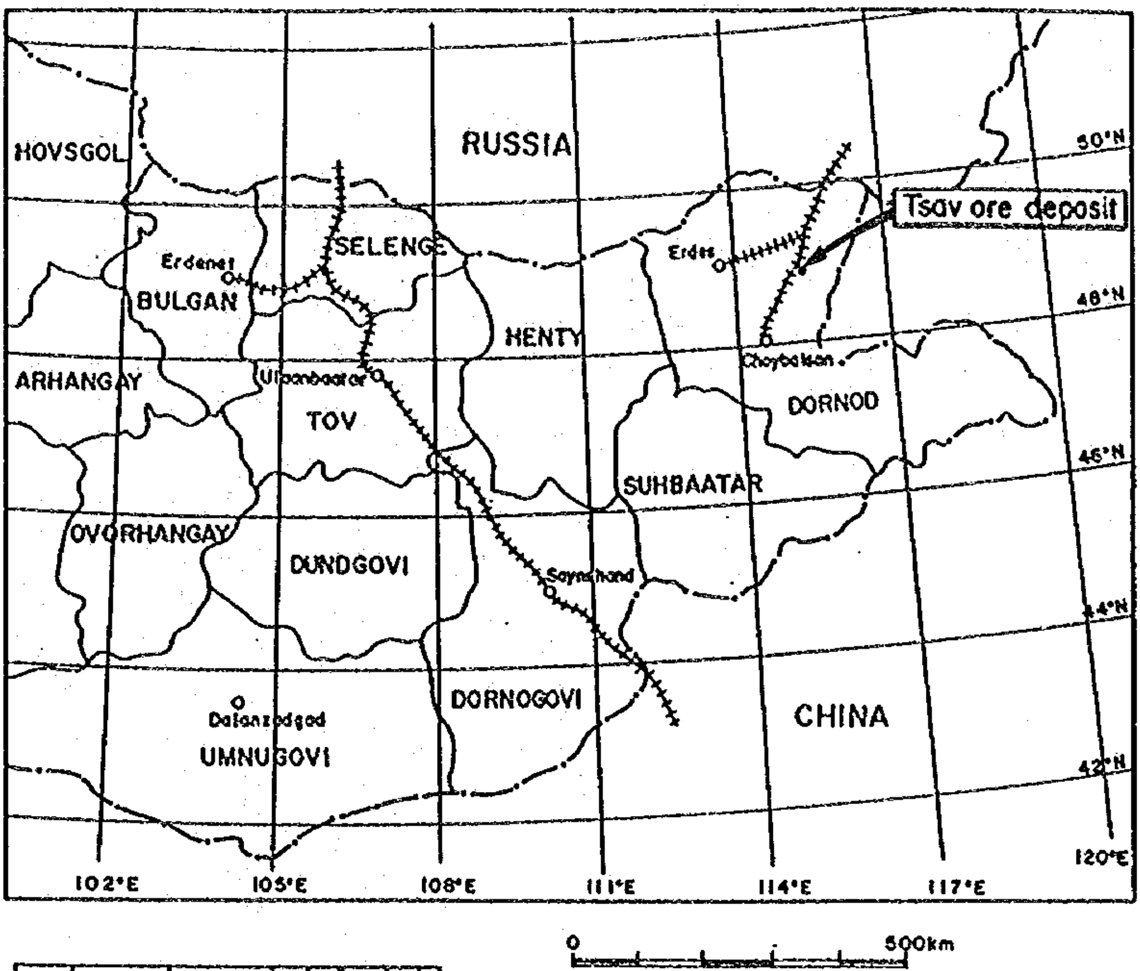
The survey of the designated area was completed as scheduled, under the cooperation rendered by Mongolian Government agencies concerned. This Report summarized the fourth year's survey and its results, and is intended as the final report of the 4-year survey, as well.

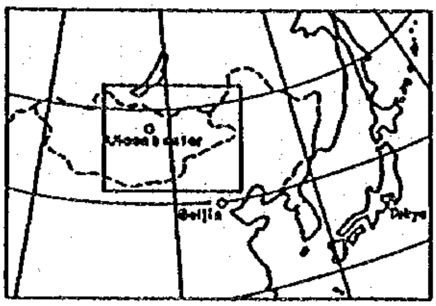
We, on behalf of JICA and MMAJ, should like to take this opportunity to express our cordial appreciation to the Mongolian government organizations concerned and their personnel involved, and also to the Japanese Ministry of Foreign Affairs, the Ministry of International Trade & Industry and the Japanese Embassy in Ulaanbaatar, for their valuable support.

February 1996

Kimio FUJITA
President
Japan International Cooperation Agency

Shozaburo KIYOTAKI
President
Metal Mining Agency of Japan



Location Tsav Deposit

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## Summary

This report summarizes results of the preliminary feasibility study for development of the Tsav deposits, based upon findings of the site survey conducted from 1992 to 95 and of the final survey in 1995.

Conclusions of the study may be summed up as follows:

### (1) Ore reserves

The ore reserves of Tsav deposits totaling the proved, probable and possible ores amounts to 1,544,627 tons, grading Au 1.5g/t*, Ag 263g/t, Pb 6.84%, Zn 4.01% and Cu 0.23%*. [ * The grade of Au and Cu are averages of proved and probable ore reserves.]

### (2) Minable crude ore

Minable crude ore comes to 322,464 tons totaling the proved, probable and possible ores, grading Au 1.2g/t*, Ag 161g/t, Pb 6.37%, Zn 2.94% and Cu 0.22%*. [ * The Au and Cu grades are averages of proved and probable ores.]

The average mining recovery and the average dilution are 18% and 21% respectively. The low rate of the average mining recovery is attributable to the fact that the group of ore deposits at Tsav is composed of fine veins occurring in deep part whereby the immediate mining targets had to be limited to the No.4 vein above 630m(above sea level) and to the other orebodies up to 100m from the surface.

### (3) Mineral dressing process

Of the two alternative processes, the straight differential flotation and bulk differential flotation, the latter process has been chosen, because the Tsav zinc ore partially contains oxidic copper ore, which makes it difficult to depress zinc in the process of separating lead and zinc.

### (4) Type of concentrate

Study was made as to whether to produce lead concentrate and zinc concentrate, or only bulk concentrate. It has been concluded that it would be more favorable to ship lead and zinc concentrates separately.

(5) Overall project appraisal

1) The internal financial rate of return(IFRR) calculated on the project assumptions is (−)3.0%. At this rate, it would be very difficult for a private investor to realize the project, relying on borrowings.

2) In contrast, the internal economic rate of return(IERR) comes to 8.3%, a delicate rate for judging whether to go ahead. Since the study places emphasis on mine profitability, it does not take into consideration such factors pertinent to the economic valuation as social benefits and indirect effects/impacts that the mine development, construction of the power supply facilities and road improvement would bring forth to the regional development and economy ( local civil constructors, ironworks, repair shops, transporters, stock raisers, etc.). If such effects and benefits are counted in, the IERR will certainly enhance to a higher level.

3) After all, the project feasibility depends upon whether or not the financial valuation can be improved by effective policy measures such as grant aid, soft loans and tax benefits. It leads to a conclusion that, unless such conditions are met, desirability of the mine investment remains low.

(6) Future vision

The pre−feasibility study indicates that the ore reserves so far acquired and the deposit size are not large enough to allow the mine revenue to expand by increased investment.

Around the Tsav area in Dornod, however, there are a number of promising ore deposits including Ulaan and Bajan Uul. Once the Tsav deposits are developed, it would facilitate/activate opening of these deposits as satellite mines, which will possibly permit an expansion of ore treatment at Tsav. The Tsav mine could play a leading role in the exploration, mining and operation technology aspects. The Tsav deposits, if developed, could become a model mine for underground operation since there is no underground mine in Mongolia. Besides, development of the Tsav deposits does not involve an exorbitant capital outlay.

In the light of these considerations, exploration of the other deposits in the region is highly recommendable. Using ore samples obtained through the exploration, possibility of mixed ore treatment could be studied. The decision on the final feasibility study should preferably be taken on the basis of exploration and mineral dressing test findings of the other deposits in the region.

# Ⅰ. General Remarks

# Part Ⅰ. General Remarks

## Chapter 1. Introduction

### 1.1 Purpose of the survey

The purpose of the survey is to carry out tunneling and diamond drilling in the upper parts of level 180m (630m above sea level) of the Tsav deposits to explore the No.4 vein where the investigation is most advanced, and to draw up a mine development plan through the exact recognition of the characteristics of the mentioned vein, as well as through mineral processing test to be conducted on the collected samples. The tunneling comprises the driving a inclined shaft from the surface and a drifting at −60m level (750m above sea level). The diamond drilling was carried out inside the tunnel and on the surface.

The survey also aims to promote technology transfer pertaining to excavation, mining and machinery control by means of those works and training of counterparts in Japan.

### 1.2 Scope of the survey

Mongolia possesses a high potential for the production of virtually various types of mineral resources. Presently, copper, molybdenum, tin, and flourite are produced there. By developing its mineral resources, we anticipate that Mongolia will also become an important supplier to the world's future increased demands in copper, lead, and zinc, etc.

Reform of Mongolia's economic structure has progressed steadily since 1987. However, in recent years, the scale of its resource development was decreased considerably due to reduced technological cooperation from the former Soviet Union and other Eastern European Countries. This scaling down of Mongolia's mineral industry, an industry vital to the country's acquisition of foreign currency, has caused the industry to enter a period of stagnation.

Since the integrated studies in Tsav, an area in the northeastern section of Mongolia's Dolnod prefecture, could indicate a high potential for a commercially feasible mine, the Mongolian government has expressed a strong interest in the development of the area and in February of 1991 has made a request to Japanese government a technical cooperation for the survey to develop the polymetallic deposits (Tsav,Ulaan) in the eastern area.

The development of a new mine will contribute significantly to have the economic growth of Mongolia while at the same time establishing an additional stable supply of metal ore resources for the global market will benefit to Japanese economy. For these reasons, new budgetary measures for the regional development planning studies have been taken and implementation system has been established accordingly.

Under these circumstances, Japanese government has dispatched a preliminary study team and held a conference with the Mongolian government regarding the studies to be conducted. And, on July 30, 1992, scope of work regarding the studies to be conducted were established. Under these formally established scope of work, a studies team consisting of seven members was dispatched to the survey area between August 31 and September 19 and the framing of the plans for the study project was designated. Under this plan, from July 18 to September 16, 1995, the total 15 engineers were sent to conduct tunneling and diamond drilling.

## Chapter 2. Overview of the survey area

### 2.1 Location and transportation

The Tsav mineral deposits are located in the Somon area of Choybalsan District of Dolnod Prefecture in the eastern Mongolia. It is approximately 120 kilometers northeast of Choybalsan City (refer to the map of survey area). The first years survey, conducted in 1989, encompassed an area of approximately 45 $km^2$ and included the Tsav ore deposits. As indicated on the map, this area is located between the north latitude of 48° 50′ to 49° 00′ and the east longitude of 115° 15′ to 115° 30′. The geographical coordinates at the heart of the Tsav ore deposits which includes an area of approximately 12 $km^2$ is at the north latitude of 48° 55′ 40″ and the east longitude of 115° 20′ 33″. Approximately 5 kms west of the Tsav ore deposits is the main truck line of the Siberian Railway which runs in a southerly direction from Siberian Railway's Bolsha Station across the border towards Choybalsan to Elentsav (Sorobefusuku), and then on to Choybalsan (Bayan Toumen). The distance to the neighboring Habilka Station is approximately 17 kms.

An unpaved road which is passable throughout the year runs between the Tsav ore deposits and Choybalsan. The distance takes approximately three hours by car. In July of 1992, a new customs house (the name in Chinese reads 35 Kokuzantou Customs) was established at the border of the autonomous Chinese Mongolian State located approximately 50 kms east southeast of the Tsav ore deposits. For the three months form July the customs house is opened for business during the first half of each month. From the Tsav ore deposits the customs house can be reached by car overran unpaved road in approximately ninety minutes.

### 2.2 Topography

The topography of Tsav is moderate rolling hills which consist of a gentle hilly terrain alternating with swampy plains. The slope gradient of the Tsav area does not exceed 5 to 10 degrees. It is a moderately hilly terrain with smooth slopes. The highest peak is 825 meters above sea level. The difference of the height between the mountain ridge and valley is 50 to 80 meters.

## 2.3 Vegetation and Climate

Vegetation of the Tsav ore deposit area is typical steppe lands with a variety of grain plants and lack of trees. The nearest forest is located approximately 150 kilometers towards northwest. The climate is typical continental dry environment. The daily and annual changes in temperature and pressure is remarkable. Windy days with little precipitation are prevailing in winter. The average, annual snowfall does not exceed 80 to 150 millimeters. The daily temperature changes in spring are severe. Dry air, strong winds, and sand storms are characteristic of that season. Summer is short and mild. The temperature differences between day and night are great. The principal wind direction is northwest with an average speed of three to five meters per second. The maximum wind speed is between 20 to 25 meters per second. The average temperature throughout the year is 0 ℃ . The lowest recorded temperature is −37.5 ℃ (1987). The highest recorded temperature is +37.5 ℃ (1982). According to the Choybalsan meteorological observatory, the average annual rainfall of the region is 244 millimeters. However, according to the Eldes Town (Maludai Mine) observatory, the average annual rainfall is 402 millimeters.

The table bellows shows the average temperature and rainfall on a monthly basis (courtesy of the Eldes meteorological observatory).

| Month | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Temp. (℃) | -20 | -18 | -8 | 0 | +11 | +16 | +18 | +16 | +9 | +1 | -10 | -17 |
| Rainfall (mm) | 3 | 2 | 4 | 11 | 15 | 51 | 91 | 117 | 36 | 3 | 7 | 3 |

Table 1 The average temperature and rainfall by monthly basis
(source, The Eldes meteorogical observatory)

The depth of the frozen ground in winter is between 2.4 to 4.2 meters. By the end of June, the frozen ground is completely thaws. There are no permanent frozen ground in the survey area. There are no continuously flowing springs or watercourses in the vicinity of the Tsav ore deposits.

## 2.4 Geology and ore deposits

The area of Tsav is composed of the groups of upper Proterozoic to lower Paleozoic and upper Mesozoic formations.

The lower geological units, consisting of the metamorphic rocks of upper Proterozoic Salkhit series and granite of lower Paleozoic, are located in the north to northeastern parts of the survey area. The Salkhitit series is composed of phylite, semischist, crystalline schist, gneiss, limestone and quartzite. Thickness of the formation reaches 1,500m. The granite comprises medium to coarse-grained porphyritic biotite granite ~ granodiorite. The rock faces are variable frequently showing banded structure ~ gneissose structure.

The upper geological units are composed of middle to upper Jurassic volcanic rocks (Tsav series) and late Jurassic intrusive rocks. The volcanic rocks of Tsav series is mainly composed of dacitic ~ andesitic lava and pyroclastic rocks accompanied by rhyolite and basalt, are occasionally intercalates tuffaceous conglomerate, sandstone and siltstone. The Tsav series are extends from north to south in the western part of the area and from northwest to southeast in the central to eastern part. The formation tends to be thinner in the eastern part, while, in the western part, it becomes as thick as 700m or more. The intrusive rocks of late Jurassic are mainly composed of granodiorite accompanied by some granite porphyry, syenite porphyry, monzonidiorite porphyry, and dykes of andesite and porphyrite.

The Tsav fractured zone repeats winding in the NW-SE ~ WNW-ESE directions but, as a whole, the zone extends NW-SE. In the Tsav deposit area, the fracture structure extends NW-SE in the northern section whilst, in the southern section, it extends WNE-ESE. The mineralized zone is considered to be formed at a bending point of the regional structure. It is also observed that the regional structure is cut by a NE-SW fracture.

The ore deposits of Tsav are formed in the Tsav fractured zone as fissure filling polymetallic deposits. The principal ore bodies are NO.1, 2, 4, 6 and 8 veins deposited in the continuous faults, while the subordinate veins are NO.1A, 1B, 2A, 4A, 4B, 6A and 8A deposited in the secondary echelon fractures. The veins strike NW-SE and dip steeply ( 60° ~ 85° ) eastward. The principal ore minerals are pyrite, galena and sphalerite associated with a small amount of chalcopyrite, chalcocite, marcasite and, rarely with miargyrite. The gangue minerals are mainly quartz and sericite associated with carbonates and clay minerals. Calcite and rhodochrosite accompanied by rhodonite are observed as carbonates. An oxidation zone is formed down to 30m ~ 40m below the surface where sulphide minerals of lead, zinc and copper partially turn into cerussite, smithonite, malachite and azurite.

## Chapter 3. Outline of the survey in 1995

### 3.1 Purpose of the survey

In continuation from the preceding year's survey, the 1995 survey is constituted of tunneling at −60m level(750m above sea level), diamond drilling from the tunnel toward the No.4 Vein, aimed at clarifying the occurrence of lead, zinc, gold and silver at −60m level, and surface drilling toward −30m level for investigating the continuity of mineralization from the surface to −60m level of the No.4 vein.

Also included in the 1995 survey is mineral processing test for recovery of zinc and lead concentrates via floatation from ore samples collected at the tunnel, thereby selecting the optimum ore dressing process.

Based upon survey findings, studies and test results, economic appraisal of the ore deposits, as well as mine development planning, are to be effected.

### 3.2 The survey in this year

(1)Project Name :Regional development planning studies
(2)Investigation area :Tsav area, Mongolia
(3)Period :1995.07.18～1996.02.23
(4)Contents of Survey :
·Tunneling
Quantity
Horizontal Drift :120.0m (Northward)
Fifth Waste Pit :30.0m
Designed Section :11.88m² (Width 4.0(m) x Height 3.4(m))
Designed Gradient :±1/500
Designed Direction :Ref. Tunnel and Core Boring in 1995
Temporary Construction
Electricity :Setting of control panel, switch board etc.
Water Supply :Plumbing(50A)
Ventilation :FRV Air duct(700m/mφ)
Mapping :Underground mapping(1/200):Chemical analysis

| | |
|---|---|
| ·Diamond Drilling | :14 holes totalling 475m: Core logging Location and length of the drill holes are shown in the '95 Survey plan |
| ·Ore Dressing Test | :Ore sampling in the Tsav deposits: property measurement and ore dressing test Selection of ore dressing flowsheet |
| ·Mine Development Planning | :Economic appraisal of the ore deposits: mine development planning |

## 3.3 Form of Work and Dormitory

### 3.3.1 Form of Work

① Working hours

| | |
|---|---|
| Staff | 9:00~18:00 |
| First shift | 9:00~17:00 |
| Second shift | 17:00~ 1:00 |
| Third shift | 1:00~ 9:00 |

② Number of Personal

| | |
|---|---|
| Japanese study team | 15 engineers |
| Mongolian counterparts | |
| Staff-members | 15 persons (incl. 3 foremen) |
| Driller(tunneling) | 6 |
| Mechanic | 3 |
| Electrician | 3 |
| Operator of Generator | 3 |
| Surveyor | 1 |
| Cooks for Japanese team | 3 |
| Cooks for Mongolian team | 3 |
| Clerk | 1 |
| Guards | 4 |
| Driver | 3 |
| Laundry and sweeping worker | 2 |
| Assistant worker | 5 (incl. 3 Samplers) |
| Driller(boring) | 6 |
| Sampler | 3 |
| Interpreter | 4 |

### 3.3.2 Dormitory

| | |
|---|---|
| Japanese-built Gels (5 of 10m$\phi$ + 3 of 6m$\phi$) | 8 |
| Messroom (Gel of 10m$\phi$) | 1 |
| Bathrooms (20-feet containers) | 4 |
| Kitchen (40-feet container) | 1 |
| Laundry (40-feet container) | 1 |

Drinking water (transported from a well 6km away from Tsav, stored in a 50m$^3$ tank and distributed via pipes)

Water treatment (a septic tank, cap. 21 persons)

Accommodation for Mongolian personal (a wooden prefabricated 52.5m×12.5m building)

Electricity (a 750KVA power generation systems)

## 3.4 Organization of Survey Teams

(1)Japanese team

| Name | Title | Company |
|---|---|---|
| Kunitoshi OE | General Manager | ( MINDECO* ) |
| Mamoru OOSHITA | Chief Admistrator | ( 〃 ) |
| Iwao SASAHARA | Chief Mechanical Engineer | ( 〃 ) |
| Michihiko HASEGAWA | Chief Electrical Engineer | ( 〃 ) |
| Tomio HIKAGE | Chief Foreman for Tunnelling | ( 〃 ) |
| Yoshio MORIHIRO | Chief Foreman for Tunnelling | ( 〃 ) |
| Kiyoshi MITANI | Chief Foreman for Tunneling | ( 〃 ) |
| Kazuo YOKOKAWA | Vice Foreman for Tunnelling | ( 〃 ) |
| Masaki SUMIYA | Vice Foreman for Tunnelling | ( 〃 ) |
| Hiroyuki HASHIMOTO | Vice Foreman for Tunnelling | ( 〃 ) |
| Yuji KATABE | Chief Foreman for Drilling | ( 〃 ) |
| Takenori IKEDA | Chief Foreman for Drilling | ( 〃 ) |
| Tomoyuki OIKAWA | Chief Foreman for Drilling | ( 〃 ) |
| Kiyohisa SHIBATA | Chief Geologist | ( 〃 ) |
| Masatoshi MURATA | Metallurgist | ( 〃 ) |

*MINDECO : MITSUI MINERAL DEVELOPMENT ENGINEERING CO.,LTD.

(2)Mongolian team

| Name | Title | Company |
|---|---|---|
| Lodoin AYUR | General Project Manager<br>Director of TSAV Company | (MEGM* and TSAV Co.) |
| L. BYAMBAJAV | Deputy Director of TSAV Company | (TSAV Co.) |
| T. ERDENE | Mining Engineer, Economist | ( 〃 〃 ) |
| R. BATBAYAR | Mechanical Engineer | ( 〃 〃 ) |
| Y. LUTBAATAR | Tunnel Superintendant | ( 〃 〃 ) |
| SH. NAMHAINYAMBUU | Electrical and mechanical Engineer | ( 〃 〃 ) |
| TS. NOROVSAMBUU | Chief Geologist | ( 〃 〃 ) |
| YA. DOLGOR | Geologist | ( 〃 〃 ) |
| N. TSOLMON | Economist | ( 〃 〃 ) |
| B. KHALZAN | Administration Manager | ( 〃 〃 ) |
| D. MUNHTSETSEG | Chief Accountant | ( 〃 〃 ) |
| TS. DASHZEVGE | Clerk | ( 〃 〃 ) |
| V. KHURELTUMUR | Foreman for Tunnelling | ( 〃 〃 ) |
| L. KHURELBAATAR | Foreman for Tunnelling | ( 〃 〃 ) |
| B. BAATARHUU | Foreman for Tunnelling | ( 〃 〃 ) |

*MEGM : THE MINISTRY OF ENERGY, GEOLOGY AND MINING OF MONGOLIA

## 3.5 Project Schedule

| | Contents of Study | Quantity | June | July | August | September | October | November | December | January | February | Note |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Trip to Mongolia | | | | | | | | | | | |
| | Return trip | | | | | | | | | | | |
| 2 | Transportation of Equipment | | | | | | | | | | | |
| | To Mongolia | | | | | | | | | | | |
| | From Mongolia | | | | | | | | | | | |
| 3 | Tunneling | | | | | | | | | | | |
| | Drift Northward | 120.0m | | | | | | | | | | |
| | No.5 Waste Pit | 30.0m | | | | | | | | | | |
| | Core Boring | | | | | | | | | | | |
| | Underground(6pcs) | 185.0m | | | | | | | | | | |
| | Surface (8pcs) | 290.0m | | | | | | | | | | |
| | Pre-F/S and Mineral Dressing Test | | | | | | | | | | | |
| 4 | Report | | | | | | | | | | | |
| 5 | Delegates | | | | | | | | | | | |
| | General Manager | 1people | | | | | | | | | | |
| | Administor | 1 | | | | | | | | | | |
| | Chief Tunneling | 3 | | | | | | | | | | |
| | Assistant Tunneling | 3 | | | | | | | | | | |
| | Chief Mechanical | 1 | | | | | | | | | | |
| | Chief Electrical | 1 | | | | | | | | | | |
| | Chief Geologist | 1 | | | | | | | | | | |
| | Chief Boring | 3 | | | | | | | | | | |
| | Metallurgist | 1 | | | | | | | | | | |

Table 2 Project Schedule

# II . Progress of Survey

## Part Ⅱ . Progress of Survey

### Chapter 1. Tunneling Survey

#### 1.1. Location of Tunneling

Drifting in 1995 was commenced at the ending point of a northward drift(750.68m above sea level) driven in the preceding year (refer to Project in 1995).

