# AUTOMATION IN COPPER FLOTATION CIRCUITS

**A Thesis Submitted to the**
**Graduate School of Natural and Applied Sciences of**
**Dokuz Eylul University**
**In Partial Fulfillment of the Requirements for**
**the Degree of Master of Science in Mining Engineering, Mineral Processing Program**



**by**
**Ahmet Taner ARIKAN**

November, 1998
**IZMIR**

## M. Sc. THESIS EXAMINATION RESULT FORM

We certify that we have read this thesis and that in our opinion it is fully adequate, in scope and in quality, as a thesis for the Master of Science Degree in Mining Engineering Program.

**Assoc. Prof. Dr. Yaşar ÇİLİNGİR**
**(Adviser)**

**Prof. Dr. Ali AKAR**

**Assoc. Prof. Dr. Hasan MORDOĞAN**

Approved by the
Graduate School of Natural and Applied Sciences

**Prof. Dr. Cahit HELVACI**
Director

# ACKNOWLEDGEMENTS

I would like to express my appreciation to Assoc. Prof. Dr. Yasar ÇİLİNGİR for his kind supervision and valuable suggestions. Also I would like to express my thanks to my wife Mumine ARIKAN, Muhsin DOĞAN and Research Assist. Ahmet Vedat ÖZSEVER for their helpful attitude and effort while preparing this thesis.

Ahmet Taner ARIKAN

# ABSTRACT

The variables of flotation are generally controlled by means of conventional devices. However, due to the physical and chemical properties of ore, their efficiency is relatively limited, if an optimal operation is sought for. Especially the varying composition of the raw ore is a disturbance input to which the plant should adapt rather than try to keep to fixed setpoints.

Expert system or knowledge-based system are a branch of artificial intelligence which has recently gained an increasing interest. An expert control strategy may control the setpoints of process controllers and record the effects of such changes and of similar change made by the human operator, in order to decide on further actions, e.i. in flotation of such minerals which are not sensed by on-line analysis instrumentation, the implementation of new control logics has an obvious potential.

This thesis describes the determination of the hierarchy of manipulated variables and expert rules by giving examples from Etibank Kure Copper Concentrator.

# ÖZET

Flotasyonda üretim degişkenleri klasik proses kontrol cihazlarıyla kontrol edilmektedir. Ancak optimum noktada bir üretim söz konusu olduğunda, bu cihazların etkileri sınırlı kalmaktadır. Özellikle tüvönan cevherin fiziksel ve kimyasal özelliklerindeki değişiklikler flotasyonun tüm noktalarında hissedilmektedir ve klasik kontrol cihazlarının prosese müdahale süreleri uzamaktadır. Neticede ham madde tüketimi artmakta ve maliyetler yükselmektedir.

Son yıllarda büyük bir gelisme gösteren madencilikte otomasyon uygulamaları ile proses cihazlarının "set - point" leri, belirlenen kontrol stratejisi ile ayarlanabilmekte ve sonuçlar kayıt edilmektedir. Böylece operatörün yapabileceği tüm müdahaleler daha kısa sürede ve daha doğru olarak bilgisayarla otomatik olarak yapmak mümkün olmaktadır.

Bu tez ile bakır flotasyonundaki mantık stratejisi, Etibank Küre Bakırlı Pirit İşletmesi'ndeki çalışmalardan örnekler verilerek açıklanmıştır.

# CONTENTS

**Page**



## Chapter One
## INTRODUCTION

## Chapter Two
## CONTROL LOOPS



## Chapter Three
## PROCESS CONTROL

## Chapter Four
## PROCESS STABILIZATION



## Chapter Five
## CONCLUSIONS

# LIST OF TABLES

# LIST OF FIGURES

CHAPTER ONE

# INTRODUCTION

## 1. General

In flotation which is an important part of ore dressing technology, control of process is of course an important phenomenon. As known, flotation is an continuous system and is used to enrich low grade sulfurous ores by taking advantage of physical and chemical properties. In flotation plants hundreds of $m^3$ of pulp can be processed at suitable pH level and certain reactives are added to this pulp. Surface of precious minerals is turned to hydrophobic structure with these reactives added to the system. Then it is forced to adhere formed foam and with an appropriate method the foam is taken away and concentrate is obtained.

Since the reactives used in concentrate production, are imported and they are the second most important item in plant cost after grinding. In order to be able to consume the reactives at optimum level the process should always be kept under control effectively. It is very clear that reactive usage at desired volume and at right places will lower the cost. In the same way control of other raw materials used in flotation process, products, middlings and tailing has an big effect on cost.