#### 1.2 Survey Method

##### 1.2.1 Length of drift

| Work | Project | Actual |
|---|---|---|
| Horizontal Drift | (120.0m) | (121.0m) |
| Type 1 | 50.0m | 43.0m |
| Type 2 | 70.0m | 78.0m |
| Fifth Waste Pit | (30.0m) | (33.1m) |
| Type 1 | 10.0m | 9.3m |
| Type 2 | 23.0m | 16.6m |
| Type 3 | | 7.2m |
| Total | 150.0m | 154.1m |

Table 3 Length of drift

##### 1.2.2 Specifications of drift

(1) Effective section for each drift:

| | Area(m2) | Width(m) Height(m) |
|---|---|---|
| Tunnel type 1 | 11.88 | ( 4.0 × 3.4 ) |
| Tunnel type 2 | 11.88 | ( 4.0 × 3.4 ) |
| Tunnel type 3 | 11.88 | ( 4.0 × 3.4 ) |

(2) Gradient of horizontal drift: ± 1/500

(3) Elevation of starting point in 1995: GL=750.68m

(4) Direction of horizontal drift: Ref. Project in 1995

### 1.2.3 Main equipment

| Equipment | Specifications | Quantity | Note |
|---|---|---|---|
| Drill Jumbo | Hydraulic 2 Boom | 1 | use as rock-bolt |
| Mortal Charger Car | TOYOTA Hilux | 1 | Mortal Pump(MM151) |
| Load Haul Dump | $3.8m^3$ class | 2 | |
| Explosive Chager | TOYOTA Hilux | 1 | AN-FO charger(75Kg) |
| Compressor | $21m^3$/min | 1 | |
| Mini-Backhoe | $0.1m^3$ class | 1 | with Breaker |
| Jack-hammer | 30Kg class | 2 | |
| Track | 2t,attached with crane | 1 | |
| Track | 1t, Hilux,W-Cabin | 1 | |
| Wagon | Landcruser 80type | 1 | |
| Wagon | Landcruser 70type | 1 | |
| Generator | 750KVA | 2 | |
| Generator | 55KVA | 1 | |
| Generator | 10KVA DCA-13SPK | 2 | |
| Submersible Pump | 5.2KW BS-2102HT | 3 | |
| Submersible Pump | 2.2KW BS-2066 | 2 | |
| Deep-well Pump | 3.7KW SP-5A-19type | 2 | |
| Fan | 1,000mmϕ,300mmAq.75Kw | 1 | |
| | 900mmϕ,100mmAq.18.5Kw | 2 | |
| Car Washer | Pressure $65Kg/cm^2$ | 1 | |
| High-speed Cutter | | 2 | |
| Baby-Compressor | 3.7P-14V5, 230ℓ | 1 | |
| Electric Welder | BPZ-400-3 | 2 | |
| Engine Welder | BLW-150SS | 1 | |
| Feed-Pump | 25BISND5.4 | 1 | |
| Fuel-Pump | | 2 | for vehicles |
| Fuel-Pump | | 1 | for generator |
| INMARSAT Communication | | 1 | |
| Travelling Crane | 5t electric crane | 2 | |

Table 4 Main Equipment in Tunneling

## 1.3 Details of Tunneling

### 1.3.1 Horizontal drifting

A northward tunnel (327° E, 120m) was driven in 1995 from the ending point of a northward drift which driven in the preceding year.

The tunneling has been carried out by truckless method using hydraulically-operated mobile jumbo (drilling blast holes) and load haul dump (diesel driving underground wheel loader). Burn cut blasting method was adopted. In order to maintain the wall rock of the smooth blasting was used. In some places, steel frames or rock bolts were applied to support the tunnel wall.

### 1.3.2 Fifth Waste pit

A drift of 33.1m long with 32° E direction was driven at 196m from the entrance of the northward drift. The tunneling has been carried out by same truckless method as in the case of drifting.

## Chapter 2. Diamond Drilling Survey

### 2.1 Locations of drilling

Surface drilling to −30m level and underground drilling at intervals of 20m from within the horizontal tunnel at −60m level driven in 1995, both aimed at the No.4 vein, were executed. (Ref. Project in 1995)

### 2.2 Survey Method

#### 2.2.1 Length of drilling

| No. of hole | Project | Actual | Bearing | Inclination |
|---|---|---|---|---|
| MJMT-15 | 20.00m | 20.20m | 0° | 0° |
| MJMT-16 | 25.00m | 25.70m | 0° | 0° |
| MJMT-17 | 30.00m | 30.10m | 0° | 0° |
| MJMT-18 | 35.00m | 35.30m | 0° | 0° |
| MJMT-19 | 35.00m | 35.20m | 0° | 0° |
| MJMT-20 | 40.00m | 40.00m | 0° | 0° |
| MJMT-21 | 40.00m | 40.50m | -60° | 0° |
| MJMT-22 | 40.00m | 43.60m | -60° | 0° |
| MJMT-23 | 40.00m | 40.70m | -60° | 0° |
| MJMT-24 | 35.00m | 37.60m | -60° | 0° |
| MJMT-25 | 35.00m | 36.50m | -60° | 0° |
| MJMT-26 | 30.00m | 34.70m | -60° | 0° |
| MJMT-27 | 35.00m | 36.60m | -60° | 0° |
| MJMT-28 | 35.00m | 35.50m | -60° | 0° |
| Total | 475.00m | 492.20m | | |

Table 5 Length of drilling

#### 2.2.2 Drilling Method

The wireline method with NQ and BQ size was used. At the collar of bore holes, single tubes with diameters of 96mm and 66mm were used, and afterwards, outer tubes were inserted.

### 2.2.3 Main Equipment

| Item | Model | Quantity | Note |
|---|---|---|---|
| Drilling Machine | L-38-98 | 1 | Capacity NQ 565m BQ 725m |
| | | | Inner Diameter of Spindle 76mm |
| Engine | | 1 | 48Ps/1800rpm |
| Drilling Pump | MG-15h | 1 | Pistone 68mm, |
| | | | Pressure 45kg/cm² |
| Engine | NF-19K | 1 | 15~20Ps |
| Pump | LB-400 | 1 | Capacity 120l/min |
| Motor | | 1 | 0.4kw/200v |
| Drilling Rig | | 1 | |
| Hand Mixer | UM-22 | 1 | 0.4kw/100v |
| Water Tank | | 5 | 2,000l |
| Drill Rod | NQ-WL | 15 | 3.0m/pc |
| Core Barel Ass'y | NQ-WL | 1 | 1.50m/pc |
| Core Tube Ass'y | 96mm | 1 | 0.50m/pc |
| 〃 | NS76-55 | 1 | 1.50m/pc |
| Inner Tube Ass'y | NQ-U WL | 1 | 1.50m/pc |
| Inner Tube | NQ-U WL | 1 | 1.50m/pc |
| Outer Tube | NQ-U WL | 1 | 1.50m/pc |
| Over Shot Ass'y | NQ-U WL | 1 | |
| Water Swivel | C-U WL | 1 | |
| W-L Hoist | | 1 | Capacity NQ 565m |

Table 6 List of Main Equipment for Surface Drilling

| Item | Model | Quantity | Note |
|---|---|---|---|
| Drilling Machine | L-24-62 | 1 | Capacity BQ 150m AQ225m |
| | | | Inner Diameter of Spindle 62mm |
| Motor | | 1 | 5.5kw 4P |
| Drilling Pump | MG-5h | 1 | Pistone 68mm |
| | | | Pressure 25～60kg/cm² |
| Motor | Inverter motor | 1 | 3.7kw 4P |
| Pump | LB-400 | 1 | Capacity 120ℓ/min |
| Motor | | 1 | 0.4kw/200v |
| Column Jack | | 1 | Screw Jack Style |
| Hand Mixer | UM-22 | 1 | 0.4kw/100v |
| Water Tank | | 1 | 2,000ℓ |
| Drill Rod | BQ-WL | 35 | 1.5m/pc |
| Core Barel Ass'y | 66mm | 2 | 0.50m/pc |
| Core Tube Ass'y | BQ-U WL | 1 | 1.50m/pc |
| Inner Tube Ass'y | BQ-U WL | 1 | 1.50m/pc |
| Inner Tube | BQ-U WL | 1 | 1.50m/pc |
| Outer Tube | BQ-U WL | 1 | 1.50m/pc |
| Over Shot Ass'y | BQ-U WL | 1 | |
| Water Swivel | BQ-U WL | 1 | |
| W-L Hoist. | | 1 | Capacity 50m |

Table 7 List of Main Equipment for Underground Drilling

2.2.4 Core storage

All the drill cores were kept in the core boxes and stored in the core house.

| Size | Length | Width | Height | Core box |
|---|---|---|---|---|
| 96mm | 1,030mm | 390mm | 90mm | wooden |
| NQ | 1,040mm | 441mm | 55mm | plastic |
| BQ | 1,040mm | 435mm | 43mm | plastic |

2.2.5 Water for drilling

Water for the surface drilling, together with drinking water, was carried from the well. For the underground drilling, water from the underground tunnels and drill holes was pumped up.

2.3 Results of drilling

2.3.1 MJMT–15

| | | |
|---|---|---|
| Drill length | : | 20.25m |
| Core length | : | 19.90m |
| Core recovery | : | 98.5% |
| Drilling commenced | : | Sep. 4, 1995 |
| Drilling completed | : | Sep. 5, 1995 |

From the collar to 1.40m, a 66mm single diamond bit was used, then redrilled by a 76mm pilot bit. From 1.40m to 18.84m, a NQ wireline diamond bit was used to drill through granodiorite. A quartz vein accompanied by ores and clay was observed from 14.00m to 15.20m. The amount of spring water was 3 ℓ /min.

2.3.2 MJMT–16

| | | |
|---|---|---|
| Drill length | : | 25.70m |
| Core length | : | 25.70m |
| Core recovery | : | 100.0% |
| Drilling commenced | : | Sep. 5, 1995 |
| Drilling completed | : | Sep. 7, 1995 |

Drilled from the collar to 1.40m by a 66mm single diamond bit and redrilled by a 76mm pilot bit. From 1.40m to 25.70m, a NQ wireline diamond bit drilled through granodiorite. A quartz vein accompanied by clay and ores was encountered at 14.65m ～ 15.00m. The amount of spring water was 3 ℓ /min.

### 2.3.3 MJMT－17

| | | |
|---|---|---|
| Drill length | : | 30.10m |
| Core length | : | 30.00m |
| Core recovery | : | 99.7% |
| Drilling commenced | : | Sep. 7, 1995 |
| Drilling completed | : | Sep. 9, 1995 |

Drilled from the collar to 1.60m by a 66mm single diamond bit and then redrilled by a 76mm pilot bit. Drilled from 1.60m to 15.70m by a NQ and from 15.70m to 30.10m by a BQ wireline diamond bit through granodiorite. Quartz veins with ores and clay were encountered at 19.80m ～ 20.60m and 25.00m ～ 25.30m. The amount of spring water was 10 ℓ /min.

### 2.3.4 MJMT－18

| | | |
|---|---|---|
| Drill length | : | 35.30m |
| Core length | : | 35.10m |
| Core recovery | : | 99.4% |
| Drilling commenced | : | Sep. 9, 1995 |
| Drilling completed | : | Sep. 12, 1995 |

Drilled from the collar to 1.20m by a 66mm single diamond bit and redrilled by a 76mm pilot bit. Drilled from 1.20m to 21.80m by a NQ wireline diamond bit through granodiorite. Quartz Veins with mineral and clay were encountered at 18.90m ～ 19.00m and 29.70m ～ 29.80m. The amount of spring water was 7 ℓ /min.

### 2.3.5 MJMT－19

| | | |
|---|---|---|
| Drill length | : | 35.20m |
| Core length | : | 34.30m |
| Core recovery | : | 97.4% |

Drilling commenced : Aug. 31, 1995
Drilling completed : Sep. 2, 1995

Drilled from the collar to 1.00m by a 66mm single diamond bit and redrilled by a 76mm pilot bit. Drilled from 1.20m to 22.00m by a NQ and 22.00m ~ 35.20m by a BQ wireline diamond bit through granodiorite. Mineralization with clay was encountered at 23.95m ~ 25.00m. The amount of spring water was 8 ℓ /min.

2.3.6 MJMT-20

Drill length : 40.00m
Core length : 39.70m
Core recovery : 99.3%
Drilling commenced : Aug. 28, 1995
Drilling completed : Aug. 31, 1995

Drilled from the collar to 18.20m by a NQ and from 18.20m to 40.00m by a BQ wireline diamond bit through granodiorite(0.00m ~ 1.30m) and andesitic tuff(1.30m ~ 40.00m). A quartz vein with ores and clay was encountered at 32.00m ~ 32.60m. The amount of spring water was 10 ℓ /min.

2.3.7 MJMT-21

Drill length : 40.50m
Core length : 37.70m
Core recovery : 93.1%
Drilling commenced : Aug. 25, 1995
Drilling completed : Aug. 28, 1995

Drilled from the collar to 2.40m by a 96mm single diamond bit, from 2.40m to 3.00m by NS76-55 core pack and from 3.00m to 40.50m by a NQ wireline diamond bit throughout surface soil(0.00m ~ 2.30m), andesite(2.30m ~ 30.30m) and andesitic tuff(30.30m ~ 40.50m). A quartz vein with ores and clay was encountered at 27.20m ~ 30.30m.

### 2.3.8 MJMT-22

| | |
|---|---|
| Drill length | : 43.60m |
| Core length | : 42.90m |
| Core recovery | : 98.4% |
| Drilling commenced | : Aug. 21, 1995 |
| Drilling completed | : Aug. 24, 1995 |

Drilled from the collar to 2.30m by a 96mm single metal bit, from 2.30m to 5.80m by NS76--55 core pack and from 5.80m to 43.60m by a NQ wireline diamond bit through surface soil(0.00m ~ 5.80m) and granodiorite(5.80m ~ 43.60m). Quartz vein with clay was observed from 32.20m to 39.30m.

### 2.3.9 MJMT-23

| | |
|---|---|
| Drill length | : 40.70m |
| Core length | : 39.90m |
| Core recovery | : 98.0% |
| Drilling commenced | : Aug. 16, 1995 |
| Drilling completed | : Aug. 19, 1995 |

Drilled from the collar to 3.00m by a 96mm single metal bit, from 3.00m to 7.30m by NS76-55 core pack and 7.30m to 40.70m by a NQ wireline diamond bit through surface soil(0.00m ~ 3.00m) and granodiorite(3.00m ~ 40.70m). Quartz veins were observed from 34.05m to 34.15m, 36.00m to 36.60m and from 38.00m to 39.10m, respectively.

### 2.3.10 MJMT-24

| | |
|---|---|
| Drill length | : 37.60m |
| Core length | : 36.80m |
| Core recovery | : 97.9% |
| Drilling commenced | : Aug. 11, 1995 |
| Drilling completed | : Aug. 15, 1995 |

Drilled from the collar to 3.00m by a 96mm single metal bit, from 3.00m to 4.90m by NS76-55 core pack and 4.90m to 37.60m by a NQ wireline diamond bit through surface soil(0.00m ~ 3.60m) and granodiorite(3.60m ~ 37.60m). A quartz vein was observed from 36.10m to 36.6m.

### 2.3.11 MJMT-25

| | |
|---|---|
| Drill length | : 36.50m |
| Core length | : 33.70m |
| Core recovery | : 92.3% |
| Drilling commenced | : Aug. 7, 1995 |
| Drilling completed | : Aug. 10, 1995 |

Drilled from the collar to 2.70m by a 96mm single metal bit, from 2.70m to 4.90m by NS76-55 core pack and 4.90m to 36.50m by a NQ wireline diamond bit through surface soil(0.00m ~ 4.10m) and granodiorite. A quartz vein was observed from 30.50m to 30.70m.

### 2.3.12 MJMT-26

| | |
|---|---|
| Drill length | : 34.70m |
| Core length | : 30.90m |
| Core recovery | : 89.0% |
| Drilling commenced | : Aug. 2, 1995 |
| Drilling completed | : Aug. 5, 1995 |

Drilled from the collar to 3.20m by a 96mm single metal bit and from 1.50m to 34.70m by a NQ wireline diamond bit through surface soil(0.00m ~ 3.20m) and granodiorite(3.20m ~ 34.70m). Quartz veins were encountered from 27.85m to 28.15m and from 30.60m to 31.70m.

### 2.3.13 MJMT-27

| | |
|---|---|
| Drill length | : 36.60m |
| Core length | : 33.70m |
| Core recovery | : 92.1% |
| Drilling commenced | : Jul. 29, 1995 |
| Drilling completed | : Aug. 1, 1995 |

Drilled from the collar to 2.10m by a 96mm single metal bit and from 2.10m to 36.65m by NQ wireline diamond bit through surface soil(0.00m ~ 3.10m) and granodiorite(3.10m ~ 36.60m). A quartz vein was observed from 24.95m to 25.30m.

### 2.3.14 MJMT-28

| | | |
|---|---|---|
| Drill length | : | 35.50m |
| Core length | : | 33.80m |
| Core recovery | : | 95.2% |
| Drilling commenced | : | Jul. 25, 1995 |
| Drilling completed | : | Jul. 28, 1995 |

Drilled from the collar to 1.90m by a 96mm single metal bit and from 1.90m to 35.50m by a NQ wireline diamond bit through surface soil(0.00m ~ 1.80m) and granodiorite(1.80m ~ 35.50m). A quartz vein was observed from 25.60m to 26.75m.

## Chapter 3. Survey Findings

### 3.1 Tunneling Survey

Tunneling sketches with a 1/200 scale are exhibited in Fig.1 ~ 3, PL−4(1) ~ (3), a 1/500 geological map at the tunnel level in Fig.4, PL−5, a sampling location map in Apx−10, a list of assay results in Apx−11, and a list of the mineralized zones in Table 8, respectively. Observation of thin sections of rocks appears in Apx−14, their photomicrographs in Apx−15, observation of polished sections of ores in Apx−16, and their photomicrographs in Apx−17, respectively.

| Drift | Location m | Sample Length m | Grade | | | | |
|---|---|---|---|---|---|---|---|
| | | | Au g/t | Ag g/t | Pb % | Zn % | Cu % |
| No. 5 Waste pit north side | 23. 85~ 28. 05 | 4. 20 | 1. 2 | 64. 8 | 6. 97 | 6. 54 | 0. 09 |
| No. 5 Waste pit south side | 19. 80~ 23. 20 | 2. 30 | 0. 5 | 80. 7 | 8. 42 | 5. 43 | 0. 17 |

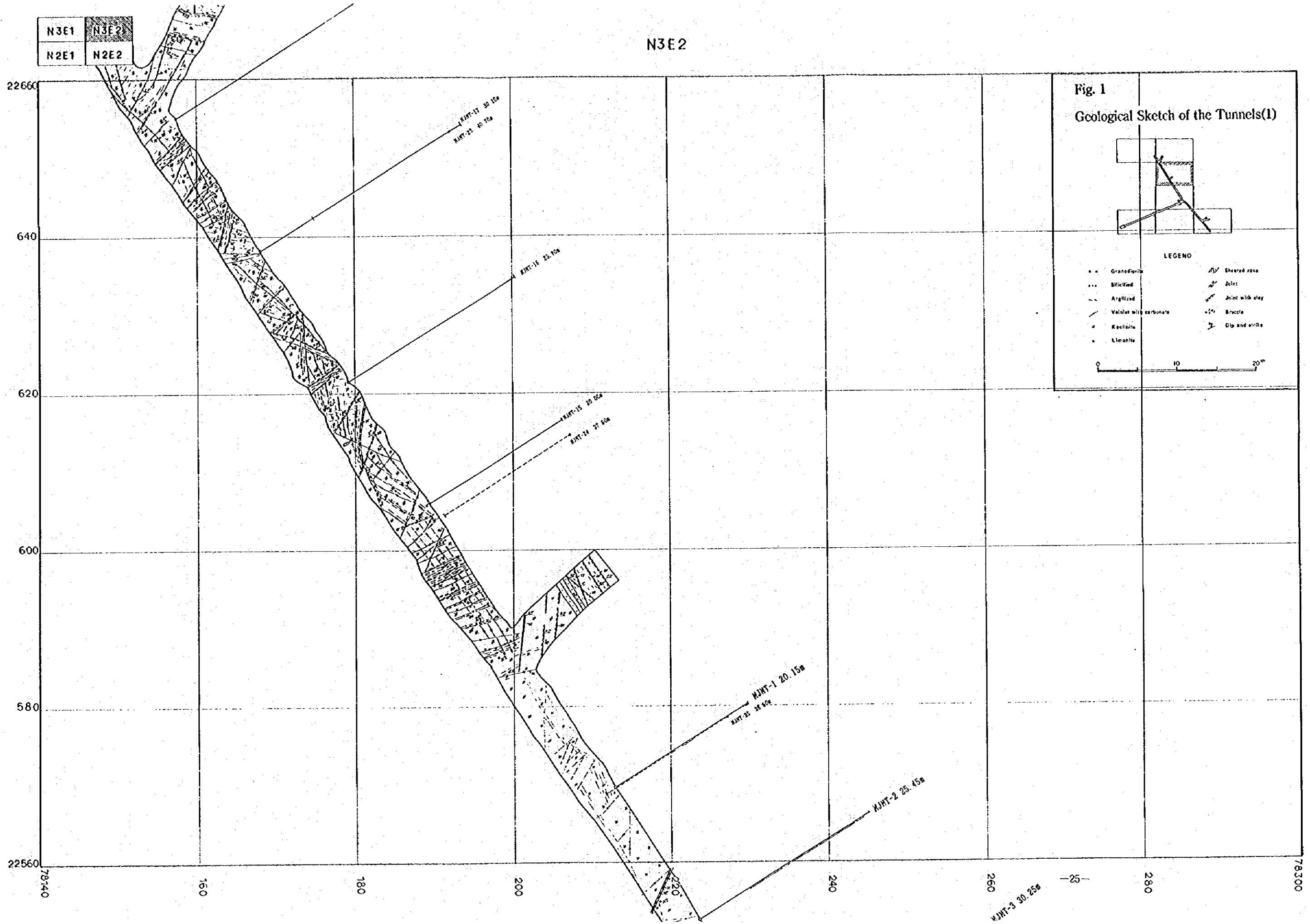
Table 8 Mineralized Zones in the Tunnels

#### 3.1.1 Northward tunnel

The starting point of the northward tunnel corresponds with the previous year's terminal point of the same tunnel. Henceforth, tunnel depths are expressed in horizontal distance on the center line of the tunnel from the northward starting point.

The tunnel is dominated over the full length by chloritized granodiorite. It abounds in joints with NEN−WSW and NW−SE trends accompanied by carbonate or clay of several millimeters to 1 centimeter in width. Relatively large fracture zones with clay are observed in the sections of 9.7 ~ 11.7m(N65° E, 80° S), 39.1 ~ 39.9m(N55° E, 70° S), 60.9 ~ 62.3m(N15° E, 65° E) and 73.2 ~ 74.6m(N45° E, 70° ~ 80° S).