Mining engineers should know about what kind of devices are to used and what they can perform in flotation plants for control purposes. Because mining engineers are fully responsible for all steps in production. At modern flotation plants electronics engineers also work and they always keep in touch with mining engineers. For this reason these two engineers of different disciplines need to be responsible in solving problems.

In this work basic subjects that a mining engineer should know about at an automated flotation plant, are studied. Similarly at an flotation plant that is due to be automated, subject of matters that should be determined by mining engineer in choosing of instruments, is also studied. Also in this work, particle size control, grade analyzer, pH and reactive controls, pulp level control and integration of these devices with automation system are attempted to be explain respectively by giving examples from Kure concentrator and datas used in process control are showed in form of tables where necessary.

### 1.1. General Knowledge About Copper Flotation

Flotation , which is applied to very fine particles that are very difficult to separate, is an enrichment method. Flotation method has lead to developments in mining industry by letting different types of low grade or complex structured ore deposits which are not possible to enrich by gravity method, be processed. Nowadays big part of the world demands of some metals like Cu, Zn, Pb, Ag is produced by flotation.

#### 1.1.1. Surface Chemistry in Copper Flotation

Apart from a few exceptions, surface of inorganic masses get completely wet in water. The reason that causes wetting is the bonding between ions on the mass surface and hydrated ions in water. As well as these ions could be hydrogen and hydroxyl ions. They could also be other electrolytes existing in water. First step of flotation is to form a solid-air interface on the mineral which is to be floated instead of forming a water-solid interface. This can be done by adding the chemicals that are suitable for water phase. As a result of a reaction occurring between mineral surface and reactives, a hydrofob (i.e. non - watery) layer is formed on the mineral surface. Explanation and finding of the reaction between mineral surface and chemical substance is possible with thermodynamic principles (Atak, S., 1982).

Flotation of copper sulphur is generally easy, cheap and effective ore dressing method. The main problem is to depress pyrite. By using lime, the media can be made alcaline (pH, 10-12). To depress pyrite, to blow air to pulp by strong stirring, hence it is possible to prevent the flotation of easy oxidised pyrite surfaces by oxidation compared to copper ores. As well as rising pH level, adding some amount of cyanide also easy to depress pyrite. Because cyanide is very expensive comparing to lime and excess amount of cyanide can also depress copper, it should be used only with small amounts (i.e. 5 - 20 gr./ton). After depressing pyrite, copper ore is floated by using xanthate and dithiophosphate type collector and a type of frother. Even though flotation of copper mineral is generally quite easy, some factors that makes flotation difficult could be encountered. The followings are these factors (Atak, S., 1982) :

- Excess amount of pyrite and insufficient amount of copper mineral in ore; it is generally excepted that ore deposits having more than 1% Cu are good ores. At pyrite deposits with such a small amount of copper, depressing of pyrite and obtaining clean copper concentrate can be difficult. By introducing staged cleaning circuits, clear copper concentrate can be produced from such ores.

- Oxidation of ore deposit; concentration of copper minerals can be very hard when it comes to oxidation rate. Recovery of copper can drop and due to oxidation, increased quantities of copper ions in the pulp can activate pyrite. At the result, consumption of reactives can be increased to obtain clean concentrate.

- Lack of suitable liberation between pyrite and copper minerals; existence of chalcopyrite as very fine particles in pyrite or coating of pyrite surface with thin layer of chalkosine can cause both a drop in concentrate grade and loss in metal output.

- Obtaining of precious metals; it is desired that gold and silver in ore is wanted to be gathered in concentrated copper after enrichment. But if they exist in pyrite rather than existing in copper minerals, instead of obtaining a high grade copper concentrate, it can be thought increase gold and silver output with a lower grade copper concentrate.

### 1.2. Flotation Conditions in Kure Concentrator

Raw ore that is feeding to concentrator is containing 1.71 % Cu and % 37 S and after grinding % 70 of pulp is under 38μ.

- Average raw ore feeding is 100 t/h
- Total conditioning time in copper circuit is 20 minutes.
- Total flotation time in copper circuit is about 70 minutes

- pH in copper circuit is between 11-12
- pH in pyrite circuit is between 4-5

Reactives and its consumptions are given below ( Arikan, A. T., 1992 - 1995 ) :

**1- pH Regulators**

- Lime :

| | |
|---|---|
| Type | : Lump lime ( 80 % CaO) |
| Feeding rate | : 10 kg/ton. |
| Feeding Point | : RM |

- Acid :

| | |
|---|---|
| Name | : $H_2SO_4$ |
| Addition rate | : Average 2 kg/ton |
| Feeding Point | : PK1 |