Analysis at seven sampling points(T−6049 ~ 6055) in these fracture zones and small argillized veins did not result in discovery of a mineralized zone; at any of the sampling points, Au, Ag and Cu grades were lower than detection limits while Pb



N3E1
N3E2
N2E1
N2E2
N3E2
Fig. 1
Geological Sketch of the Tunnels(1)
LEGEND
Granodiorite
Silicified
Argillized
Veinlet with carbonate
Kaolinite
Limonite
Sheared zone
Joint
Joint with clay
Breccia
Dip and strike
0
10
20m
22660
640
620
600
580
22560
78140
160
180
200
220
240
260
280
78300
MJMT-1 20.15m
MJMT-2 25.45m
MJMT-3 30.25m

N4E2

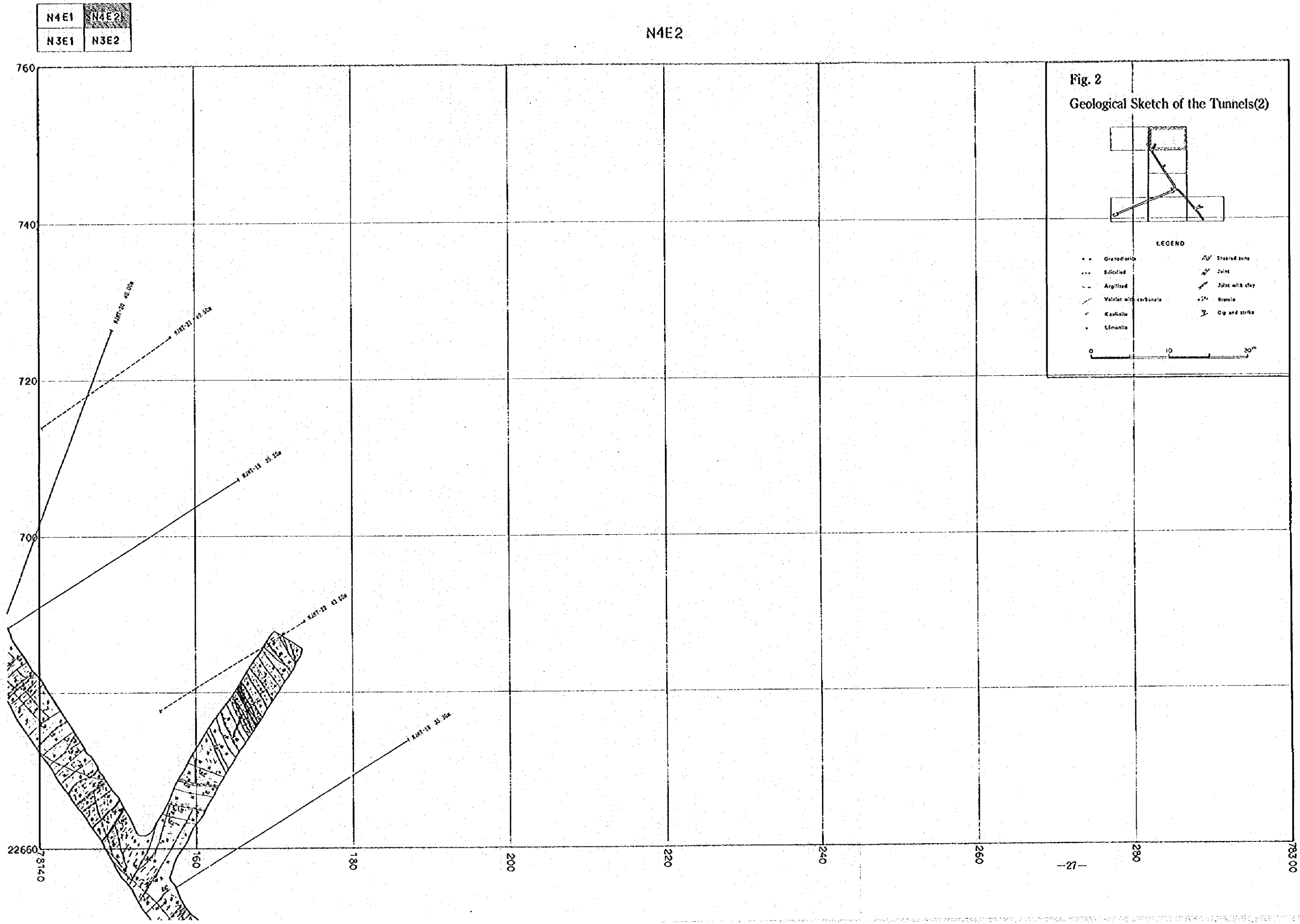



Fig. 2
Geological Sketch of the Tunnels(2)

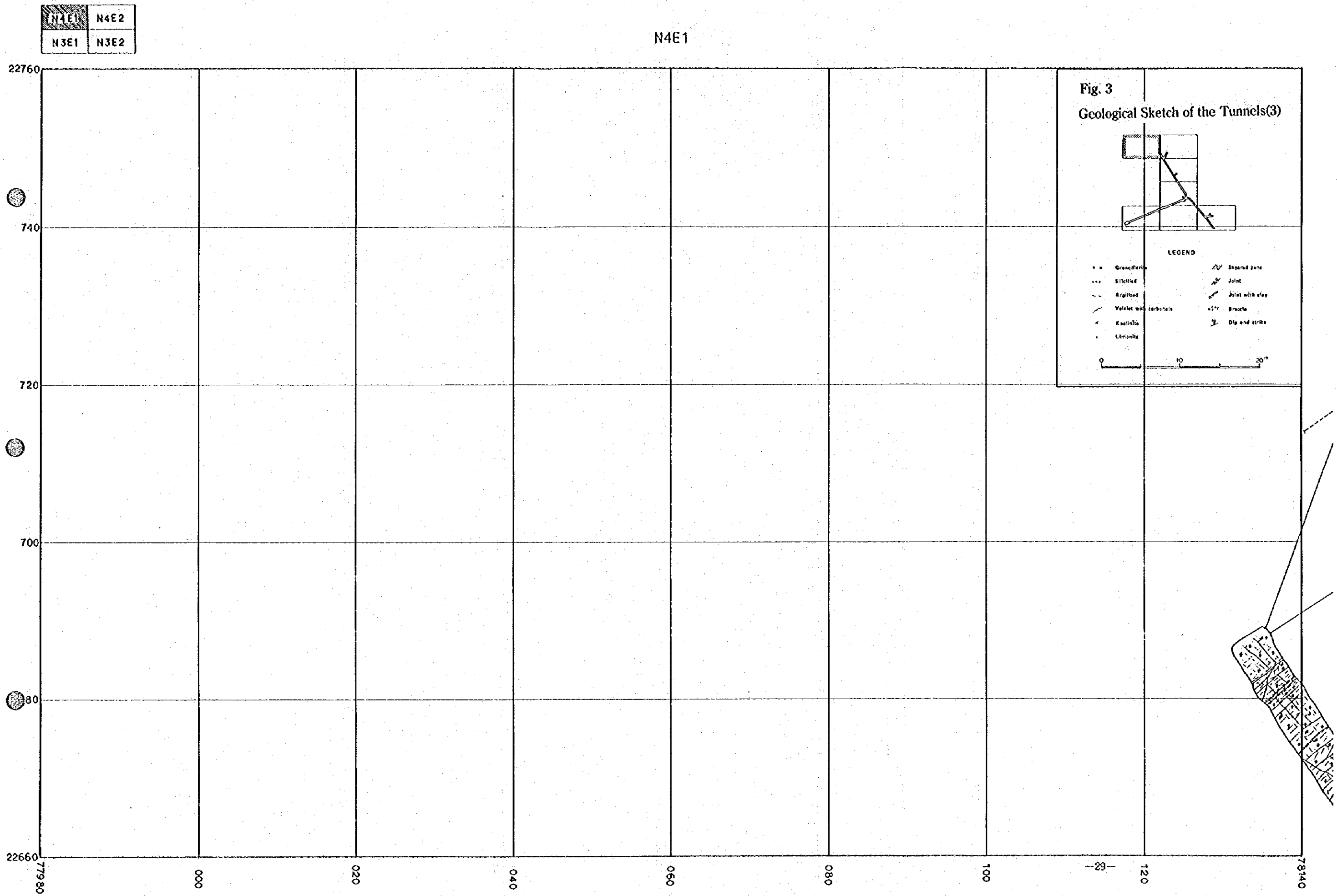



Fig. 3
Geological Sketch of the Tunnels(3)

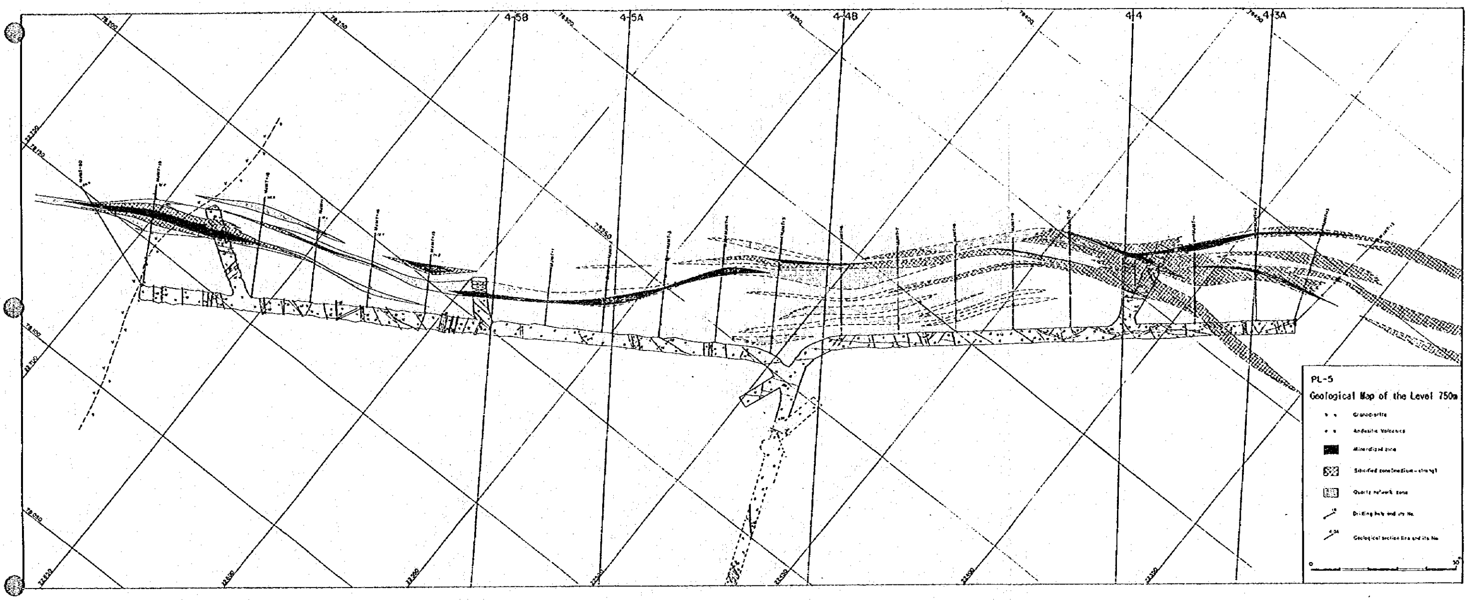



Fig. 4 Geological Map of the Level 750m

and Zn grades were as low as 0.01 ~ 0.04% and 0.02 ~ 0.21%, respectively.

Microscopic observation of granodiorite(sample 60-1-229) taken at the 115m point shows holocrystalline, equigranular textures. The main mineral components are sericitized plagioclase, chloritized mafic minerals, quartz and orthoclase of irregular shapes, associated with euhedral ~ subhedral opaque minerals and apatite.

### 3.1.2 No.5 waste pit

The starting point of the No.5 dump tunnel corresponds approximately with 197.0m point of the northward tunnel. Depths are expressed in horizontal distance from the starting point along the drifting direction line.

This tunnel comprises of granodiorite over the full length, which receives chloritization up to the 20.3m point. In the shallow part, dominant are clay veins as well as carbonate mineral veins with WNW-ESE trends, whereas carbonate mineral veins with NW-SE trends dominate in the deep part. From 20.3m to 28.2m, there are silicification-sericitization zones. Between 21.2m and 24.1m, there appears a mineralized zone (N65° W, 75° ~ 80° N) which bears quartz and carbonate mineral veins accompanied by high grade Pb and Zn ore veins of 10 ~ 15cm wide. Thereafter, sericitized and chloritized granodiorite continues till the end.

Under the microscope, silicified-sericitized zones(samples 60-5-1, -7 and -10) reveals that feldspar almost completely alters to sericite and carbonate minerals while mafic minerals change into sericite(muscovite), chlorite and opaque minerals; their original textures being unknown. Quartz is of irregular shapes and shows weak wavy extinction. The alteration zones are accompanied by quartz and carbonate mineral veinlets, occasionally associated with sphalerite. Microscopic observation of mineralized veins(sample 60-5-3) clearly indicates brecciated texture. Quartz contained in breccia often takes the form of aggregates with an approximate grain size of 0.21 x 0.41mm, which are considered to have been recrystallized.

Analysis of the mentioned mineralized zones indicates relatively high grades:

| | sampling length(m) | Au (g/t) | Ag (g/t) | Pb (%) | Zn (%) | Cu (%) |
|---|---|---|---|---|---|---|
| Southeast wall | 2.3 | 0.5 | 80.7 | 8.42 | 5.43 | 0.17 |
| Northwest wall | 4.2 | 1.2 | 64.8 | 6.97 | 6.54 | 0.09 |

Under the microscope, ore minerals occurring in quartz–carbonate veins accompanied by high grade Pb and Zn ores(samples 60–5–3, –5, –6 and –8) are mainly pyrite, sphalerite, galena and chalcopyrite, always accompanied by tetrahedrite. Also recognized are polybasite(sample 60–5–3), native silver(60–5–5) and electrum(60–5–8). On the other hand, the silicified rock containing quartz–carbonate veinlets(60–5–1, –7 and –9) is characterized with the facts that, among the mentioned ore minerals, chalcopyrite occurs only as inclusion in sphalerite and that no tetrahedrite occurs in it.

## 3.2 Drilling Survey

Drill core loggings(1/200 scale) are shown in Apx–12, a geological map by tunnel levels in Fig.4, PL–5, assay results in Apx–13, and a list of the mineralized zones in Table 9, respectively. Besides, observation of thin sections appears in Apx–14 and the photomicrographs in Apx–15 while observation of polished sections is shown in Apx–16 and the photomicrographs in Apx–17.

The survey results by boreholes are described in the following paragraphs:

### 3.2.1 MJMT–15

Granodiorite extends all over the borehole, through which weak chloritization is observed. In the sections between 8.1m and 11.5m(from the starting point of the borehole), argillization and weak silicification accompanied by quartz veinlets are observed. Between 14.1m and 16.8m, there are quartz–carbonate veins accompanied by galena and sphalerite, assaying Au 0.3g/t, Ag 90.2g/t, Pb 8.58%, Zn 5.09% and Cu 0.1%(core length 2.7m).

Under the microscope, it is composed mainly of pyrite, galena, sphalerite and chalcopyrite, associated with tetrahedrite and polybasite. Chalcopyrite alters partially to chalcocite and covelline; and, cerussite is produced along the peripheries and cracks of galena.

| Hole No. | Location<br>m | Sample<br>Length<br>m | Grade<br>Au<br>g/t | <br>Ag<br>g/t | <br>Pb<br>% | <br>Zn<br>% | <br>Cu<br>% |
|---|---|---|---|---|---|---|---|
| MJMT-15 | 14.10~16.80 | 2.70 | 0.3 | 90.2 | 8.58 | 5.09 | 0.10 |
| MJMT-17 | 19.80~20.60 | 0.80 | 4.3 | 23.0 | 3.19 | 2.91 | 0.08 |
| 〃 | 25.00~25.30 | 0.30 | 0.3 | 2.3 | 0.62 | 4.56 | 0.06 |
| MJMT-18 | 18.90~19.00 | 0.10 | 1.7 | 30.0 | 3.80 | 3.09 | 0.04 |
| MJMT-19 | 23.95~26.20 | 2.25 | 0.2 | 176.9 | 12.50 | 5.26 | 0.26 |
| MJMT-20 | 27.65~27.75 | 0.10 | <0.1 | <0.1 | 0.88 | 4.72 | 0.04 |
| 〃 | 32.00~33.20 | 1.20 | 1.2 | <0.1 | 0.28 | 13.46 | 0.46 |
| MJMT-21 | 26.30~30.30 | 4.00 | 2.1 | 0.3 | 1.81 | 1.34 | 0.18 |
| 〃 | 31.50~34.30 | 2.80 | <0.1 | <0.1 | 0.06 | 2.17 | <0.01 |
| MJMT-22 | 32.90~35.20 | 2.30 | 1.2 | 145.6 | 11.23 | 1.75 | 0.18 |
| MJMT-23 | 33.90~34.30 | 0.40 | <0.1 | <0.1 | 0.03 | 2.27 | <0.01 |
| 〃 | 38.00~39.10 | 1.10 | 0.2 | 16.2 | 1.39 | 0.57 | 0.14 |
| 〃 | 39.70~40.70 | 1.00 | <0.1 | 15.8 | 0.09 | 2.18 | 0.03 |
| MJMT-24 | 36.10~36.60 | 0.50 | 4.8 | <0.1 | 0.27 | 0.21 | 0.09 |
| MJMT-26 | 25.30~25.50 | 0.20 | 1.0 | <0.1 | 0.68 | 0.09 | 0.04 |
| 〃 | 29.20~31.70 | 2.50 | 4.4 | 2.7 | 0.45 | 0.26 | 0.06 |
| MJMT-27 | 31.90~32.75 | 0.85 | 6.7 | <0.1 | 0.39 | 0.10 | 0.02 |
| MJMT-28 | 26.10~26.75 | 0.65 | 2.3 | 17.9 | 3.24 | 0.18 | 0.14 |

Table 9 Mineralized zones by Boreholes

### 3.2.2 MJMT−16

Granodiorite predominates the entire borehole. Weak ~ medium grade chloritization is observed throughout. In sections 2.7 ~ 3.8m and 6.5 ~ 7.2m, strong argillization is observed, which are considered to be fault zones. Sections 5.3 ~ 6.5m and 14.2 ~ 16.1m are silicification zones. Between 14.65m and 15.00m, strong silicification zones are formed. Analysis of these alteration zones indicates grades that are too low for a mineralized zone.

### 3.2.3 MJMT−17

Granodiorite is dominant throughout. Weak to medium grade chloritization is observed almost over the full length of the borehole. In sections 13.85 ~ 15.15m, 19.4 ~ 21.6m and 24.4 ~ 25.5m, medium to strong silicification zones accompanied by quartz veinlets/network veins are formed. Between 19.8m and 20.6m and also between 25.0m and 25.3m, swarmed zones of quartz−carbonate veins accompanied by galena and sphalerite are formed.

Under the microscope(sample 17−25.1), brecciated texture with strong silicification, sericitization and carbonatization is observed. The carbonate minerals are distinguished into two types by the difference in index; those of lower indices, presumably calcite, occur mainly as the matrix of breccia, whilst those of relatively high indices, presumably rhodochrosite, occur mainly in veinlets.

Analysis of these swarmed zones of quartz−carbonate veins is as follows:

| Section(m) | Core length(m) | Au (g/t) | Ag (g/t) | Pb (%) | Zn (%) | Cu (%) |
|---|---|---|---|---|---|---|
| 19.8~20.6 | 0.8 | 4.3 | 23.0 | 3.19 | 2.91 | 0.08 |
| 25.0~25.3 | 0.3 | 0.3 | 2.3 | 0.62 | 4.56 | 0.06 |

Ore minerals(samples 17−19.85 and −25.1) consist of pyrite, galena and sphalerite. Chalcopyrite occurs only as inclusion in sphalerite. In the sample 17−19.85, occurrence of tetrahedrite and polybasite are confirmed.

### 3.2.4 MJMT−18

This borehole also comprises of granodiorite over the full length, through which weak chloritization is observed. Sections 17.5 ~ 19.7m, 25.9 ~ 26.55m and 29.7-32.5m are silicified zones containing quartz–carbonate veinlets. In a quartz–carbonate vein between 18.9m and 19.0m, a mineralized zone is confirmed, which assays Au 1.7g/t, Ag 30.0g/t, Pb 3.8%, Zn 3.09% and Cu 0.04%(core length 0.1m).

### 3.2.5 MJMT–19

From 0.0m to 8.8m point, granodiorite is dominant, while there is a breccia zone with sub–angular breccia(0.5 ~ 2cm in diameter) of andesite and andesitic tuff between 8.8m and 10.2m. Thereafter, andesitic tuff continues down to the hole end. Sections 10.2 ~ 15.3m and 21.5 ~ 27.2m are relatively strong silicified zones, while quartz veinlet zones with weak silicification are formed in sections 18.7 ~ 21.5m and 17.2 ~ 30.5m. In the 2.25m section between 23.95m and 26.2m, a large mineralized zone is found, which assays Au 0.2g/t, Ag 176.9g/t, Pb 12.50%, Zn 5.26% and Cu 0.26%. The 1.0m section from 26.2m to 27.2m represents a disseminated zone assaying Pb 0.09% and Zn 1.44%.

Microscopic observation of the brecciated zone(sample 19–9.3) indicates that andesitic rock fragments, which constitute the breccia, are of holocrystalline, porphyritic texture, and the phenocryst is composed of plagioclase while the groundmass is composed of lath–shaped plagioclase. All mafic minerals alter to chlorite ~ clay minerals. Andesitic tuff – constituent of the breccia – is lapilli tuff which has, as phenocryst, glass altering to clay minerals as well as somewhat corroded quartz. The breccia, chloritized and argillized, is replaced by calcite. The matrix is microcrystalline, chloritized and argillized, occasionally cut by rhodochrosite veinlets. Calcite veinlets also appear at the final stage of cutting all the structure. Microscopic observation of the mineralized zone(sample 19–24.6) indicates that there remain no original texture due to strong silicification–sericitization and has intrusion of quartz–rhodochrosite veinlets in all directions. Ore minerals are chiefly pyrite, as well as galena, sphalerite and chalcopyrite, accompanied by rhodochrosite. Chalcopyrite is accompanied by tetrahedrite of a minute quantity. The periphery of chalcopyrite partially alters to chalcocite and covelline.

### 3.2.6 MJMT–20

In the borehole, granodiorite occurs between 0.0m point to the 1.3m point, after which andesitic tuff occurs down to 40.0m, the hole end. Silicification zones with quartz veinlets occur in sections 27.65 ~ 27.75m and 29.90 ~ 33.20m. The former section has a mineralized zone assaying Au <0.1g/t, Ag <0.1g/t, Pb 0.88%, Zn 4.72% and Cu 0.04% (core length 0.1m) while, in the latter section, there is a 1.2m mineralized zone (32.0 ~ 33.2m) assaying Au 1.2g/t, Ag <0.1g/t, Pb 0.28%, Zn 13.46%, and Cu 0.46%.

Under the microscope, andesitic tuff(sample 20-9.0) is of holocrystalline brecciated texture. Rock fragments or portions, presumably glass, are mainly sericitized and also receives chloritization and epidotization, whilst the matrix is replaced by calcite. Occasionally, it is cut by chlorite, calcite or rhodochrosite-opaque minerals veinlets. Microscopic observation of the mineralized zone(sample 20-32.3) shows that it is composed mainly of sphalerite containing a large quantity of inclusion of chalcopyrite, and also of pyrite, chalcopyrite and galena(in a minute quantity). Along cracks of sphalerite, electrum is observed, whereas association with chalcocite and covelline is seen along the peripheries and cracks of chalcopyrite.

### 3.2.7 MJMT-21

This borehole comprises of andesitic tuff over the whole length. Between 26.0m and 34.3m, a strong ~ medium silicification zone accompanied by quartz veinlets is formed. In this section, mineralized zones in two layers have been caught, which assay as follows:

| Section(m) | Core length(m) | Au (g/t) | Ag (g/t) | Pb (%) | Zn (%) | Cu (%) |
|---|---|---|---|---|---|---|
| 26.3~30.3 | 4.0 | 2.1 | 0.29 | 1.81 | 1.34 | 0.18 |
| 31.5~34.3 | 2.8 | <0.1 | <0.1 | 0.06 | 2.17 | <0.01 |

The intermediate section(1.2m) held between the two mineralized sections assaying Au 0.5g/t, Ag<0.1g/t, Pb 0.24%, Zn 0.95% and Cu 0.08%, these three sections will possibly unify into a single mineralized zone.