**2- Collectors**

| | |
|---|---|
| Commercial name | : HOSTAFLOAT X231 |
| Addition rate | : see Table 1.1 |
| Feeding Points | : BK1, BU2, BK5, BU5 |
| Feeding method | : mixed with water (5 % Hostafloat) |

| | |
|---|---|
| Name | : KAX |
| Feeding rate | : 5 lt/min |
| Feeding point | : PK2 |
| Feeding method | : mixed with water (5 % KAX) |

**3- Frother**

| | |
|---|---|
| Commercial name | : DOWFROTH 250 E |
| Specific gravity | : 1.01 g/cc |
| Total addition rate | : 26 gr/ton |
| Maximum ore feeding | : 140 t/h |
| Maximum frother feeding | : 3640 g/h = 60 gr/min |
| Feeding points | : BK4, BK5, PK2 |
| Feeding method | : Pure |

**Table 1.1 Hostafloat feeding according to metal content in feeding**

| Feeding | | Hostafloat feeding to | | | Total | Weight | Dosing | Metal | Dosing | Dosing | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Tonnage t/h | Grade % | BK1+BU2 cc/min | BK5 cc/min | Ball Mill cc/min | Feeding cc/min | g/min (*) | g/t | Copper t/h | g/t (Cu) | BK1+BU2 cc/t (Cu) | BK5 cc/t (Cu) |
| (1) | (2) | (3) | (4) | (5) | (6) = (3)+(4)+(5) | (7) = 0.05*(6) | (8) = (7)*60/(1) | (9) = (1)*(2)/100 | (10) = (7)*60/(9) | (11) = (3)*60/(9) | (12) = (4)*60/(9) |
| 70 | 0.8 | 470 | 70 | 70 | 610 | 30.50 | 26.14 | 0.56 | 3267.86 | 50357.14 | 7500.00 |
| **70** | **0.9** | **540** | **75** | **75** | **690** | **34.50** | **29.57** | **0.63** | **3285.71** | **51428.57** | **7142.86** |
| 70 | 1 | 600 | 80 | 80 | 760 | 38.00 | 32.57 | 0.70 | 3257.14 | 51428.57 | 6857.14 |
| **70** | **1.1** | **660** | **90** | **90** | **840** | **42.00** | **36.00** | **0.77** | **3272.73** | **51428.57** | **7012.99** |
| 70 | 1.2 | 720 | 100 | 100 | 920 | 46.00 | 39.43 | 0.84 | 3285.71 | 51428.57 | 7142.86 |
| **70** | **1.3** | **780** | **110** | **110** | **1000** | **50.00** | **42.86** | **0.91** | **3296.70** | **51428.57** | **7252.75** |
| 70 | 1.4 | 830 | 120 | 120 | 1070 | 53.50 | 45.86 | 0.98 | 3275.51 | 50816.33 | 7346.94 |
| **70** | **1.5** | **890** | **130** | **130** | **1150** | **57.50** | **49.29** | **1.05** | **3285.71** | **50857.14** | **7428.57** |
| 70 | 1.6 | 950 | 140 | 140 | 1230 | 61.50 | 52.71 | 1.12 | 3294.64 | 50892.86 | 7500.00 |
| **70** | **1.7** | **1020** | **150** | **150** | **1320** | **66.00** | **56.57** | **1.19** | **3327.73** | **51428.57** | **7563.03** |
| 70 | 1.8 | 1070 | 160 | 160 | 1390 | 69.50 | 59.57 | 1.26 | 3309.52 | 50952.38 | 7619.05 |
| **70** | **1.9** | **1135** | **165** | **165** | **1465** | **73.25** | **62.79** | **1.33** | **3304.51** | **51203.01** | **7443.61** |
| 70 | 2 | 1190 | 170 | 170 | 1530 | 76.50 | 65.57 | 1.40 | 3278.57 | 51000.00 | 7285.71 |
| **70** | **2.1** | **1240** | **180** | **180** | **1600** | **80.00** | **68.57** | **1.47** | **3265.31** | **50612.24** | **7346.94** |
| 70 | 2.2 | 1310 | 190 | 190 | 1690 | 84.50 | 72.43 | 1.54 | 3292.21 | 51038.96 | 7402.60 |
| **70** | **2.3** | **1370** | **200** | **200** | **1770** | **88.50** | **75.86** | **1.61** | **3298.14** | **51055.90** | **7453.42** |
| 70 | 2.4 | 1430 | 210 | 210 | 1850 | 92.50 | 79.29 | 1.68 | 3303.57 | 51071.43 | 7500.00 |
| **70** | **2.5** | **1490** | **220** | **220** | **1930** | **96.50** | **82.71** | **1.75** | **3308.57** | **51085.71** | **7542.86** |
| 70 | 2.6 | 1560 | 230 | 230 | 2020 | 101.00 | 86.57 | 1.82 | 3329.67 | 51428.57 | 7582.42 |
| **70** | **2.7** | **1620** | **240** | **240** | **2100** | **105.00** | **90.00** | **1.89** | **3333.33** | **51428.57** | **7619.05** |
| 70 | 2.8 | 1680 | 250 | 250 | 2180 | 109.00 | 93.43 | 1.96 | 3336.73 | 51428.57 | 7653.06 |
| **70** | **2.9** | **1740** | **260** | **260** | **2260** | **113.00** | **96.86** | **2.03** | **3339.90** | **51428.57** | **7684.73** |
| 70 | 3 | 1800 | 270 | 270 | 2340 | 117.00 | 100.29 | 2.10 | 3342.86 | 51428.57 | 7714.29 |
| **70** | **3.1** | **1860** | **280** | **280** | **2420** | **121.00** | **103.71** | **2.17** | **3345.62** | **51428.57** | **7741.94** |
| 70 | 3.2 | 1920 | 290 | 290 | 2500 | 125.00 | 107.14 | 2.24 | 3348.21 | 51428.57 | 7767.86 |