Under the microscope, the andesitic tuff near the mineralized zone(sample 21-36.0) is of holocrystalline, brecciated texture and receives strong sericitization and chloritization. Volcanic glass receives argillization, and microcrystals of epidote align along the external periphery. Quartz–epidote veinlets and clay veinlets are also observed. Microscopic observation of the sample 21–30.0 indicates that sulphide minerals are almost oxidized, changing into Fe–Mn oxides.

### 3.2.8 MJMT–22

The borehole comprises of granodiorite, except section 21.2 ~ 22.0m where a dike occurs. A strong ~ medium silicification zone appears between 32.2m and 42.25m. Especially between 32.9m and 33.95m, there is a zone where quartz veins/veinlets are aggregated. Analysis of the 2.3m section(32.9 ~ 35.2m) resulted in Au 1.2g/t, Ag 145.6g/t, Pb 11.23%, Zn 1.75% and cu 0.18%, which represents a high grade mineralized zone.

The dike(sample 22–21.6) under the microscope is of holocrystalline, seriate texture and has tonalite composition consisting of plagioclase, quartz and mafic minerals(in minute quantity) with combinations varying from chlorite–opaque minerals to chlorite–epidote–opaque minerals. Alteration is relatively weak but partial replacement by calcite is observed.

### 3.2.9 MJMT–23

Granodiorite predominates throughout. Silicification zones are observed in sections 6.7 ~ 7.8m, 22.0 ~ 23.3m and 33.4 ~ 40.7m. All the alteration zones have low–grade disseminated portion of small sizes. The following are those which meet the ore reserve calculation criteria:

| Section(m) | Core length(m) | Au (g/t) | Ag (g/t) | Pb (%) | Zn (%) | Cu (%) |
|---|---|---|---|---|---|---|
| 33.9~34.3 | 0.4 | <0.1 | <0.1 | 0.03 | 2.27 | <0.01 |
| 38.0~39.1 | 1.1 | 0.2 | 16.2 | 1.39 | 0.57 | 0.14 |
| 39.7~40.7 | 1.0 | <0.1 | 15.8 | 0.09 | 2.18 | 0.03 |

### 3.2.10 MJMT–24

The borehole is also predominated by granodiorite over the full length. Silicification zones, though small in sizes, are found in sections 18.2 ~ 20.8m, 23.8 ~ 24.7m, 30.3 ~ 31.8m and 33.6 ~ 36.75m. The 0.5m section from 36.1m to 36.6m represents a quartz vein covered by Mn oxides, forming an Au–rich mineralized zone assaying Au 4.8g/t, Ag<0.1g/t, Pb 0.27%, Zn 0.21% and Cu 0.09%. Microscopic observation of relatively fresh granodiorite(sample 24–16.3) indicates holocrystalline, seriate texture. It is composed mainly of quartz, plagioclase and chloritized mafic minerals, accompanied by orthoclase of micrographic texture. Chloritization and epidotization are conspicuous.

### 3.2.11 MJMT–25

Granodiorite again predominates throughout. In sections 8.7 ~ 10.25m, 13.7 ~ 14.4m, 19.5 ~ 21.2m, 25.2 ~ 26.1m and 30.2 ~ 33.5m, there are silicification zones where no mineralized zones satisfying the ore reserve calculation criteria are found, though. Microscopic observation of quartz veins occurring in the silicification zone(sample 25–30.5) indicates that no sulphide minerals remain, having completely altered to Fe–Mn oxides, which suggests the possibility that a mineralized zone has leached out through supergene alteration.

### 3.2.12 MJMT–26

Granodiorite predominates except section 17.7 ~ 18.1m where a dike is found. Silicified zones embracing quartz veins/veinlets, stained by Mn oxides, are observed in sections 9.2 ~ 10.0m, 11.3 ~ 16.25m, 20.35 ~ 26.1m and 27.85 ~ 34.0m. These silicified zones are relatively rich in Au. Mineralized zones exceeding the ore calculation criteria are found in two sections:

| Section(m) | Core length(m) | Au (g/t) | Ag (g/t) | Pb (%) | Zn (%) | Cu (%) |
|---|---|---|---|---|---|---|
| 25.3~25.5 | 0.2 | 1.0 | <0.1 | 0.68 | 0.09 | 0.04 |
| 29.2~31.7 | 2.5 | 4.4 | 2.7 | 0.45 | 0.26 | 0.06 |

Under the microscope, the dike is of holocrystalline, seriate texture almost same as that of the tonalite confirmed in MJMT–24; however, alteration is somewhat weaker.

### 3.2.13 MJMT–27

Excepting dikes in sections 14.85 ~ 15.4m and 17.0 ~ 17.7m, the bore–hole is dominated by granodiorite. Silicification zones are found in sections 18.2 ~ 21.6m, 24.2 ~ 25.3m and 31.9 ~ 32.75m; the last section has a mineralized zone assaying Au 6.7g/t, Ag <0.1g/t, Pb 0.39%, Zn 0.10% and Cu 0.02%.

The dikes, under the microscope, are of holocrystalline, porphyritic texture and the phenocryst is mainly plagioclase accompanied by augite, associated with anhedral quartz and orthoclase which has micrographic texture. The groundmass is composed of lath-shaped plagioclase, augite and aggregates of chlorite-sericite-opaque minerals or of epidote-sericite-opaque minerals presumably altered from mafic minerals, which corresponds with that of diorite porphyrite. The quartz vein(sample 27-25.3), under the microscope, is composed mainly of quartz and rhodochrosite, associated with sericite, clay minerals and opaque minerals of minute quantities. Opaque minerals almost completely alter into limonite, in which cerussite is observed in a minute quantity.

### 3.2.14 MJMT-28

Granodiorite occurs over the full length of borehole. Silicified zones are observed in sections 8.6 ~ 11.0m, 19.5 ~ 20.6m, 24.6 ~ 26.9m and 28.0 ~ 30.1m. As the result of analysis, a mineralized zone is found in the 0.65m section(26.1 ~ 26.75m) which assays Au 2.3g/t, Ag 17.9g/t, Pb 3.24%, Zn 0.18% and Cu 0.14%.

## Chapter 4. Ore Reserve Calculation

### 4.1 Continuity of Orebody

Although there remain some doubts as to continuity of the mineralized zones captured by the 1995 survey, the mineralized zones are assumed to be connected with each other, on the following suppositions:

1) The orebodies occur controlled by faults and fissures; therefore, mineralized zones occur in parallel with confirmed/presumed fissures.

2) The mineralized zones are assumed to continue vertically almost at a certain dip, an average range of which is assumed to be 70° ~ 80°. In case a mineralized zone captured at an intermediate level of 775 ~ 780m above sea level by inclined drillings from the surface is located out of the mentioned average dip, it is to be treated as an echelon vein.

Surface drilling profiles prepared on the basis of these assumptions are shown in Fig.5, PL-6.

### 4.2 Proved Ore Reserve

One of the features of 1995 survey was its target having been set at the level corresponding to the maximum height of a probable ore block. Since the horizontal continuity of ore deposits has been calculated at 37m on average, based on the preceding year's revision of ore reserve calculation criteria, the vertical continuity can also be assumed to be 37m; and, if the ore deposit area can be delimited at the maximum height of a probable ore block, it will enable a block of near proved ore to be assumed as one and the same orebody.

For mineralized zones captured at the intermediate level by the 1995 survey, which are assumed to continue to the mineralized zones confirmed by the surface trenching or drilling at 750m level, proved ore reserve is calculated in accordance with the following formula :

Calculation of volume : (top area + base area) × 1/2 × height
Grade of a block : weighted average of grades of areas
Specific gravity : 3.1

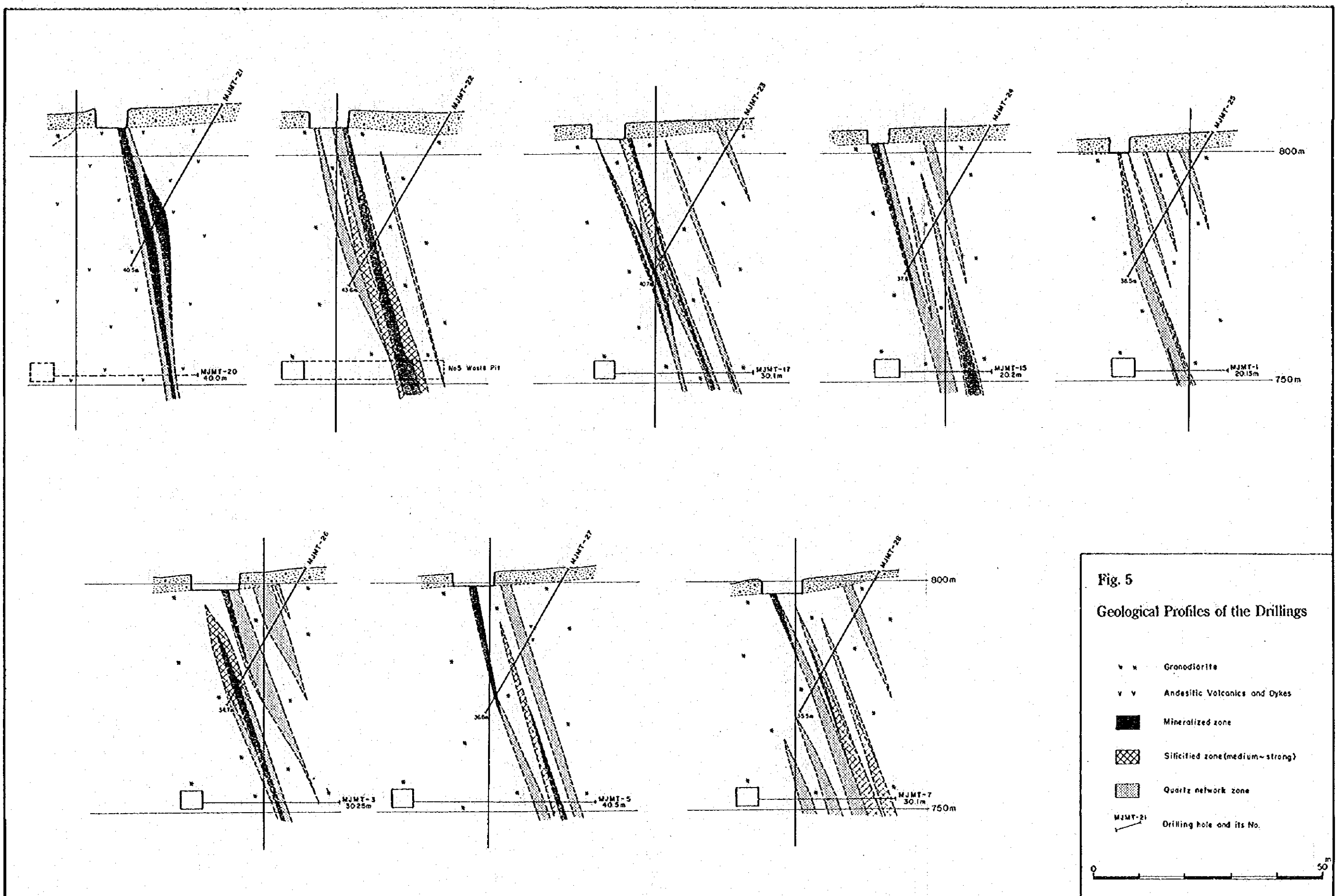



Fig. 5

Geological Profiles of the Drillings

### 4.3 Probable and Possible Ore Reserve

The preceding year's criteria are maintained except for specific gravity which is changed to 3.1. According to this criteria, mineralized zones captured by one single drill are to be excluded from the ore reserve calculation, which applies to the following sections explored during the 1995 survey:

| No. of Hole | Depth (m) | Sample length (m) | Au g/t | Ag g/t | Pb % | Zn % | Cu % |
|---|---|---|---|---|---|---|---|
| MJMT-15 | 14.1~16.8 | 2.7 | 0.3 | 90.2 | 8.58 | 5.09 | 0.10 |
| MJMT-17 | 25.0~25.3 | 0.3 | 0.3 | 2.3 | 0.62 | 4.56 | 0.06 |
| MJMT-21 | 26.3~30.3 | 4.0 | 2.1 | 0.29 | 1.81 | 1.34 | 0.18 |
| MJMT-23 | 38.0~39.1 | 1.1 | 0.2 | 16.2 | 1.39 | 0.57 | 0.14 |

### 4.4 Results of Ore Reserve Calculation

A summary of this year's ore reserve calculation is shown in Table 10, detailed calculations of respective ore blocks and ore block profiles in Apx.18. In this year's ore reserve calculation, the specific gravity of 3.1 measured as a part of the pre-feasibility study was applied. Consequently, the reserves totaling proved, probable and possible ores came to 1,544,627 t, an increase of 67,886 t from the preceding year's 1,476,742 t.

| Vein | | Area ($m^2$) | Width (m) | Hight (m) | Volume ($m^3$) | S.F. (%) | 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 proved ore | | 545.5 | 1.2 | 26.4 | 14,362.5 | 100 | 3.1 | 44,524 | 1.96 | 85.37 | 6.06 | 1.93 | 0.19 | 87,362.26 | 3,801.03 | 2,700.08 | 860.50 | 83.27 |
| Vein | | Area ($m^2$) | Width (m) | Hight (m) | Volume ($m^3$) | S.F. (%) | S.G. | Reserve (t) | Au (g/t) | Ag (g/t) | Pb (%) | Zn (%) | Cu (%) | Au (g) | Ag (kg) | Pb (t) | Zn (t) | Cu (t) |
| No.4 | trench | 237 | 0.8 | 22.8 | 2,696.8 | 100 | 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 | 165 | 0.9 | 20.3 | 1,675.5 | 100 | 3.1 | 5,194 | 1.56 | 138.46 | 6.07 | 4.41 | 0.23 | 8,089.96 | 719.16 | 315.52 | 229.26 | 11.76 |
| | 750m lower | 399 | 1.2 | 25.8 | 5,140.0 | 100 | 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 |
| | 630m | 559 | 1.0 | 55.5 | 15,511.5 | 100 | 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 | 149.57 |
| | total | 1,360 | 1.0 | 36.8 | 25,023.8 | 100 | 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 | | 252 | 0.9 | 28.6 | 3,604.5 | 100 | 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 | | 135 | 1.2 | 54.3 | 3,663.0 | 100 | 3.1 | 11,355 | 0.07 | 637.81 | 1.49 | 3.98 | 0.03 | 840.27 | 7,242.57 | 169.05 | 452.45 | 2.86 |
| Probable ore | | 1,747 | 1.0 | 37.0 | 32,291.3 | 100 | 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 |
| Proved・probable total | | 2,293 | 1.0 | 40.7 | 46,653.8 | 100 | 3.1 | 144,627 | 1.50 | 187.22 | 6.43 | 3.26 | 0.23 | 217,213.74 | 27,076.57 | 9,294.25 | 4,717.85 | 326.06 |
| Vein | | Area ($m^2$) | Width (m) | Hight (m) | Volume ($m^3$) | S.F. (%) | S.G. | Reserve (t) | Au (g/t) | Ag (g/t) | Pb (%) | Zn (%) | Cu (%) | Au (g) | Ag (kg) | Pb (t) | Zn (t) | Cu (t) |
| No.1 | No.1 | 122,460 | 0.47 | | 57,733.1 | 45 | 3.1 | 80,538 | | 206 | 5.37 | 8.39 | — | | 16,572.02 | 4,324.50 | 6,753.40 | |
| | No.1A | 175,510 | 0.40 | | 70,431.3 | 45 | 3.1 | 98,252 | | 141 | 4.74 | 6.18 | | | 13,835.91 | 4,652.45 | 6,072.44 | |
| | No.1B | 45,060 | 0.34 | | 15,271.3 | 45 | 3.1 | 21,303 | | 111 | 5.29 | 2.55 | | | 2,356.16 | 1,127.32 | 544.12 | |
| | total | 343,030 | 0.42 | | 143,435.7 | 45 | 3.1 | 200,093 | | 164 | 5.05 | 6.68 | | | 32,764.09 | 10,104.27 | 13,369.96 | |
| No.2 | No.2 | 184,920 | 0.94 | | 173,359.6 | 45 | 3.1 | 241,837 | | 267 | 8.78 | 2.37 | | | 64,619.52 | 21,230.36 | 5,723.04 | |
| | No.2A | 29,970 | 0.74 | | 22,094.5 | 45 | 3.1 | 30,822 | | 131 | 3.07 | 4.29 | | | 4,046.86 | 947.25 | 1,320.84 | |
| | No.2B | 27,500 | 0.56 | | 15,403.0 | 45 | 3.1 | 21,487 | | 107 | 5.63 | 0.89 | | | 2,304.19 | 1,209.26 | 191.82 | |
| | No.2HW | 23,660 | 0.92 | | 21,767.2 | 45 | 3.1 | 30,365 | | 129 | 6.99 | 4.66 | | | 3,917.12 | 2,122.53 | 1,415.02 | |
| | total | 266,050 | 0.87 | | 232,624.3 | 45 | 3.1 | 324,511 | | 231 | 7.86 | 2.67 | | | 74,887.69 | 25,509.40 | 8,650.73 | |
| No.4 | 630m upper | 55,585 | 0.71 | | 39,666.1 | 65 | 3.1 | 79,927 | | 186 | 8.00 | 5.55 | | | 14,889.81 | 6,397.03 | 4,438.02 | |
| | 630m lower | 56,670 | 1.35 | | 76,278.2 | 45 | 3.1 | 106,408 | | 154 | 5.33 | 4.86 | | | 16,398.73 | 5,674.01 | 5,173.09 | |
| | North | 84,360 | 0.51 | | 42,803.7 | 45 | 3.1 | 59,711 | | 75 | 4.63 | 1.91 | | | 4,461.87 | 2,767.19 | 1,140.10 | |
| | total | 196,615 | 0.81 | | 158,748.0 | 50 | 3.1 | 246,046 | | 145 | 6.03 | 4.37 | | | 35,750.41 | 14,838.23 | 10,751.22 | |
| No.4 total | | 265,850 | 0.40 | | 107,201.5 | 45 | 3.1 | 149,546 | | 295 | 9.17 | 3.96 | | | 44,188.73 | 13,717.46 | 5,924.77 | |
| No.6 total | | 178,900 | 0.91 | | 163,572.8 | 45 | 3.1 | 228,184 | | 207 | 7.69 | 3.98 | | | 47,222.35 | 17,556.28 | 9,072.06 | |
| No.8 | No.8 | 151,600 | 0.94 | | 142,736.2 | 49 | 3.1 | 215,020 | | 488 | 4.93 | 3.64 | | | 104,871.32 | 10,598.88 | 7,819.35 | |
| | No.8A | 27,060 | 0.37 | | 10,106.7 | 47 | 3.1 | 14,706 | | 398 | 6.73 | 2.94 | | | 5,847.80 | 990.24 | 432.72 | |
| | No.8FW | 26,440 | 0.35 | | 9,254.0 | 45 | 3.1 | 12,909 | | 1708 | 18.51 | 6.64 | | | 22,049.14 | 2,389.52 | 857.18 | |
| No.8 total | | 205,100 | 0.79 | | 162,096.9 | 48 | 3.1 | 242,635 | | 547 | 5.76 | 3.75 | | | 132,768.26 | 13,978.63 | 9,109.24 | |
| No.10 total | | 22,710 | 0.28 | | 6,440.5 | 45 | 3.1 | 8,984 | | 1306 | 4.81 | 2.98 | | | 11,733.72 | 432.46 | 267.73 | |
| Possible ore | | 1,478,255 | 0.66 | | 974,119.7 | 46 | 3.1 | 1,400,000 | | 271 | 6.87 | 4.08 | | | 379,365.25 | 96,136.73 | 57,145.71 | |
| Proved・probable・possibele total | | 1,480,548 | 0.69 | | 1,020,773.5 | 49 | 3.1 | 1,544,627 | | 263 | 6.84 | 4.01 | | | 406,441.82 | 105,430.98 | 61,863.56 | |

Table 10 Ore reserves in this year

Chapter 5. Summary of Survey Results

## 5.1 Geology

Geological units confirmed by the Survey which consisted of the surface trenching survey, surface drilling survey and the tunneling and drilling survey at 750m level were the upper Proterozoic metamorphic rocks, the middle ~ upper Jurassic pyroclastic rocks and the upper Jurassic granite, porphyry granite and granodiorite which intrude into the former two. Metamorphic rocks were confirmed at 3 trenches at No.10 vein and at 2 trenches in the south of No.1 vein. At the 3 trenches of No.10 vein, they are composed of relatively massive semischist of pyroclastic rock origin whilst green schist of relatively clear schistosity(NS trends, 40° ~ 50° W) prevails at the 2 trenches of No.1 vein. Pyroclastic rocks were confirmed at 4 trenches in the center and 3 trenches in the south of No.2 vein, the northern side of detailed survey area and 2 trenches in the southern part of the summary survey area of No.4 vein, 5 trenches of No.6 vein, and 9 trenches of No.8 vein. In general, pyroclastic rocks comprize mainly of andesitic tuff and lapilli tuff, occasionally accompanied by lava, tuff breccia and welded tuff. Welded tuff confirmed at the detailed survey area of No.4 vein has strike of NW-SE trends, where there is a gentle anticlinal structure with its northeastern side dipping northeastward while the southwestern side dipping southwestward.

Granite porphyry was observed at the northern tip of No.1 vein, of which coarse-grained quartz phenocryst is characteristic. Granite is confirmed at 2 trenches in the north of No.2 vein, which is light pink-colored, medium-grained, equigranular rock. Granodiorite was confirmed in the north of No.1 vein, central south of No.2 vein, south of the detailed survey area of No.4 vein, and south of No.8 vein. The surface drilling, as well as tunneling and drilling at 750m level, also confirmed occurrence of granodiorite. Generally, granodiorite is a medium-grained, equigranular rock. Under the microscope, it is composed of plagioclase, quartz, orthoclase and mafic minerals; orthoclase is partially of micrographic texture. Plagioclase and mafic minerals alter to sericite and, occasionally, to carbonate minerals, chlorite and epidote so completely that they can only be identified by the crystal system and relics of twins.

The surface drillings confirmed tonalite-diorite porphyry dikes intruding into granodiorite. Tonalite is of holocrystalline, seriate texture, composed mainly of plagioclase and quartz and also of mafic minerals replaced by chlorite, epidote, opaque and occasionally carbonate minerals.

Diorite porphyry is of holocrystalline, porphyritic texture, having plagioclase and augite as the phenocryst and is composed of lath-shaped plagioclase, augite and mafic minerals replaced by chlorite, sericite, epidote and opaque minerals. For these dyke rocks occurring in a mineralization area, their alteration is weak. Some of the surface and underground drilling confirmed Jurassic pyroclastic rocks which, under the microscope, are andesitic tuff ~ lapilli tuff containing volcanic glass in abundance.

### 5.2 Alteration

Megascopically, medium-grade chloritization prevails, which is increasingly overlapped by argillization as it comes nearer to either gossan zones, quartz veins or quartz veinlets ~ network vein zones. As argillization strengthens, chloritization diminishes. Argillization zones are generally several meters in size. Silicification is often accompanied by quartz veinlets. Many of weak to medium-grade silicification zones are generally overlapped by argillization zones. It is frequently observed that, on the surface, silicification zones gradually change into gossan zones, or they are accompanied by sulphide minerals of copper, lead and zinc. Most of silicification zones are under 1 m in size.

Under the microscope, sericitization is salient. As far as thin sections are observed, plagioclase and mafic minerals are entirely replaced by sericite, with no original minerals left. Silicification is characterized with recrystallization of quartz grains and also with minute veins of 0.1mm or less in width which are megascopically identified with a network vein. The recristallized quartz grains have been described as seriate texture but some quartz grains show wavy extinction, which suggests a possibility that there were at least two stages of silicification. Worthy of special mention in the microscopic observation is carbonatization which is unobservable megascopically. Under the microscope, relatively strong carbonatization and thin veins composed of carbonate minerals are observed, which are presumably carbonates of some metals heavier than Mg, considering the fact that an on-the-spot test with dilute hydrochloric acid caused no foaming. Since examination of silicified veins by a reflecting microscope has detected a large amount of manganese oxide minerals and the interim report on the mineral processing test refers to rhodochrosite, the carbonate minerals are inferred to be rhodochrosite.

Although uncaptured as a large-scale alteration zone, occurrence of tourmaline was confirmed by the preceding year's survey while 1995 survey confirmed fluorite, though small in quantity, accompanying quartz veins. This indicates the possibility of mineralization taking place over a substantially long range of time.

## 5.3 Mineralization

The six mineralized zones in the survey area, ie., the veins Nos. 1, 2, 4, 6, 8 and 10 are referred to in the past records. An overview of the six mineralized zones indicates that observed as ore minerals are sulphide minerals such as galena, chalcopyrite, tetrahedrite, pyrite, as well as oxide minerals including cerussite, green copper, limonite and oxides of manganese. These minerals are usually observed at gossan zones, quartz veins, strongly silicified zones and at quartz veinlets ~ network vein zones, which form mineralized zones. Some of such mineralized zones have a width of several meters while most of them are as small as only several decimeters. The form of occurrence is controlled by fissure systems.

Under the microscope, observed are polybasite, argentite, electrum, native silver, chalcocite, covelline, pyrrhotite and marcasite, in addition to those megascopically identified. Sphalerite is distinguished into two types: one including microcrystal of chalcopyrite and the other not including them, the former being dominant. Blackish brown-colored and light brown-colored sphalerite can be discriminated megascopically, of which the former is dominant; therefore, blackish brown-colored sphalerite is presumed to correspond to that including chalcopyrite. Galena tends to be replaced by cerussite, which indicates that supergene alteration extends to 60m below the surface. Some chalcopyrite occurs independently in gangue minerals or in association with galena or sphalerite, whereas the other occurs as dot-like inclusion in sphalerite. Tetrahedrite is often associated with chalcopyrite but, occasionally, it is covered by film-like sphalerite. Argentite is in some cases associated with chalcopyrite or covelline, surrounding sphalerite. Polybasite occurs in galena. Electrum occurs either independently in gangue minerals or along cracks in sphalerite.

In the light of an overall interpretation of the occurrence of these ore minerals and assay result fluctuations, the process of mineralization in the survey area may be summarized as follows:

1) Polymetallic mineralization which occurs in quartz veins and contains noble metals.
2) Auri-argentiferous mineralization accompanied by quartz veinlets ~ network veins. Argentiferous minerals and tetrahedrite occur in argentiferous mineralization.
3) Base-metallic mineralization accompanied by carbonatization (disseminated mineralization).
4) Pb and Cu enrichment by supergene alteration(cerussite, covelline, chalcocite).

5.4 Ore Reserves

Changes in the ore reserves from the preceding year are detailed in the following table:

| 1. Increase/decrease due to ore reserve determination | Increase (t) | Decrease (t) | Balance (t) |
|---|---|---|---|
| Addition of 7 new proved blocks | 44,524 | | |
| Addition of 4 new probable blocks | 4,604 | | |
| Cancellation of 5 probable blocks | | 23,261 | |
| Revision of height of 1 probable blocks | | 315 | |
| Revision of area of 1 possible block | | 5,908 | |
| Sub-total | 49,128 | 29,484 | 19,644 |
| 2. Increase/decrease due to revision of specific gravity(SG) | SG=3.1 | SG=3.0 | Difference |
| Probable ore(t) | 95,499 | 92,421 | 3,078 |
| Possible ore(t) | 1,400,000 | 1,354,837 | 45,163 |
| Sub-total | 1,495,499 | 1,447,258 | 48,241 |
| TOTAL | 1,544,627 | 1,476,742 | 67,885 |

As regards the grade of ore reserves, only the proved and probable ores are referred to in the following paragraphs. [No mention is given to possible ores, of which only the specific gravity was revised excepting an ore block at the No.4 vein that underwent revision of the area, since comparison with the preceding year is insignificant.]

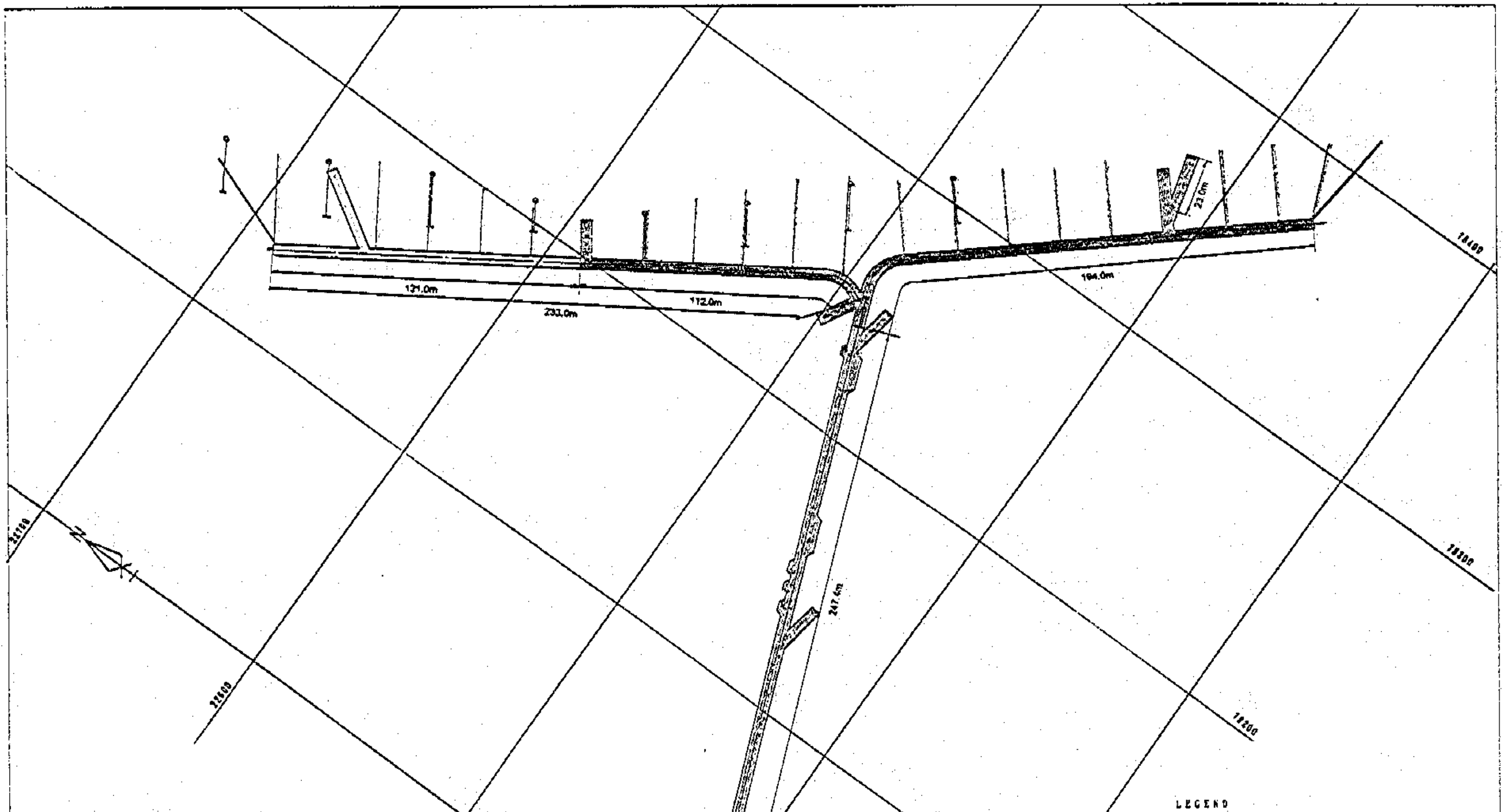
This year's (1995) grades of the total reserves of proved and probable ores come to Au 1.50g/t, Ag 187.22g/t, Pb 6.43%, Zn 3.26% and Cu 0.23%, respectively representing decreases/increases from those of the preceding year of Au (−)0.1.g/t, Ag (−)27.16g/t, Pb (−)0.27%, Cu (−)0.02% and Zn (+)0.07%. The same comparison, limited only to the No.4 vein, is shown below:

| Year | Ore Reserves t | Au g/t | Ag g/t | Pb % | Zn % | Cu % |
|---|---|---|---|---|---|---|
| 1995 | 122,098 | 1.72 | 148.77 | 6.54 | 3.41 | 0.25 |
| 1994 | 94,194 | 1.90 | 172.44 | 6.98 | 3.36 | 0.29 |
| Decrease Increase | +27,904 | △0.18 | △23.67 | △0.44 | +0.05 | △0.04 |

The above comparison may be interpreted to imply a tendency that, as exploration comes down to a lower portion, Au, Ag, Pb and Cu decline in grades whilst Zn grade goes up.

### 5.5 Drifting and Diamond Drilling

Drifting and Diamond Drilling Survey quantity and locations(from 1993 ~ 1995) are shown in Fig.6.




Result in Tsav

| | 1993 | 1994 | 1995 | Total |
|---|---|---|---|---|
| Inclined Shaft | 247.4m | | | 247.4m |
| Drift | | 329.0m | 121.0m | 450.0m |
| Waste Pit | 31.5m | 39.0m | 33.1m | 103.6m |
| Drainage Pond | 15.0m * | | | 15.0m * |
| Pump Station | | 15.0m | | 15.0m |
| Transform Station | 15.0m * | 30.0m * | | 45.0m * |
| Depository | 130.0m * | | | 130.0m * |
| Core Boring | | | | |
| Underground | | 447.4m (14pcs) | 186.5m (6pcs) | 633.9m (20pcs) |
| Surface | | | 305.7m (6pcs) | 305.7m (6pcs) |

| | | | |
|---|---|---|---|
| Title | Fig. 6 Result in Tsav | | |
| Scale | 1:1,000 | Date | |
| Drawing No. | | | |

# Ⅲ. Mineral Dressing Test

## Part Ⅲ. Mineral Dressing Test

### Chapter 1. Purpose of Test

In order to select the optimum mineral dressing process for the Tsav ore, mineral dressing test by the flotation process was conducted to recover lead and zinc concentrates. For preparing test samples, representative run–of–mine ores from the Tsav deposits were so adjusted that they might be of the expected grades of mill feed.

### Chapter 2. Outline of Test Conducted

#### 2.1 Sample adjustment

Sample ores and waste were collected in August, 1995 respectively from and around the three veins exposed on the right side wall of the crosscut of the Tsav No.4 vein. These samples were individually crushed by a sample breaker and a sample grinder to −28 mesh for assaying lead and zinc. In an effort to approximate their grades to the expected grades of mill head, the ore samples were mixed as indicated in Table 11, which were fed to the mineral dressing test as sample mill feed.

| Samples | Ore-1 | Ore-2 | Ore-3 | Waste-1 | Waste-2 | Waste-3 | Mill head | Expected grade of mill head |
|---|---|---|---|---|---|---|---|---|
| Mixing ratio(%) | 9.94 | 6.25 | 6.53 | 10.51 | 40.35 | 26.42 | | |
| Pb(%) | 25.66 | 38.04 | 30.11 | 0.88 | 0.09 | 0.10 | 7.05 | 6.4 |
| Zn(%) | 24.16 | 6.08 | 0.94 | 1.38 | 0.16 | 0.13 | 3.09 | 2.9 |

Table 11 Mixing ratio of Mineral Dressing Test Samples

#### 2.2 Physical Properties and Assay of Mill Feed

Of the testing samples, emission spectroscopic analysis and chemical analysis of main components were carried out to acquire basic knowledge for the mineral dressing test, as well as measurement of physical properties such as the absolute specific gravity and work index. In addition, microscopic observation of ores and mineral dressing test products were done in order to study occurrence of each minerals.

### 2.2.1 Emission spectroscopic analysis

Results of the emission spectroscopic analysis of the mill feed samples are shown in Table 12, which may be summarized as follows:

(1) Al, Fe, Pb, Si and Zn are abundant.
(2) Ca, Cu, Mg, Mn, and Ti are found in small amounts.
(3) Ba and Cr are rare.
(4) Ag, As, Pb and Zr are extremely rare.
(5) No other components are detected.

| Elements | Intensity | Elements | Intensity | Elements | Intensity |
|---|---|---|---|---|---|
| Ag | • | Cr | △ | Rb | • |
| Al | ◎ | Cu | ○ | Si | ◎ |
| As | • | Fe | ◎ | Ti | ○ |
| Ba | △ | Mg | ○ | Zn | ◎ |
| Ca | ○ | Mn | ○ | Zr | • |
| Cd | • | Pb | ◎ | | |

Remarks: ◎:Abundant ○:A little △:Rare •: Extremely rare

Table 12 Emission Spectroscopic Analysis of Mill Head

### 2.2.2 Chemical analysis of main component

Based on the results of the emission spectroscopic analysis which were used as reference for determining elements for assay, the chemical analysis was conducted of main components. The fluorescent X-ray spectroscopy was also conducted, which recognized Cl, F, K, Na, P, etc., simultaneously verifying the results of the chemical analysis of main components.

Results of the chemical analysis and fluorescent X-ray spectroscopy are shown in Tables 13 and 14, respectively, which may be summarized as follows:

(1) As valuable metal elements, Pb and Zn are abundant. Ag grade is as high as 139g/t while Au is also contained.

(2) Cu grade is as low as 0.15%.

(3) The rare components(Cd, Ba, Cr, etc.) are contained in the order of 0.0X%.

(4) As content is in the order of ppm.

(5) Components of rock minerals are mainly $SiO_2$ and $Al_2O_3$, which are accompanied by $K_2O$, MgO, CaO and $Na_2O$. Besides, Mn, Ti, F, P, etc. are also recognized.

| Element | Grade | Element | Grade | Element | Grade |
|---|---|---|---|---|---|
| Ag | 139 g/t | Cr | 0.01 % | S | 5.23 % |
| Al | 5.6 % | Cu | 0.15 % | Si | 24.4 % |
| As | 83 ppm | Fe | 4.64 % | Sn | 5 ppm |
| Au | 0.7 g/t | Mg | 0.42 % | Ti | 0.28 % |
| Bi | 29 ppm | Mn | 1.94 % | V | 35 ppm |
| Ca | 0.46 % | Mo | 4 ppm | Zn | 3.31 % |
| Cd | 0.02 % | Pb | 8.36 % | Zr | 56 ppm |

Table 13 Chemical Analysis of Main Components of Mill Feed

| Element | Grade | Element | Grade | Element | Grade |
|---|---|---|---|---|---|
| $Ag_2O$ | 0.0035 % | F | 0.291 % | PbO | 4.0 % |
| $Al_2O_3$ | 18.7 % | $Fe_2O_3$ | 5.16 % | $Rb_2O$ | 0.0286 % |
| $As_2O_3$ | 0.0042 % | $In_2O_3$ | 0.0044 % | $SO_3$ | 5.2 % |
| BaO | 0.0207 % | $K_2O$ | 4.69 % | $Sb_2O_3$ | 0.0079 % |
| CaO | 0.891 % | MgO | 1.21 % | $SiO_2$ | 52.7 % |
| CdO | 0.0225 % | MnO | 2.51 % | $TiO_2$ | 0.59 % |
| Cl | 0.0214 % | $Na_2O$ | 0.608 % | $V_2O_5$ | 0.0056 % |
| CuO | 0.187 % | NiO | 0.0052 % | ZnO | 2.84 % |
| $Cr_2O_3$ | 0.0232 % | $P_2O_5$ | 0.252 % | $ZrO_2$ | 0.0137 % |

Table 14 Fluorescent X-ray Spectroscopy on Mill Feed

### 2.2.3 Absolute specific gravity

Specific gravity of the sample mill feed, as well as lead concentrate, zinc concentrate and tailing obtained under the best testing conditions were measured by the pycnometer, results of which are shown in Table 15. The specific gravity of the mill feed, Pb concentrate, Zn concentrate and tailing were 3.1, 6.9, 4.1 and 2.7, respectively.

| Mesurement | Mill Head | Pb Conc. | Zn Conc. | Tailing |
|---|---|---|---|---|
| No. 1 | 3.301 | 6.785 | 4.028 | 2.678 |
| No. 2 | 2.968 | 6.991 | 3.984 | 2.743 |
| No. 3 | 3.042 | 6.875 | 4.321 | 2.704 |
| Average | 3.10 | 6.88 | 4.11 | 2.71 |

Table 15 Absolute Specific Gravity of Mill Head, Lead Concentrate, Zinc Concentrate and Tailing

### 2.2.4 Work index

Crushing work index(Wi) of each samples was measured by the Hardgrove tester. Measurement was repeated three times to obtain average indices. Six kinds of samples were measured, ie., mill feed, three kinds of ores and two kinds of waste. Measurement results are shown in Table 16.

Work indices were 11 to 14 for the waste and 8 for the ores, showing substantial difference. The index of the mill feed was similar to the results obtained in 1993, which is presumed to be 10 to 11, indicating that it is easy to crush.

| Samples | +200mesh(g) | -200mesh(g) | H.G.I. | Wi (kWh/st) |
|---|---|---|---|---|
| Ore-1 | 40.10 | 9.90 | 81.607 | 7.92 |
| Ore-2 | 39.84 | 10.16 | 83.409 | 7.77 |
| Ore-3 | 41.16 | 8.84 | 74.261 | 8.63 |
| Mill Head | 43.30 | 6.70 | 59.431 | 10.57 |
| Waste-2 | 43.74 | 6.26 | 56.832 | 11.01 |
| Waste-3 | 45.54 | 4.46 | 43.908 | 13.92 |

Table 16 Work Indices

### 2.2.5 Microscopic observation

(1) Samples used and methods of study

To study flotation behavior of copper, lead and zinc minerals, polished sections of sample ores, mill feed and mineral dressing products were prepared and microscopic observation of occurrence of each minerals was done. For a part of the sample products, X-ray diffractive analysis, EPMA analysis and chemical analysis of non-sulfide minerals were conducted in an effort to complement the microscopic observation.

(2) Results of microscopic observation

Results of the microscopic observation of each sample are described in the following paragraphs.(Ref. Apx-19 List of Microscopic Observation Results, Apx-20 Microscopic photographs and Apx-21 EPMA analysis)

Ore-1 : This ore is composed of abundant sphalerite and galena, a little chalcopyrite, rare chalcocite, covelline, tetrahedrite and pyrite. Sphalerite is 1cm or less in size, euhedral and, partially, has dot-like inclusions of abundant chalcopyrite.

Galena is 1.5cm or less in size, massive or euhedral, and, on rare occasions, includes dot-like sphalerite.

Chalcopyrite, 0.02mm or less in size, is dot-like or semi-euhedral and frequently occurs as inclusions in sphalerite or crystallizes in spots at boundaries between sphalerite and galena. A part of chalcopyrite is replaced by chalcocite and covelline, starting from the periphery.

Chalcocite and covelline penetrate, in hair-like fine veins(with a width of 0.01mm or less), into sphalerite crystals. Tetrahedrite is dispersed in galena in 0.005 ~ 0.03mm dots.

Pyrite, 0.01 ~ 0.02mm in size, is euhedral and occurs abundantly in rock minerals. Goethite occurs in film-like crystallization in cracks of sphalerite crystals and at boundaries with rock minerals.

Ore−2 : This ore is composed of abundant sphalerite and galena, a little chalcopyrite and pyrite, and rare chalcocite, covelline, tetrahedrite and electrum.

Sphalerite, 0.2 ~ 0.5mm in size, is euhedral and has banded cracks(0.02mm in width). Chalcopyrite and galena penetrate into the cracks or occur in dot−like inclusions.

Galena, 0.2mm or larger in size, is euhedral and penetrates in amoebic forms into the sphalerite side. Galena crystals has no other inclusions than rare sphalerite.

Chalcopyrite, 0.001 ~ 2mm in size, is anhedral and penetrates into sphalerite in dots or spots. Chalcocite replaces a part of chalcopyrite while crystallizing along cracks in sphalerite. Some chalcocite partially replaced by covelline is recognized. Tetrahedrite, 0.005 ~ 0.03mm in size, occurs in galena and sphalerite in veins.

Pyrite 0.03 ~ 0.1mm in size, is euhedral and included in sphalerite and chalcopyrite.

As to electrum, a grain of 0.015mm in size in rock mineral and two grains of 0.05mm associated with chalcopyrite and sphalerite were observed.

The paragenesis is presumed to be in the order of pyrite → Sphalerite → galena·chalcopyrite → chalcocite·covelline.

Ore−3 : Abundant galena, a little chalcopyrite and rare tetrahedrite, chalcocite and pyrite comprize the ore.

Galena, approximately 2mm in size, is euhedral, altered along the grain boundaries, recrystallized, replaced by cerussite and partially pseudomorphized.

Chalcopyrite, 0.03 ~ 0.2mm in size, is anhedral and penetrate, including pyrite, into galena. A part of chalcopyrite is associated with tetrahedrite. Tetrahedrite, 0.02 ~ 0.07mm in size, is anhedral and crystallizes in galena. Chalcocite replaces a part of chalcopyrite.

Pyrite, 0.03 ~ 0.06mm in size, is euhedral, included in galena.

Mill Feed : The grain sizes are 0.1 ~ 0.7mm, mostly 0.2mm or more. It is composed of abundant galena, a medium quantity of sphalerite and pyrite and a little chalcopyrite, chalcocite, covelline, tetrahedrite and goethite.

Galena, 0.1 ~ 0.3mm in size, is euhedral and easy to liberate.

Sphalerite is 0.2 ~ 0.7mm in size and its liberation grade is about 90%. Some of sphalerite includes chalcopyrite in dots.

Pyrite, 0.03 ~ 0.7mm in size, some 50% of which comprises free particles while the remaining 50% forms middling with rock minerals.

Chalcopyrite is included in sphalerite in dots, forming middling with chalcocite. Chalcocite, 0.1 ~ 0.2mm in size, forms middling which partially replaces chalcopyrite. Covelline and tetrahedrite forms middling with chalcocite and/or pyrite.

Goethite often forms middling with rock minerals.

Pb concentrate-1(Straight differential flotation-"SDF") :

The grain sizes are 0.03 ~ 0.2mm, comprising abundant galena, a little sphalerite, rare pyrite, tetrahedrite, chalcocite, covelline and electrum.

Galena is 0.02 ~ 0.2mm in size, of which some 90% are free particles while the balance is middling with cerussite, sphalerite and/or pyrite.

Sphalerite is 0.06 ~ 0.2mm in size, consisting half and half of free particles and middling mainly with cerussite and/or galena.

Pyrite, 0.03 ~ 0.6mm in size, comprises free particles(30%) and middling mainly with galena and/or chalcocite(70%).

Tetrahedrite forms middlings with galena. Chalcocite and covelline consist half and half of free particles and middling with chalcopyrite, sphalerite and/or pyrite.

Electrum is 0.01 ~ 0.02mm in size. A grain of free particle and two grains in sphalerite have been recognized.

Pb concentrate−2(bulk differential flotation−"BDF") :

The grains are 0.01 ~ 0.3mm in size, comprising extremely abundant galena, a little sphalerite, rare pyrite, chalcopyrite, tetrahedrite, chalcocite, covelline and electrum.

Galena is 0.01 ~ 0.3mm in size, comprising free particles(90%) and middling with cerussite and/or chalcopyrite(10%).

Sphalerite is 0.02 ~ 0.2mm in size, composed of free particles and middling at a ratio of 2 : 8. Middling is often covered by chalcocite, whilst some middling with galena is also recognized.

Pyrite is 0.02mm or less in size, composed of free particles(30%) and middling with galena and/or chalcocite(70%).

Chalcopyrite is 0.1mm or less in size, composed half and half of free particles and middling mainly with galena, sphalerite and/or chalcocite. Chalcocite and covelline are 0.03 ~ 0.2mm in size, of which free particles account for some 60% whilst the balance are middling with chalcopyrite, sphalerite and/or pyrite. Tetrahedrite forms middling with galena.

An electrum grain of 0.03mm in size has been found.

Zn concentrate−1(SDF) :

The grains are 0.01 ~ 0.2mm in size, composed of abundant sphalerite, a little pyrite, galena and rare chalcopyrite.

Sphalerite is 0.03 ~ 0.2mm in size, in which free particles account for some 70% while the remaining portion mainly consists of those including chalcopyrite in dots and middling with galena.

Galena, 0.02 ~ 0.2mm in size, is composed of free particles(60%) and middling with sphalerite, chalcocite and/or rock minerals(40%).

Pyrite is 0.01 ~ 0.04mm in size, composed of free particles(80%) and middling(20%) mainly with galena and/or chalcocite.

Chalcopyrite is in grains of 0.01mm or less, composed half and half of free particles and middling with sphalerite and/or chalcocite, excepting a portion occurring in dots within sphalerite.

Zn concentrate－2(BDF) :

The grains are 0.02 ～ 0.7mm in size, composed of extremely abundant sphalerite, abundant pyrite, rare galena, chalcopyrite, chalcocite, covelline and electrum.

Sphalerite are 0.02 ～ 0.2mm in size, 80% of which is in the form of free particles. The remaining 20% portion comprises mainly of those which include dot－like chalcopyrite. In some middling, chalcocite intrudes along fine veins.

Pyrite is 0.05 ～ 0.7mm in size, mostly in free particles with rare middling with chalcocite.

Galena is 0.3mm or less in size, 75% of which are middling with sphalerite and/or rock minerals.

Chalcopyrite is 0.02 ～ 0.1mm in size, halved in free particles and middling mainly with sphalerite and/or rock minerals. Chalcocite and covelline are 0.04 ～ 0.06mm in size, 60% of which are middling with chalcopyrite, pyrite, sphalerite and/or galena.

A 0.02mm grain of electrum was observed in sphalerite which includes dot－like chalcopyrite.

Middling－1(Middling of Pb differential flotation in SDF) :

The grains are 0.07 ～ 0.3mm in size, composed of abundant sphalerite, middle pyrite and galena, a little chalcopyrite, rare chalcocite and rock minerals.

Sphalerite, 0.1 ～ 0.3mm in size, is composed of free particles(70%) and middling(30%), the latter being those which include dot－like chalcopyrite and which have fine veins filled with chalcocite.

Pyrite is halved in free particles and middling with chalcocite, tetrahedrite, sphalerite and/or galena.

Galena, 0.1 ~ 0.2mm in size, is halved in free particles and middling. Some of the middling is replaced by cerrussite at the periphery.

Chalcopyrite is 0.1 ~ 0.3mm in size. Free particles account for about 10% only while most of the grains are in the form of middling with chalcocite and/or sphalerite. Chalcocite is 0.1 ~ 0.2mm in size. There remains original chalcopyrite in many of the grains.

Rock minerals, 0.1 ~ 0.2mm in size, are mostly in free particles but, partially, in middling with pyrite and/or galena.

Middling-2(Middling of Zn differential flotation in SDF ) :

The grains are 0.03 ~ 0.3mm in size, composed of abundant sphalerite, middle pyrite, a little chalcopyrite, galena and rock minerals, and rare tetrahedrite and cerussite.

Sphalerite, 0.1 ~ 0.3mm in size, composed of free particles(80%) and middling(20%). The middling are halved in those which include dot-like pyrite and those with chalcocite.

Pyrite is 0.1 ~ 0.3mm in size, composed of free particles(90%) and middling(10%). Most of middling is formed with chalcocite intruding along fine fissures.

Chalcopyrite is 0.03 ~ 0.2mm in size and halved in free particles and middling. Some 60% of the middling is with sphalerite while the balance is with chalcocite and/or pyrite. Tetrahedrite, about 0.02mm in size, is either in the form of vein-like inclusion in chalcopyrite or of middling with pyrite.

Galena is 0.1 ~ 0.2mm in size, halved in free particles and middling. Most of the middling is replaced by cerussite.

Rock minerals are 0.1 ~ 0.3mm in size, composed of free particles(70%) and middling with pyrite(30%).

Compared with the middling−1, the middling−2 has the characteristics that it has lesser sphalerite middling with chalcocite and more abundant middling of cerussite and galena.

Middling−3(Middling of differential flotation in BDF) :

The grains are 0.03 ~ 0.3mm in size, composed of extremely abundant pyrite, a little sphalerite and rock minerals, and rare galena, chalcopyrite, chalcocite and tetrahedrite.

Pyrite is 0.03 ~ 0.3mm in size, composed of free particles(90%) and middling(10%) mainly with fissure−filling chalcocite(partially covelline).

Spalerite is 0.1 ~ 0.3mm in size, composed of free particles(90%) and middling(10%). The middling is halved in that with pyrite, galena, chalcopyrite and/or chalcocite and that which is coated by chalcocite around the periphery.

Galena is 0.06 ~ 0.2mm in size, halved in free particles and middling. The middling consists of those replaced by cerussite or chalcocite and those with sphalerite in the order of quantity.

Chalcopyrite and chalcocite are 0.3mm or less in size, mostly middling of these and have little free particles. Tetrahedrite, 0.01 ~ 0.02mm in size, is in the form of middling with pyrite and/or chalcopyrite, included in galena.

Rock minerals are 0.03 ~ 0.3mm in size and mainly in free particles.

Middling−4(Middling of Pb differential flotation − in BDF) :

The grains are 0.03 ~ 0.3mm in size, composed of extremely abundant pyrite, a little sphalerite and galena, and rare chalcopyrite, tetrahedrite, chalcocite and rock minerals.

Chalcocite is 0.03 ~ 0.3mm in size, of which the ratio between free particles and middling is 95:5. A half of the middling is with sphalerite and the rest with chalcocite.

Galena is 0.07 ~ 0.3mm in size, halved in free particles and middling. Some 60% of the middling are those with cerussite while the rest are those with sphalerite.

Chalcopyrite is 0.05 ~ 0.2mm in size, halved in free particles and middling with chalcocite. Tetrahedrite is 0.015 ~ 0.02mm in size, some crystallizing in association with chalcopyrite within sphalerite while the others being middling with chalcopyrite. Chalcocite is 0.02mm or less in size, forming middling with sphalerite, chalcopyrite and/or galena.

Rock minerals are 0.2 mm or less in size, 80% of which are free particles. The rest is middling mainly with pyrite.

Tailing-1(SDF) :

The grains are 0.06 ~ 0.3mm in size, composed of extremely abundant rock minerals, middle pyrite, rare galena, sphalerite and cerussite.

Galena is 0.1mm or less in size, 60 ~ 70% of which is in free particles. The rest is middling with pyrite and/or rock minerals. Cerussite is 0.09mm or less in size, mainly in the form of middling with rock minerals.

Sphalerite, 0.09 ~ 0.2mm in size, are composed of free particles(60-70%) and middling with pyrite and/or rock minerals.

Pyrite is 0.03 ~ 0.2mm in size, composed of free particles(90%) and middling with rock minerals(10%).

Sphalerite generally exceeds galena.

Tailing-2(BDF) :

The grains are 0.06 ~ 0.3mm in size, composed of extremely abundant rock minerals, middle pyrite, and rare galena, sphalerite and cerussite.

Pyrite is 0.06 ~ 0.3mm in size, 70% of which is middling with rock minerals.

Galena is 0.06mm or less in size, of which 90% is in free particles while the rest is middling with pyrite. Cerussite, 0.09 ~ 0.1mm in size, is in the form of middling with rock minerals.

Sphalerite is 0.09mm or less in size, of which 60 ~ 70% is in free particles and the rest is middling with pyrite and/or rock minerals.

Tailing-3(BDF) :

The grains are 0.005 ~ 0.3mm in size, composed of extremely abundant rock minerals, a little pyrite, and rare sphalerite, chalcopyrite, galena and cerussite.

Pyrite is 0.005 ~ 0.9mm in size, composed half and half of free particles and middling. The middling is mainly with rock minerals and/or sphalerite.

Sphalerite is 0.06 ~ 0.09mm in size, several grains of which are observed in the sample. 70% is accounted for by middling with rock minerals and/or dot-like chalcopyrite.

As for chalcopyrite, several grains of 0.01mm or less, mainly middling with rock minerals have been observed.

A grain of galena has also been observed, which forms middling with pyrite and rock minerals. Cerussite, 0.07 ~ 0.1mm in size, is in the form of middling with rock minerals.

EMPA Surface Analysis :

Under the microscope, a mineral presumed to be cerussite is observed. To confirm the mineral which replaces galena, EPMA surface analysis was done on the sample ore-3 which contains the mineral in abundance, resulting in detection of Pb and Cu contents. In the light of the EPMA findings and also the occurrence of the mineral under the microscope, the mineral is identified with cerussite, although it was not detected by the X-ray diffraction.

As described in the item of Pb concentrate-2, a texture of sphalerite grains encircled by film-like chalcocite was observed under the microscope, which was confirmed by the EPMA analysis. The analysis also revealed that Cu content does not penetrate into sphalerite grains.

Ag content was scarcely detected in the associating galena grains. The checking of Ag content in tetrahedrite by EPMA surface analysis verified substantial content, thereby confirming that it is argentiferous tetrahedrite.

Chemical analysis :

From the results of microscopic observation and EPMA surface analysis, occurrence of non-sulfide lead minerals had been presumed. To confirm it, chemical analysis of the sample mill feed, tailings-1 and -2 was conducted, resulting in the findings shown in Table 17. Most of the tailings are composed of non-sulfide minerals while sulfide components are minimal.

| | Mill Feed | Tailing-1 | Tailing-2 |
|---|---|---|---|
| T-Pb (%) | 9.1 | 0.81 | 0.97 |
| Non-sulfide Pb (%) | 1.2 | 0.77 | 0.79 |
| sulfide Pb (%) | 7.9 | 0.04 | 0.18 |

Table 17 Chemical Analysis for Non-sulfide Lead

(3) X-ray Diffraction

X-ray diffraction analysis was applied to the sample mill feed, tailings-1 and -2, results of which are shown in Table 18. (Ref. Apx-22 X-ray Diffraction Chart)

The analysis resulted in the following findings:

1) The ore minerals are sphalerite, galena and pyrite.
2) Main rock minerals are quartz, rhodocrocite, sericite and chlorite. Potassium feldspar is partially recognized, as well as rare plagioclase.
3) No other minerals were recognized.

| Mineral | Mill feed (flotation feed) | Waste-1 | Tailing-1 (straight differential) | Tailing-2 (bulk differential) |
|---|---|---|---|---|
| Quartz | ◎ | ◎ | ◎ | ◎ |
| K-feldspar | △ | △ | | △ |
| Plagioclase | · | · | · | · |
| Rhodochrosite | ○ | ○ | · | ○ |
| Sericite | △ | ○ | ○ | ○ |
| Chlorite | △ | △ | | △ |
| Galena | ○ | · | △ | △ |
| Sphalerite | ◎ | | △ | |
| Pyrite | △ | · | ○ | △ |

Remarks: ◎ :abundant ○ :middle △ :a little · :rare

Table 18 X-ray Diffraction Analysis

## 2.3 Flotation Test

### 2.3.1 Grinding test

In order to verify the grindability and the liberation grade of the mill feed sample, grinding test was conducted. Grinding products at 0, 3, 5, and 10 minutes were respectively sieved out into seven groups by grain size between +65 and −325 mesh. The seven products thus obtained were subjected to component analysis, on the basis of which the size distribution of valuable metals was calculated.

Fig.7 to 10 indicates the size distribution of weight and metal elements at respective grinding time. (Ref. Apx−23 Grinding Test Results)

The test findings follow:

1) Lead and zinc − the main component minerals − are in relatively coarse grains(+65 mesh; product quantity=approx.15%) and easily liberated, which agreed with the results of the test conducted in 1993.

2) In case of the 0 minute grinding sample, the metal distribution were 50% or more for the 65 mesh portion and approx. 20% for the −325 mesh portion, which generally is similar to the size distribution of weight, although there are some variances by kind of metals.

3) In case of the 10 minute grinding sample, a half of copper and lead were −325 mesh or less, that is overgrinding for lead.

4) Metallic minerals were more grindable than waste. Certain selectivity in the grindability is recognized; copper and lead are more grindable than zinc and iron.

5) Gold and silver showed size distribution similar to that of copper, lead and zinc; especially, silver and lead had strong similarity in the size distribution.

6) Copper originally exists in fine spaces, which shows that copper minerals are of fine grain sizes.

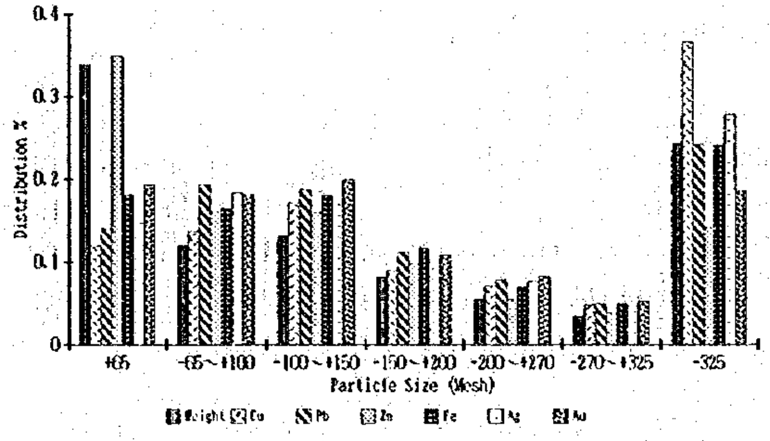



Fig.7 Size Distribution of Weight and Metal Elements
(Milling Time : 0 min)

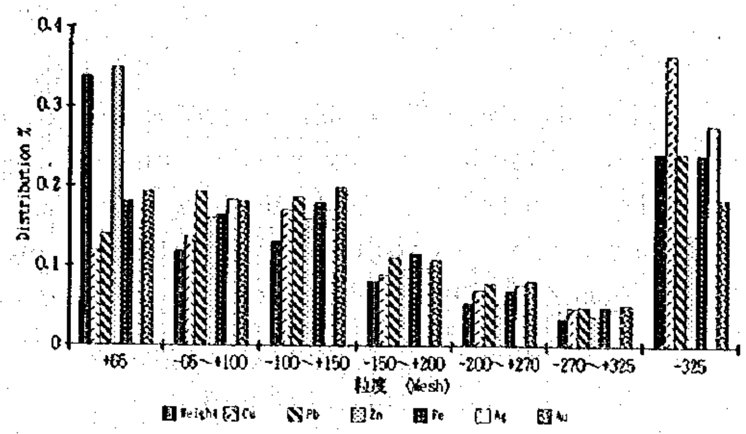



Fig.8 Size Distribution of Weight and Metal Elements
(Milling Time : 3 min)

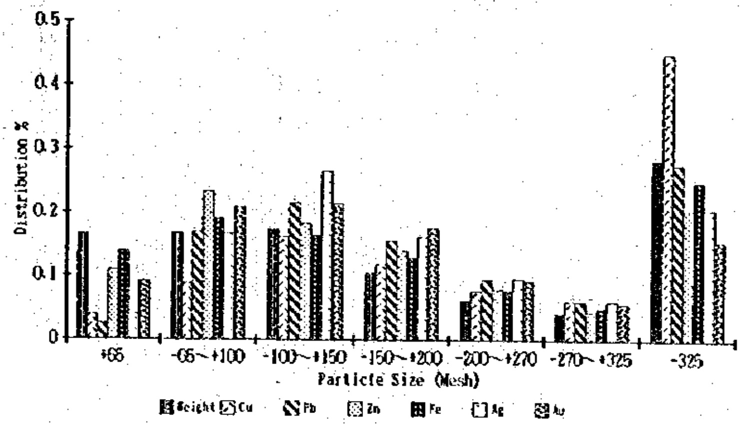



Fig.9 Size Distribution of Weight and Metal Elements
(Milling Time : 5 min)

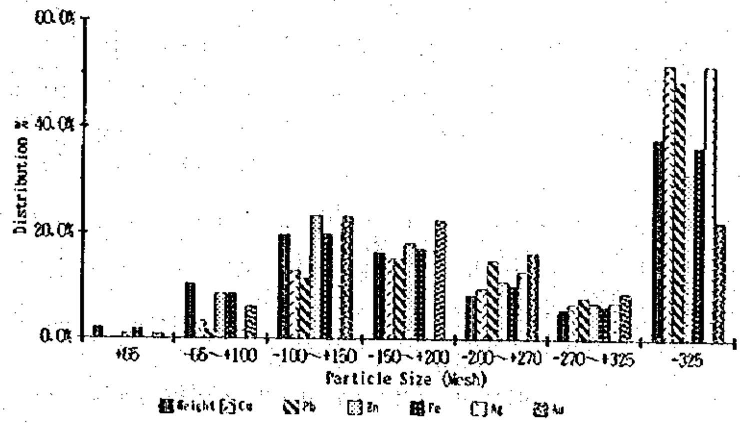



Fig.10 Size Distribution of Weight and Metal Elements
(Milling Time : 10 min)

### 2.3.2 Flotation tests by grain size

Samples, adjusted to the proper liberation grade of the mill feed(+65 mesh; product quantity=approx.15%) which was estimated from the grinding test results, were subjected to the bulk flotation test. The test flow sheet is shown in Fig.11. The floats and tailing products collected respectively at the time groups of 0 ~ 2, 0 ~ 6 and 0 ~ 15 minutes, were sieved out into seven groups by grain sizes between +65 mesh and −325 mesh.

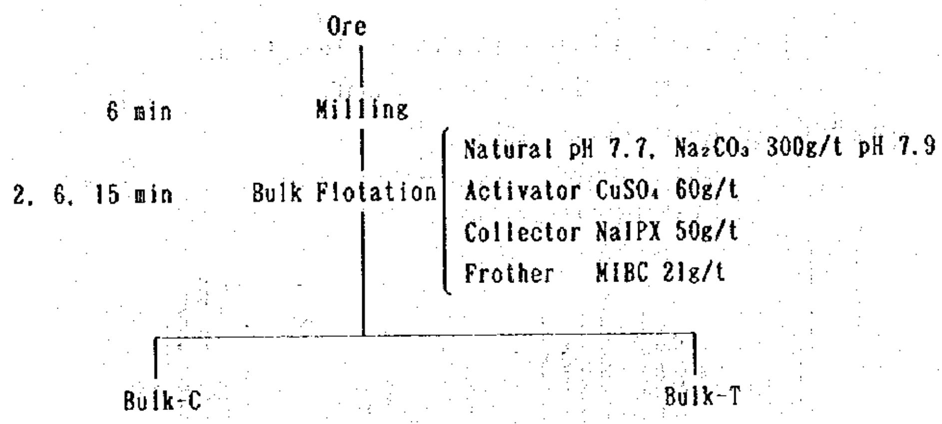



Fig.11 Flotation Test Flowsheet for Grain Size (Bulk Flotation)

The seven products thus obtained were subjected to component analysis and size distribution of metal elements were calculated. Furthermore, flotation ratios by respective flotation time were calculated to find the flotation ratio by grain size.

Since some of the test results were different from those of the 1993 test, as described later, additional flotation test by grain size without adding activator was conducted after the preliminary test.

Fig. 12 ~ 14 show the flotation ratios by grain size at each flotation time while time change of the flotation ratio in each grain size is shown in Fig. 15 ~ 23. [Ref. Apx−24 Flotation Test by Grain Size(with Activator) and Apx−25 Flotation Test by Grain Size(without Activator)]

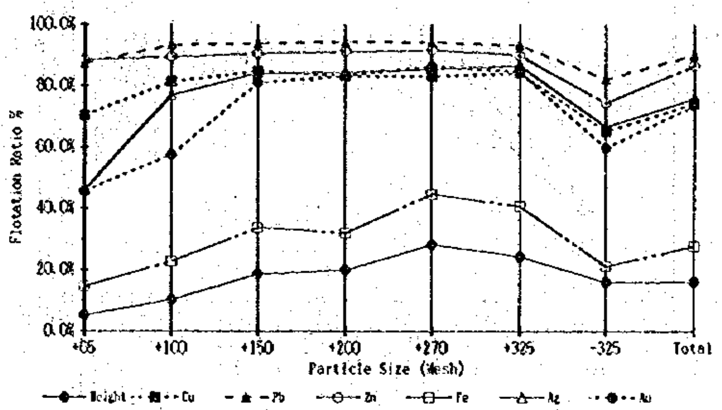



Fig.12 Particle Flotation Ratio of Weight and Metal Elements
(Flotation Time : 0 ～ 2 min)

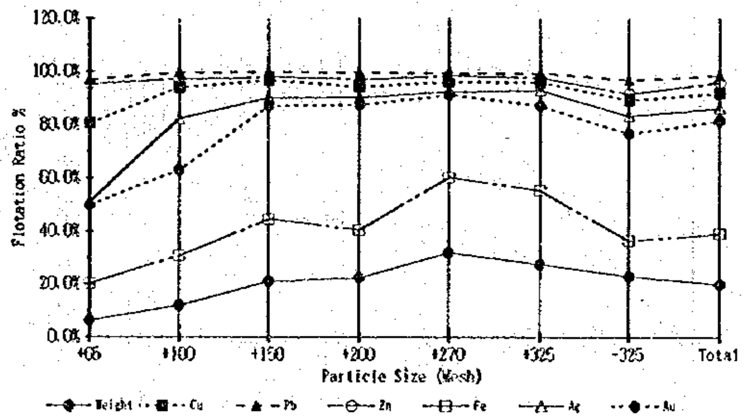



Fig.13 Particle Flotation Ratio of Weight and Metal Elements
(Flotation Time : 0 ～ 6 min)

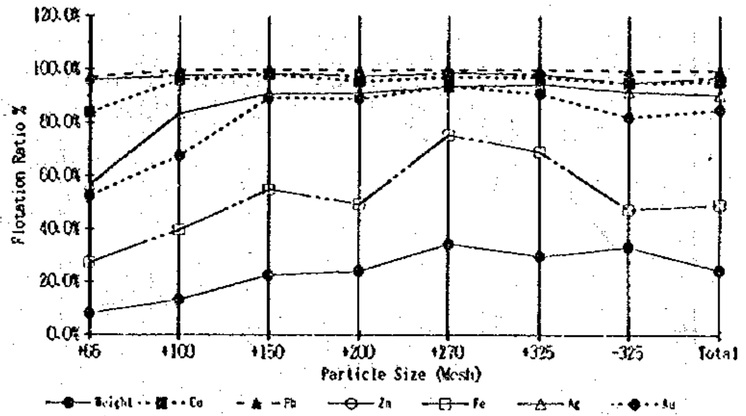



Fig.14 Particle Flotation Ratio of Weight and Metal Elements
(Flotation Time : 0 ～ 15 min)

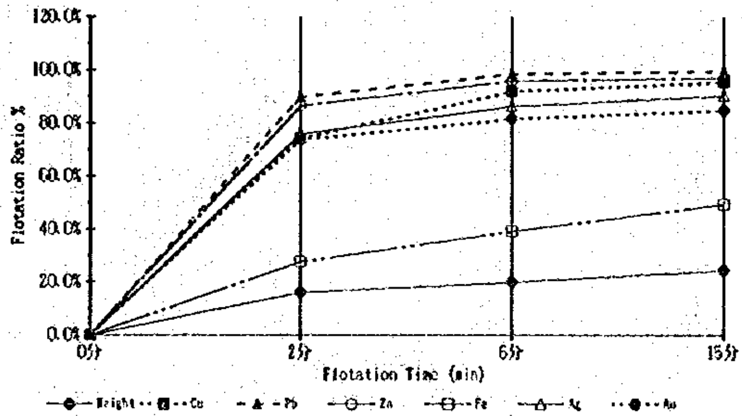



Fig.15 Particle Flotation Ratio of Weight and Metal Elements (Total Product)

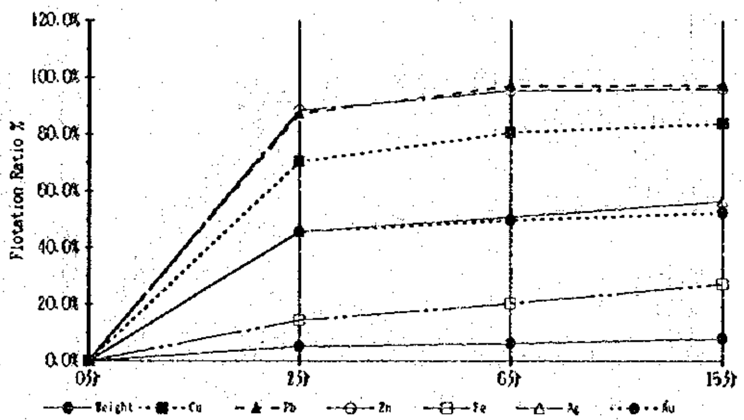



Fig.16 Particle Flotation Ratio of Weight and Metal Elements (+65Mesh Product)

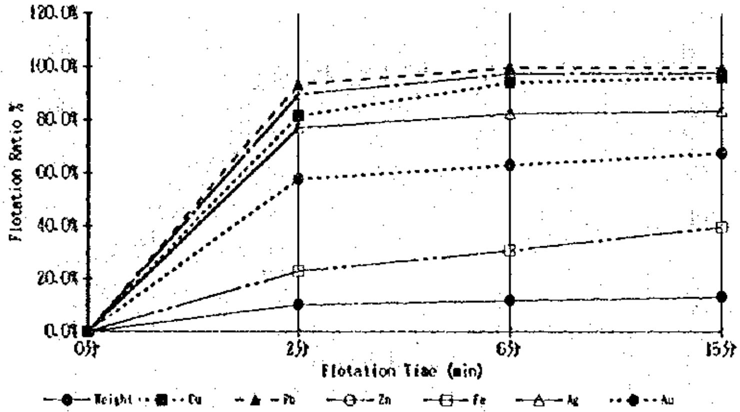



Fig.17 Particle Flotation Ratio of Weight and Metal Elements (+100Mesh Product)

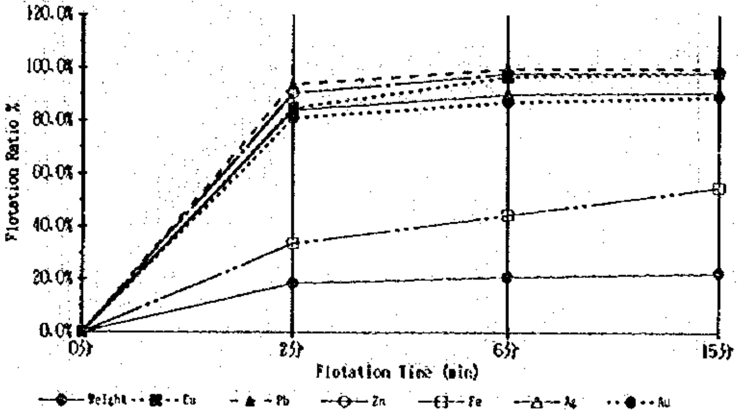



Fig.18 Particle Flotation Ratio of Weight and Metal Elements (+150Mesh Product)

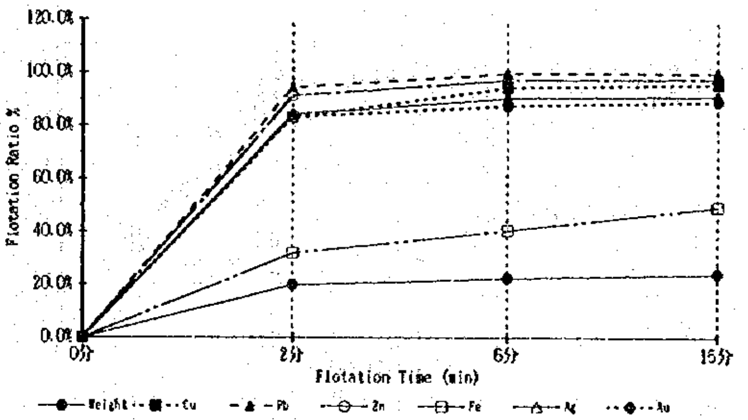



Fig.19 Particle Flotation Ratio of Weight and Metal Elements (+200Mesh Product)

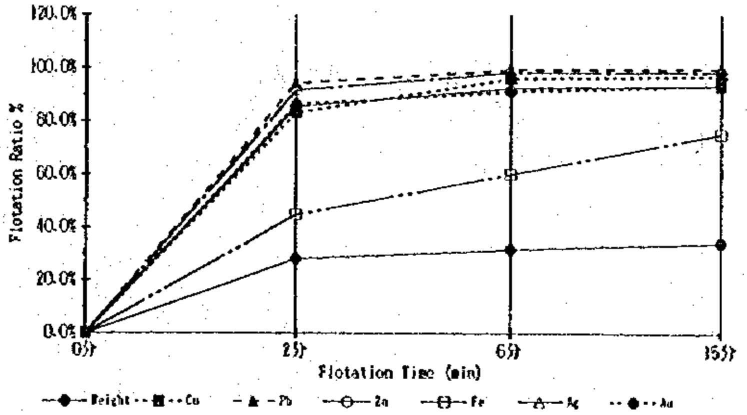



Fig.20 Particle Flotation Ratio of Weight and Metal Elements (+270Mesh Product)

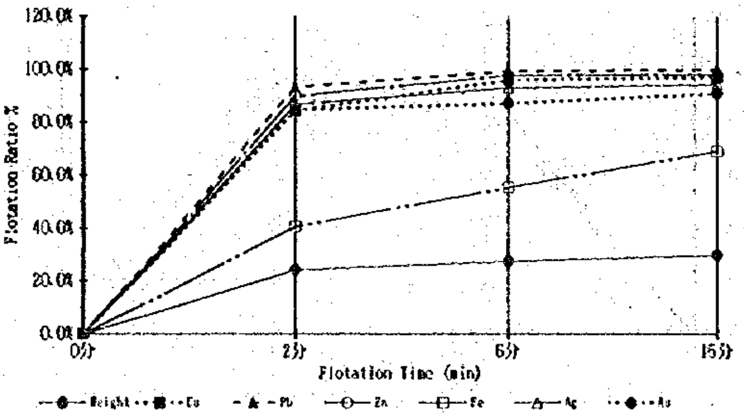



Fig.21 Particle Flotation Ratio of Weight and Metal Elements (+325Mesh Product)

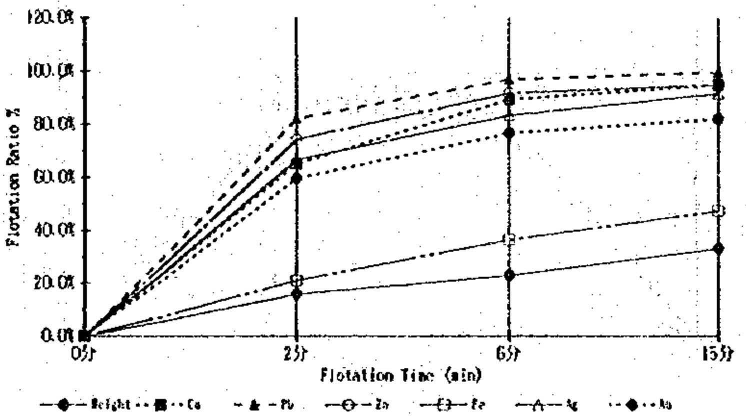



Fig.22 Particle Flotation Ratio of Weight and Metal Elements (−325Mesh Product)

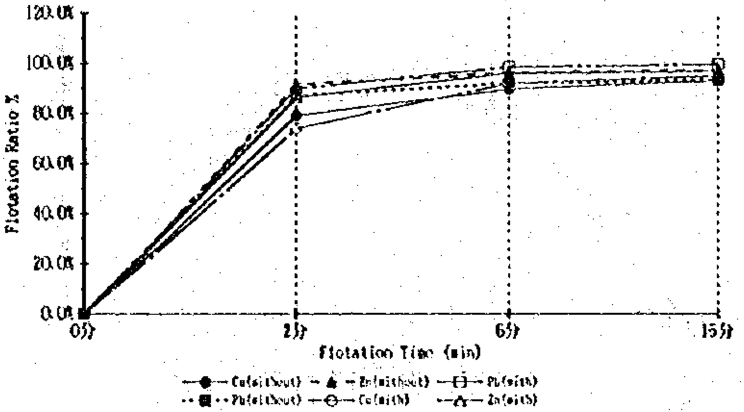



Fig.23 Particle Flotation Ratio of Main Components Metal Elements (With and without Activator, Total Product)

Following are a summary of the test findings:

1) No considerable difference was recognized between the results of flotation test with and without activators, which indicates that zinc minerals are activated by copper.

2) The grain sizes most amenable to flotation are in a range of –100 to 350 mesh, within which no significant difference in flotation ratio is recognized. The second most floatable is the fine grains(–325 mesh), which is followed by the coarse grains(+65 mesh).

3) Main metal minerals show the floatability in the order of lead, zinc and copper; especially zinc shows high flotation ratio which indicates the prominent activation of copper minerals contained in ore, influenced by activator.

4) Most of zinc and lead can be recovered within six minutes of flotation. Even to achieve a 95% roughing recovery, flotation finishes in 12 minutes or so.

5) Floatability of gold and silver is similar to that of lead, zinc and copper, while higher than that of iron.

### 2.3.3 Preliminary test of flotation

#### (1) Bulk–differential flotation

With a view to obtaining basic information on quantity of reagents to be added in the basic test to follow, a preliminary test for bulk differential flotation was carried out. The test conditions were determined in the light of the results of straight differential flotation test in 1993 and the flotation tests by grain size, as well as empirical values.

The test flow sheet, $L_8$ experimental design conditions and the test results are shown in Fig.24, Table 19 and Apx–26, respectively. The Variance analysis results are also shown in Fig.25.

| Test No. | A Collector | B Collector | C Activator | D Depressor |
|---|---|---|---|---|
| 1 | 40g/t | NaEX | CuSO4 100g/t | NaCN 40g/t |
| 2 | 40g/t | NaEX | CuSO4 130g/t | NaCN 60g/t |
| 3 | 40g/t | NaIPX | CuSO4 100g/t | NaCN 60g/t |
| 4 | 40g/t | NaIPX | CuSO4 130g/t | NaCN 40g/t |
| 5 | 60g/t | NaEX | CuSO4 100g/t | NaCN 60g/t |
| 6 | 60g/t | NaEX | CuSO4 130g/t | NaCN 40g/t |
| 7 | 60g/t | NaIPX | CuSO4 100g/t | NaCN 40g/t |
| 8 | 60g/t | NaIPX | CuSO4 130g/t | NaCN 60g/t |

Table 19 $L_8$ Experimental Design Conditions(Preliminary Test– Bulk Differential Flotation)

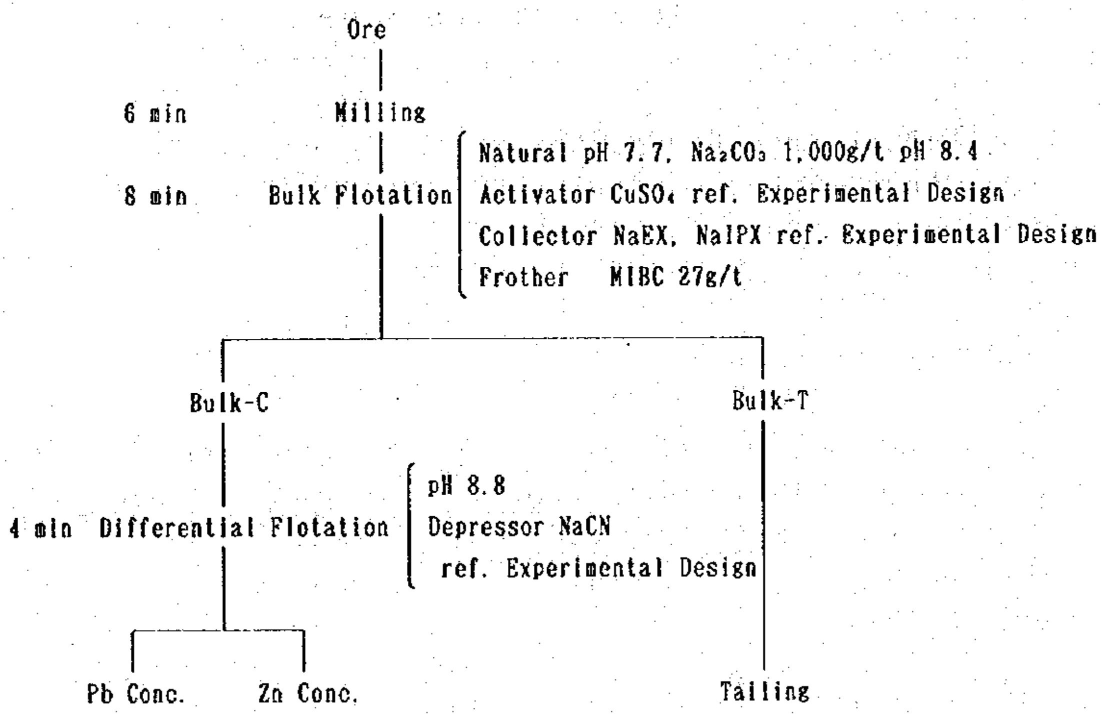



Fig.24 Preliminary Test Flowsheet (Bulk Differential Flotation)

Cu Distribution in Bulk Conc.

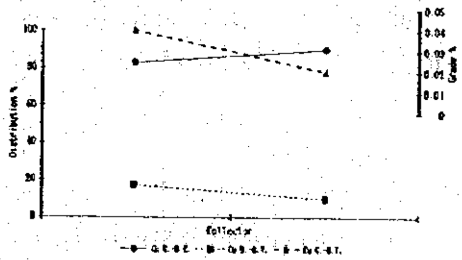
| | SS | $\phi$ | V | Fo | |
|---|---|---|---|---|---|
| A | 10585.2 | 1 | 10585.2 | 7.84 | * |
| C | 5151.1 | 1 | 5151.1 | 3.82 | |
| e | 6749.6 | 5 | 1349.9 | | |
| T | 22485.9 | 7 | | | |

Cu Grade in Bulk Conc.

| | SS | $\phi$ | V | Fo | |
|---|---|---|---|---|---|
| A | 4.5 | 1 | 4.5 | 15.00 | * |
| C | 2.0 | 1 | 2.0 | 6.67 | |
| AxB | 2.0 | 1 | 2.0 | 6.67 | |
| e | 1.0 | 4 | 0.3 | | |
| T | 9.5 | 7 | | | |

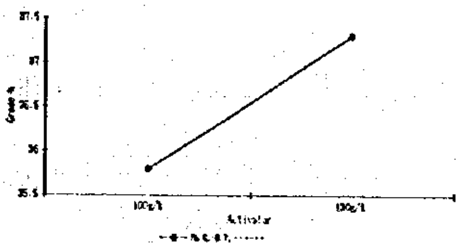
Cu Distribution in Bulk Tailing

| | SS | $\phi$ | V | Fo | |
|---|---|---|---|---|---|
| A | 10585.2 | 1 | 10585.2 | 15.95 | * |
| C | 5151.1 | 1 | 5151.2 | 7.76 | * |
| AxB | 4095.1 | 1 | 4095.1 | 6.17 | |
| e | 2654.5 | 4 | 663.6 | | |
| T | 22485.9 | 7 | | | |



Pb Grade in Bulk Conc.

| | SS | $\phi$ | V | Fo | |
|---|---|---|---|---|---|
| B | 17112.5 | 1 | 17112.5 | 2.67 | |
| C | 45904.5 | 1 | 45904.5 | 7.17 | * |
| e | 32014.5 | 5 | 6402.9 | | |
| T | 95031.5 | 7 | | | |



Fe Distribution in Bulk Conc.

| | SS | $\phi$ | V | Fo | |
|---|---|---|---|---|---|
| C | 4418.0 | 1 | 4418.0 | 7.95 | * |
| BxC | 1624.5 | 1 | 1624.5 | 2.92 | |
| e | 2779.5 | 5 | 555.9 | | |
| T | 8822.0 | 7 | | | |

Fe Grade in Bulk Tailing

| | SS | $\phi$ | V | Fo | |
|---|---|---|---|---|---|
| B | 300.15 | 1 | 300.2 | 2.6 | |
| C | 1596.15 | 1 | 1596.2 | 13.83 | * |
| BxC | 351.1 | 1 | 351.1 | 3.04 | |
| e | 461.5 | 4 | 115.4 | | |
| T | 2708.9 | 7 | | | |

Fe Distribution in Bulk Tailing

| | SS | $\phi$ | V | Fo | |
|---|---|---|---|---|---|
| C | 4418 | 1 | 4418 | 7.79 | * |
| AxB | 512 | 1 | 512 | 0.9 | |
| BxC | 1624.5 | 1 | 1624.5 | 2.87 | |
| e | 2267.5 | 4 | 566.9 | | |
| T | 8822 | 7 | | | |

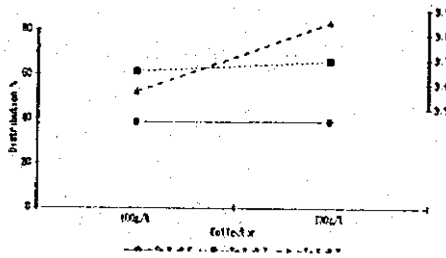


Fig. 25 Variance Analysis Results

The preliminary test of bulk–differential flotation revealed the following:

1) Concerning the type of collector, significant difference was recognized by variance analysis solely in the Pb grade in bulk concentrate. Between the types of collector and copper sulfate, a certain interaction to Pb grade and Pb recovery was recognized, which requires that the collector should be narrowed down to one single type, thereby eliminating the interaction. Considering the fact that the 1993 mineral dressing test resulted in no difference between KAX and NaIPX in terms of the flotation performance, the NaIPX was selected as the collector.

2) It was noted that as collector consumption increases, the copper, lead and zinc recovery tends to increase. The variance analysis showed significant difference only in copper recovery and copper grade. With addition of 60g/t of collector, the copper and lead recovery in bulk concentrate was approx. 90% and the zinc recovery was approx. 96%, whilst the iron distribution of 37% was also high.

3) In case 100g/t of $CuSO_4$, the zinc activator used in bulk flotation, is added, the copper, lead, zinc and iron distribution in bulk concentrate turned out to be conversely higher than in case of 130g/t. The variance analysis indicated significant difference in the iron distribution and grade. It may be interpreted that copper minerals contained in ore was ionized and leached out, activating zinc, which resulted in a state of excessive reagents and lowered the metal distribution. The basic test should therefore be conducted with reduced activator.

4) With addition of 40 ~ 50g/t of NaCN, depressor used in differential flotaion, no depression effect was observed against zinc while, against iron, minor depression effect was recognized. (Fe grade decreased from 7% to 6%.) The variance analysis showed no significant difference in the depression effect in either case. The basic test has to be conducted with increased depressor.

(2) Straight–differential flotation

The preliminary test of bulk differential flotation failed to depress zinc despite more zinc depressor was used than that of the 1993 straight differential flotation test. This indicates that zinc minerals' flotation behavior is considerably different.

As previously referred to, the flotation tests by grain size without adding activator were additionally carried out to study flotation behavior of zinc minerals, which confirmed the fact that copper in ore activated sphalerite.

Since the 1993 testing conditions for SDF do not serve as reference for the subsequent basic test, the preliminary test of lead flotation in SDF was conducted with addition of reagents in a range of two to four times as much as those of the 1993 test and with the factors limited only to collector and depressor.

The test flow sheet, the $L_9$ experimental design conditions and the test results are exhibited in Fig.26, Table 20 and Apx−27, respectively. The variance analysis results appear in Fig.27.

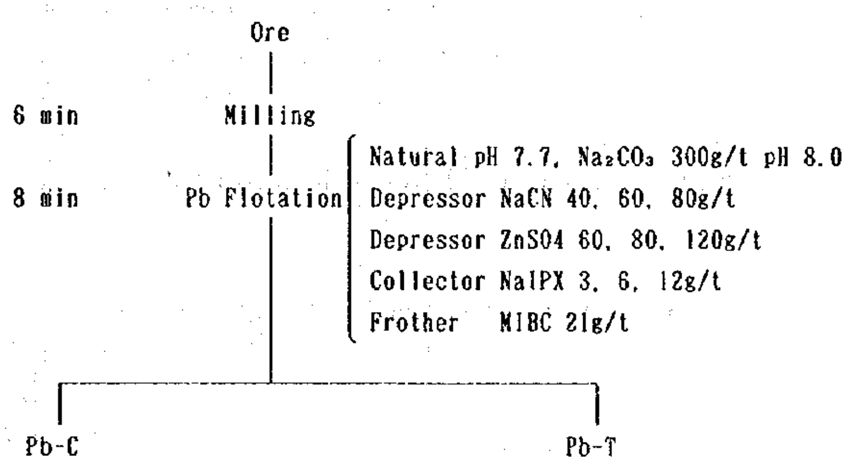



Fig.26 Preliminary Test Flowsheet
(Straight Differential Flotation)

| Test No. | A Depressor | B Collector | C Depressor |
|---|---|---|---|
| 1 | NaCN 40g/t | NaIPX 3g/t | $ZnSO_4$ 60g/t |
| 2 | NaCN 40g/t | NaIPX 6g/t | $ZnSO_4$ 80g/t |
| 3 | NaCN 40g/t | NaIPX 12g/t | $ZnSO_4$ 120g/t |
| 4 | NaCN 60g/t | NaIPX 3g/t | $ZnSO_4$ 80g/t |
| 5 | NaCN 60g/t | NaIPX 6g/t | $ZnSO_4$ 120g/t |
| 6 | NaCN 60g/t | NaIPX 12g/t | $ZnSO_4$ 60g/t |
| 7 | NaCN 80g/t | NaIPX 3g/t | $ZnSO_4$ 120g/t |
| 8 | NaCN 80g/t | NaIPX 6g/t | $ZnSO_4$ 60g/t |
| 9 | NaCN 80g/t | NaIPX 12g/t | $ZnSO_4$ 80g/t |

Table 20 $L_9$ Experimental Design Conditions(Preliminary Test—Straight differential Flotation)

The findings of the preliminary lead flotation test in SDF may be summarized as follows:

1) It was recognized that the copper, lead, zinc and iron distribution in lead concentrate tended to increase as quantity of collector increased. The variance analysis showed significant difference regarding zinc and iron. The test results indicated that collector requirement was 12g/t or more. At 12g/t, the lead recovery in lead concentrate was about 80% and lead grade was 47%; however, some 50% of zinc was mixed into lead roughing concentrate.

2) As regards NaCN, zinc depressor, addition of 60g/t or less produced no depression effect but with 80g/t, it showed a little zinc depressing tendency though insufficient. (The zinc distribution in zinc concentrate was about 20%.) The variance analysis indicated no significant difference in NaCN. The said quantity of NaCN addition is close to that for flotation of the Kuroko, from which the existence of copper sulfide can be inferred. On the other hand, iron depressing tendency of NaCN is recognizable.

**Zn Grade in Pb Conc.**

| | SS | $\phi$ | V | Fo | |
|---|---|---|---|---|---|
| A | 81789.6 | 2 | 40894.8 | 0.75 | |
| B | 783557.6 | 2 | 391778.8 | 7.19 | * |
| e | 218033.8 | 4 | 54508.5 | | |
| T | 1083380.9 | 8 | | | |

**Fe Grade in Pb Conc.**

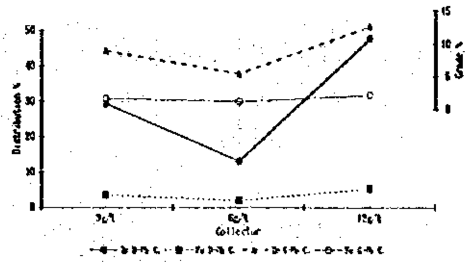
| | SS | $\phi$ | V | Fo | |
|---|---|---|---|---|---|
| B | 14946.8 | 2 | 7473.4 | 26.56 | ** |
| C | 2561.6 | 2 | 1280.8 | 4.55 | |
| e | 1125.6 | 4 | 281.4 | | |
| T | 19921.6 | 8 | | | |

**Zn Distribution in Pb Conc.**

| | SS | $\phi$ | V | Fo | |
|---|---|---|---|---|---|
| B | 178786.8 | 2 | 89393.4 | 8.40 | * |
| C | 24821.6 | 2 | 12410.8 | 1.17 | |
| e | 42576.5 | 4 | 10644.1 | | |
| T | 246184.9 | 8 | | | |

**Fe Distribution in Pb Conc.**

| | SS | $\phi$ | V | Fo | |
|---|---|---|---|---|---|
| B | 1700.7 | 2 | 850.4 | 10.94 | * |
| C | 290.7 | 2 | 145.4 | 1.87 | |
| e | 310.7 | 4 | 77.7 | | |
| T | 2302.1 | 8 | | | |



**Zn Distribution in Pb Tailing**

| | SS | $\phi$ | V | Fo | |
|---|---|---|---|---|---|
| B | 178786.8 | 2 | 89393.4 | 8.40 | * |
| C | 24821.6 | 2 | 12410.8 | 1.17 | |
| e | 42576.5 | 4 | 10644.1 | | |
| T | 246184.9 | 8 | | | |

**Fe Distribution in Pb Tailing**

| | SS | $\phi$ | V | Fo | |
|---|---|---|---|---|---|
| B | 1700.7 | 2 | 850.4 | 10.94 | * |
| C | 290.7 | 2 | 145.4 | 1.87 | |
| e | 310.7 | 4 | 77.7 | | |
| T | 2302.0 | 8 | | | |

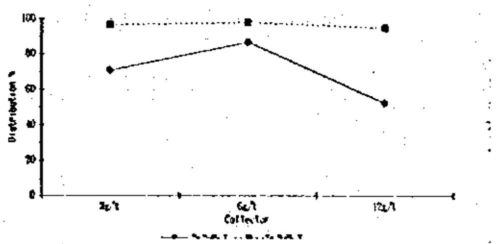


Fig. 27 Variance Analysis Results

3) Although $ZnSO_4$ shows some zinc depressing tendency, the effect is not so clear within the designed range of addition quantity. The variance analysis showed no significant difference. In the basic test to follow, the quantity of addition has to be increased for further study.

4) In the 1993 test, zinc depression was attained with a small amount of NaCN while the preliminary test required a larger amount of NaCN, which suggests that zinc minerals from the same ore body differ considerably in terms of the flotation behavior. It will also be necessary to verify whether zinc mixed into the lead flotation can be depressed in the cleaners.

### 2.3.4 Basic test of flotation

#### (1) Bulk differential flotation

#### (1)-1 Bulk flotation

Based on the results of preliminary BDF test, bulk flotation test was carried out with increased addition of collector for improving lead recovery and with reduced zinc activator($CuSO_4$). As a new test factor, quantity of lime for depressing iron was added. The test flow sheet, the $L_9$ experimental design conditions and the test results are shown in Fig.28, Table 21 and Apx-28, respectively.

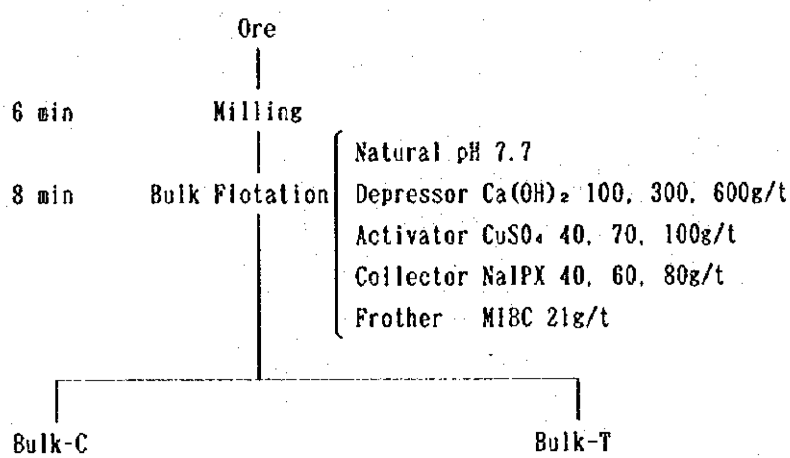



Fig. 28 Basic Test Flowsheet
(Bulk Flotation)

| Test No. | A Aotivator | B Colleotor | C Depressor |
|---|---|---|---|
| 1 | $CuSO_4$ 40g/t | NaIPX 40g/t | $Ca(OH)_2$ 100g/t |
| 2 | $CuSO_4$ 40g/t | NaIPX 60g/t | $Ca(OH)_2$ 300g/t |
| 3 | $CuSO_4$ 40g/t | NaIPX 80g/t | $Ca(OH)_2$ 600g/t |
| 4 | $CuSO_4$ 70g/t | NaIPX 40g/t | $Ca(OH)_2$ 300g/t |
| 5 | $CuSO_4$ 70g/t | NaIPX 60g/t | $Ca(OH)_2$ 600g/t |
| 6 | $CuSO_4$ 70g/t | NaIPX 80g/t | $Ca(OH)_2$ 100g/t |
| 7 | $CuSO_4$ 100g/t | NaIPX 40g/t | $Ca(OH)_2$ 600g/t |
| 8 | $CuSO_4$ 100g/t | NaIPX 60g/t | $Ca(OH)_2$ 100g/t |
| 9 | $CuSO_4$ 100g/t | NaIPX 80g/t | $Ca(OH)_2$ 300g/t |

Table 21 $L_9$ Experimental Design Conditions
(Basic Test－Bulk Flotation)

The following tendencies were observed from the results of basic test of bulk flotation:

1) The copper, lead, zinc and iron distribution in concentrate peaked at 60g/t of collector added. With its addition of 80g/t, the flotation conditions worsened due to excessive reagents, lowering the metal distribution. No significant difference was found by the variance analysis. With 60g/t of added collector, the iron distribution was high while the lead and zinc recovery was not improved so much; therefore, addition of 40g/t of collector is considered sufficient. With this amount of collector, the lead recovery in bulk concentrate was about 90%(Pb grade 39%) while zinc recovery was about 95%(Zn grade 16%).

2) Within the designed range of quantity of added lime, no significant difference was observed in the iron depression effect. The iron distribution in bulk concentrate was about 35%, indicating depression being insufficient. Quantity of added lime has to be increased for further testing.

3) The quantitative variations in the added zinc activator made no difference in mineral dressing performance. Since zinc grade at about 16% and zinc recovery at about 95% were obtained, addition of 40g/t is considered sufficient.

(1)–2 Differential flotation

Bulk flotation and differential flotation tests were carried out, with the test factors of quantities of iron depressor(lime) and zinc depressor (NaCN) to be added. The $L_9$ experimental design conditions, the test flow sheet and the test results are exhibited in Table 22, Fig.29 and Apx–29, respectively.

| Test No. | A Depressor | B Depressor |
|---|---|---|
| 1 | $Ca(OH)_2$ 300g/t | NaCN 200g/t |
| 2 | $Ca(OH)_2$ 600g/t | NaCN 100g/t |
| 3 | $Ca(OH)_2$ 1000g/t | NaCN 150g/t |
| 4 | $Ca(OH)_2$ 600g/t | NaCN 150g/t |
| 5 | $Ca(OH)_2$ 1000g/t | NaCN 200g/t |
| 6 | $Ca(OH)_2$ 300g/t | NaCN 100g/t |
| 7 | $Ca(OH)_2$ 1000g/t | NaCN 100g/t |
| 8 | $Ca(OH)_2$ 300g/t | NaCN 150g/t |
| 9 | $Ca(OH)_2$ 600g/t | NaCN 200g/t |

Table 22 $L_9$ Experimental Design Conditions(Basic Test–Pb Differential Flotation)

The basic test findings of differential flotation may be summarized as follows(ref. Fig.30 Variance Analysis):

1) As regards lime for iron depression in bulk flotation, the quantitative variations in its addition within the designed range made no difference in the depression of iron; the iron grade was 8% or so and the iron distribution was about 33%. The variance analysis showed no significant difference either. In case 1,000g/t of lime was added, however, the lead recovery in bulk concentrate declined from 90% to 89%. The variance analysis also indicated a significant difference of 10% in decrease of the lead recovery.

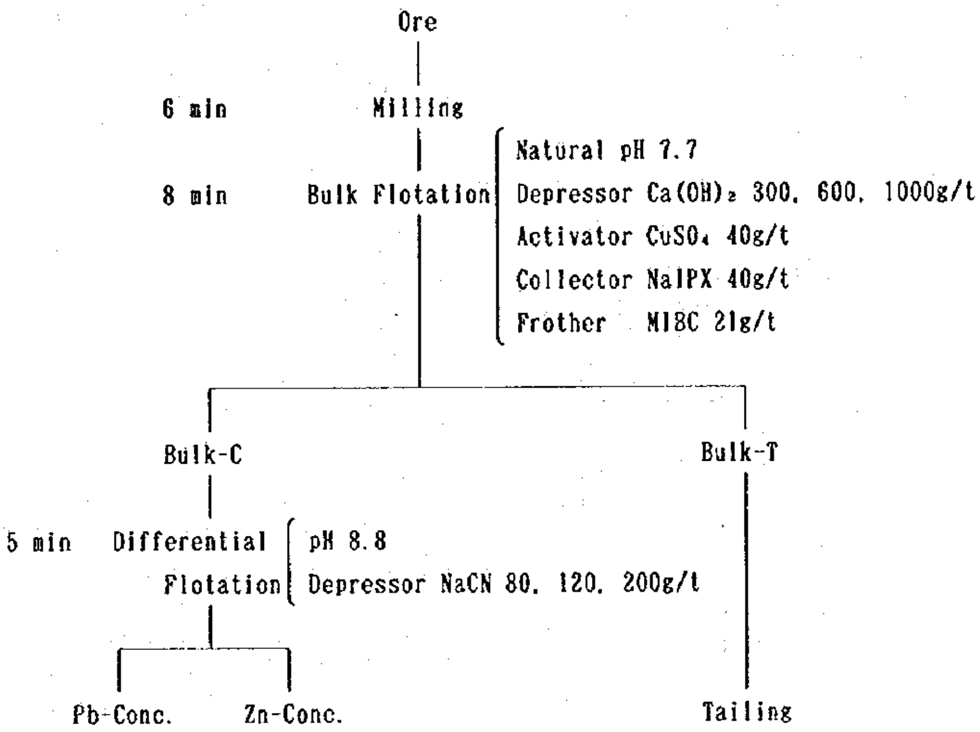



Fig. 29 Basic Test Flowsheet
(Differential Flotation)

2) No changes in the lead recovery was resulted from increased addition of zinc depressor(NaCN), but it showed tendencies to elevate the lead grade while to lower the zinc grade and distribution. (Zinc distribution: 36% → 30% → 15%) The variance analysis verified significant difference of 10% in the lead, zinc and iron grades. For depression of zinc, addition of some 200g/t was needed, while, under 150g/t, depression of zinc was insufficient. When 200g/t was added, the zinc grade in bulk concentrate declined to 5% and the zinc recovery was some 15%.

3) NaCN, depressor of zinc, proved to be effective to iron, as well. (Iron distribution: 7% → 5% → 3%) The variance analysis indicated significant difference of 10% in the iron grade. With its addition of 200g/t, the iron grade in lead concentrate declined to 2% or less while the iron distribution to 3%.

Pb grade in Pb Conc.

| | SS | φ | V | Fo | |
|---|---|---|---|---|---|
| A | 100477.9 | 2 | 50239.0 | 1.02 | |
| B | 1062818.7 | 2 | 531409.4 | 10.74 | * |
| e | 197967.4 | 4 | 49491.9 | | |
| T | 1361264.0 | 8 | | | |

Zn Grade in Pb Conc.

| | SS | φ | V | Fo | |
|---|---|---|---|---|---|
| A | 145083.5 | 2 | 72541.8 | 1.43 | |
| B | 791437.6 | 2 | 395718.8 | 7.81 | * |
| e | 202802.5 | 4 | 50700.6 | | |
| T | 1139323.6 | 8 | | | |

Fe Grade in Pb Conc.

| | SS | φ | V | Fo | |
|---|---|---|---|---|---|
| A | 3120.9 | 2 | 1560.5 | 0.64 | |
| B | 38586.9 | 2 | 19293.5 | 7.87 | * |
| e | 9803.1 | 4 | 2450.8 | | |
| T | 51510.9 | 8 | | | |

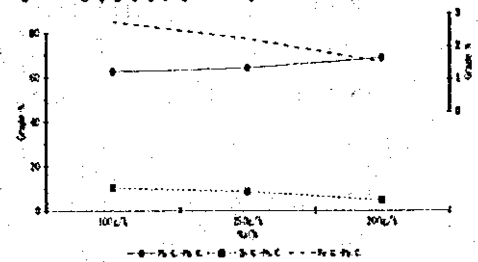


Fig. 30 Variance Analysis Results

(2) Straight-differential flotation

(2)-1 Lead flotation of SDF

Based upon the preliminary test results of SDF, the basic test was carried out, with the elevated ranges of quantities of collector(NaIPX) and zinc depressor(NaCN, $ZnSO_4$) to be added. The test flow sheet, the $L_9$ experimental design conditions and the test results are exhibited in Fig. 31, Table 23 and Apx-30, respectively.

Ore

6 min — Milling

5 min — Pb Differential Flotation

- Natural pH 7.7, $Na_2CO_3$ 300g/t pH 8.0
- Depressor NaCN 60, 100, 160g/t
- Depressor $ZnSO_4$ 150, 200, 250g/t
- Collector NaIPX 10, 15, 20g/t
- Frother MIBC 21g/t

Pb-C　　Pb-T

Fig.31 Basic Test Flowsheet
(Pb Flotation in Straight Differential Flotation)

| Test No. | A Collector | B Depressor | C Depressor |
|---|---|---|---|
| 1 | NaIPX 10g/t | NaCN 60g/t | $ZnSO_4$ 150g/t |
| 2 | NaIPX 10g/t | NaCN 100g/t | $ZnSO_4$ 200g/t |
| 3 | NaIPX 10g/t | NaCN 160g/t | $ZnSO_4$ 250g/t |
| 4 | NaIPX 15g/t | NaCN 60g/t | $ZnSO_4$ 200g/t |
| 5 | NaIPX 15g/t | NaCN 100g/t | $ZnSO_4$ 250g/t |
| 6 | NaIPX 15g/t | NaCN 160g/t | $ZnSO_4$ 150g/t |
| 7 | NaIPX 20g/t | NaCN 60g/t | $ZnSO_4$ 250g/t |
| 8 | NaIPX 20g/t | NaCN 100g/t | $ZnSO_4$ 150g/t |
| 9 | NaIPX 20g/t | NaCN 160g/t | $ZnSO_4$ 200g/t |

Table 23 $L_9$ Experimental Design Conditions(Basic Test–Pb Straight Differential Flotation)

The basic test findings of lead flotation may be summarized as follows (Ref. Fig. 32 Variance Analysis):

1) It was observed that the copper, lead, zinc and iron distribution tended to rise as quantity of added collector increased. (Pb: 76% → 83% → 86%) The variance analysis verified no significant difference. In order to improve lead recovery while depressing zinc and iron distribution, quantity of collector has to be controlled at around 15g/t. At this quantity, the lead recovery was 83% while the zinc and iron distribution was 74% and 10%, respectively. Such high distribution of zinc and iron necessitates multiple–stage cleaning; testing is required to determine the necessary number of stages.

2) The zinc depressor(NaCN) clearly depressed zinc and iron. The variance analysis, as well, verified significant difference of 10% in the iron and zinc distribution in bulk concentrate. Depression was insufficient at 100g/t or below; therefore, addition over 160g/t is required. At this quantity, the lead recovery was 84% while the zinc and iron distributions were 53% and 6%, respectively.

#### Fe Grade in Pb Conc.

| | SS | φ | V | Fo | |
|---|---|---|---|---|---|
| A | 4362.0 | 2 | 2181.0 | 9.63 | |
| B | 25640.6 | 2 | 12820.3 | 56.63 | * |
| C | 2724.7 | 2 | 1362.4 | 6.02 | |
| e | 452.7 | 2 | 226.4 | | |
| T | 33180.0 | 8 | | | |

#### Zn Distribution in Pb Conc.

| | SS | φ | V | Fo | |
|---|---|---|---|---|---|
| A | 36722 | 2 | 18361 | 11.31 | |
| B | 135528 | 2 | 67764 | 41.75 | * |
| C | 854 | 2 | 427 | 0.26 | |
| e | 3246 | 2 | 1623 | | |
| T | 176350 | 8 | | | |

#### Fe Distribution in Pb Conc.

| | SS | φ | V | Fo | |
|---|---|---|---|---|---|
| A | 1281.6 | 2 | 640.8 | 11.98 | |
| B | 3902.9 | 2 | 1951.5 | 36.48 | * |
| C | 421.6 | 2 | 210.8 | 3.94 | |
| e | 106.9 | 2 | 53.5 | | |
| T | 5712.9 | 8 | | | |

#### Zn Distribution in Pb Tailing

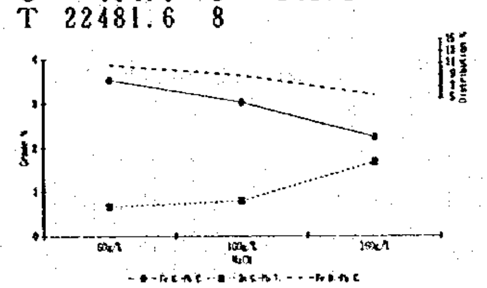
| | SS | φ | V | Fo | |
|---|---|---|---|---|---|
| A | 36722 | 2 | 18361 | 11.31 | |
| B | 135528 | 2 | 67764 | 41.75 | * |
| C | 854 | 2 | 427 | 0.26 | |
| e | 3246 | 2 | 1623 | | |
| T | 176350 | 8 | | | |

#### Zn Grade in Pb Tailing

| | SS | φ | V | Fo | |
|---|---|---|---|---|---|
| A | 4032.9 | 2 | 2016.5 | 8.33 | |
| B | 17869.6 | 2 | 8934.8 | 36.89 | * |
| C | 94.9 | 2 | 47.5 | 0.2 | |
| e | 484.3 | 2 | 242.2 | | |
| T | 22481.6 | 8 | | | |

#### Fe Distribution in Pb Tailing

| | SS | φ | V | Fo | |
|---|---|---|---|---|---|
| A | 1281.6 | 2 | 640.8 | 11.98 | |
| B | 3902.9 | 2 | 1951.5 | 36.48 | * |
| C | 421.6 | 2 | 210.8 | 3.94 | |
| e | 106.9 | 2 | 53.5 | | |
| T | 5712.9 | 8 | | | |



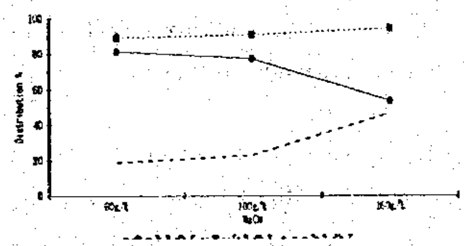


Fig. 32 Variance Analysis Results

3) The quantity of NaCN added for depressing zinc was close to that in the lead differential flotation(BDF).

4) Within the designed range for addition of copper sulfate, no changes were observed in flotation performance. It may be said that addition of copper sulfate was unnecessary for the samples used in the test.

## (2)-2 Zinc flotation in SDF

Subsequent to the lead flotation conducted under the best conditions referred to in the preceding item, basic test of zinc flotation was carried out, with the test factors of quantities of activator($CuSO_4$) and iron depressor($Ca(OH)_2$). The test flow sheet, the $L_9$ experimental design conditions and the test results are shown in Fig. 33, Table 24 and Apx-31, respectively.

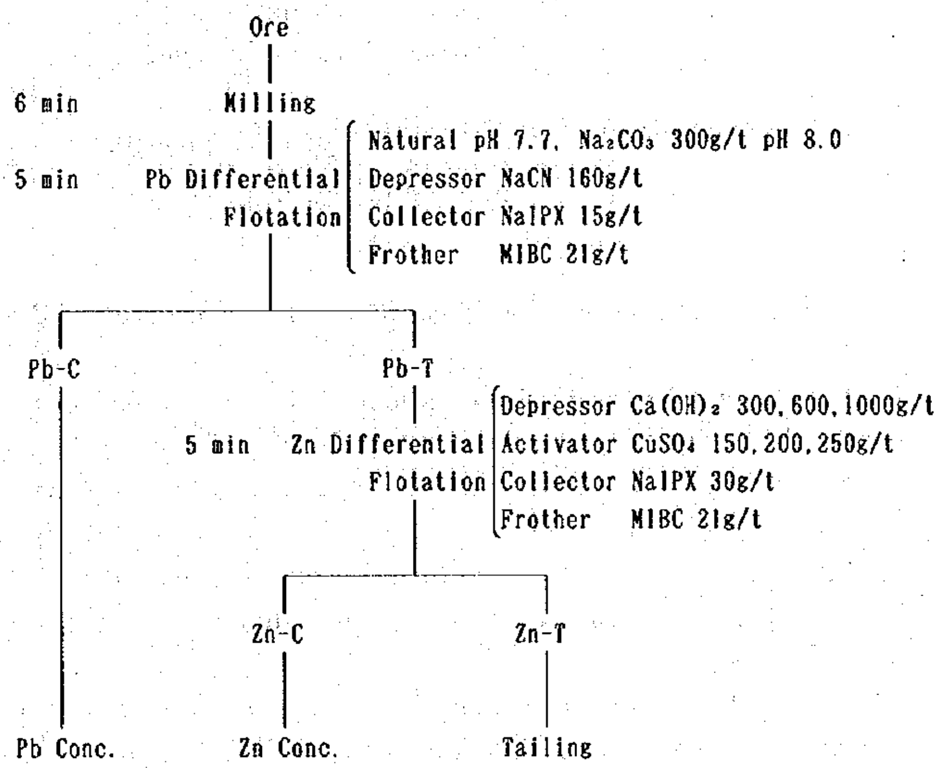



Fig. 33 Basic Test Flowsheet
(Zn Flotation in Straight Differential Flotation)

| Test No. | A Activator | B Depressor |
| --- | --- | --- |
| 1 | $CuSO_4$ 150g/t | $Ca(OH)_2$ 300g/t |
| 2 | $CuSO_4$ 200g/t | $Ca(OH)_2$ 600g/t |
| 3 | $CuSO_4$ 250g/t | $Ca(OH)_2$ 1000g/t |
| 4 | $CuSO_4$ 200g/t | $Ca(OH)_2$ 1000g/t |
| 5 | $CuSO_4$ 250g/t | $Ca(OH)_2$ 300g/t |
| 6 | $CuSO_4$ 150g/t | $Ca(OH)_2$ 600g/t |
| 7 | $CuSO_4$ 250g/t | $Ca(OH)_2$ 600g/t |
| 8 | $CuSO_4$ 150g/t | $Ca(OH)_2$ 1000g/t |
| 9 | $CuSO_4$ 200g/t | $Ca(OH)_2$ 300g/t |

Table 24 $L_9$ Experimental Design Conditions(Basic Test–Zn Straight Differential Flotation)

The basic test findings of zinc flotation may be summarized as follows:

1) Within the determined range of quantity of zinc depressor($CuSO_4$), the copper, lead and zinc distribution in zinc concentrate remained almost the same whilst the iron distribution tended to rise(10% → 12% → 14%). The variance analysis indicated no significant difference, though. It is interpreted that since most of zinc had distributed in floats already in the lead flotation, leaving only a small portion in the zinc flotation, the zinc activator served to activate copper, instead.

2) As quantity of iron depressor added increased from 300 to 600g/t, and to 1,000g/t, the iron grade in concentrate declined from 15% to 13%, and to 10% and also the iron distribution from 13% to 10%, and to 9%, indicating a tendency of iron being depressed. The variance analysis showed no significant difference.

3) The basic test results suggest that, in the final test using samples which contain a large amount of activated zinc, most zinc would be removed in the lead flotation(in SDF). It is considered unpractical, therefore, to infer continuous operation performance of the zinc flotation from the results of a batch test.