(*) Hostafloat (5 % by weight) is prepared by mixing with water

**Table 1.1 Continued**

| Feeding | | Hostafloat feeding to | | | Total | Weight | Dosing | Metal | Dosing | Dosing | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Tonnage t/h | Grade % | BK1+BU2 cc/min | BK5 cc/min | Ball Mill cc/min | Feeding cc/min | g/min (*) | g/t | Copper t/h | g/t (Cu) | BK1+BU2 cc/t (Cu) | BK5 cc/t (Cu) |
| (1) | (2) | (3) | (4) | (5) | (6) = (3)+(4)+(5) | (7) = 0.05*(6) | (8) = (7)*60/(1) | (9) = (1)*(2)/100 | (10) = (7)*60/(9) | (11) = (3)*60/(9) | (12) = (4)*60/(9) |
| **70** | **3.3** | **1980** | **300** | **300** | **2580** | **129.00** | **110.57** | **2.31** | **3350.65** | **51428.57** | **7792.21** |
| 70 | 3.4 | 2040 | 310 | 310 | 2660 | 133.00 | 114.00 | 2.38 | 3352.94 | 51428.57 | 7815.13 |
| **70** | **3.5** | **2100** | **320** | **320** | **2740** | **137.00** | **117.43** | **2.45** | **3355.10** | **51428.57** | **7836.73** |
| 140 | 0.8 | 940 | 140 | 140 | 1220 | 61.00 | 26.14 | 1.12 | 3267.86 | 50357.14 | 7500.00 |
| **140** | **0.9** | **1080** | **150** | **150** | **1380** | **69.00** | **29.57** | **1.26** | **3285.71** | **51428.57** | **7142.86** |
| 140 | 1 | 1200 | 160 | 160 | 1520 | 76.00 | 32.57 | 1.40 | 3257.14 | 51428.57 | 6857.14 |
| **140** | **1.1** | **1320** | **180** | **180** | **1680** | **84.00** | **36.00** | **1.54** | **3272.73** | **51428.57** | **7012.99** |
| 140 | 1.2 | 1440 | 200 | 200 | 1840 | 92.00 | 39.43 | 1.68 | 3285.71 | 51428.57 | 7142.86 |
| **140** | **1.3** | **1560** | **220** | **220** | **2000** | **100.00** | **42.86** | **1.82** | **3296.70** | **51428.57** | **7252.75** |
| 140 | 1.4 | 1660 | 240 | 240 | 2140 | 107.00 | 45.86 | 1.96 | 3275.51 | 50816.33 | 7346.94 |
| **140** | **1.5** | **1780** | **260** | **260** | **2300** | **115.00** | **49.29** | **2.10** | **3285.71** | **50857.14** | **7428.57** |
| 140 | 1.6 | 1900 | 280 | 280 | 2460 | 123.00 | 52.71 | 2.24 | 3294.64 | 50892.86 | 7500.00 |
| **140** | **1.7** | **2040** | **300** | **300** | **2640** | **132.00** | **56.57** | **2.38** | **3327.73** | **51428.57** | **7563.03** |
| 120 | 1.7 | 1750 | 260 | 260 | 2270 | 113.50 | 56.75 | 2.04 | 3338.24 | 51470.59 | 7647.06 |
| **120** | **1.8** | **1830** | **275** | **275** | **2380** | **119.00** | **59.50** | **2.16** | **3305.56** | **50833.33** | **7638.89** |
| 120 | 1.9 | 1950 | 285 | 285 | 2520 | 126.00 | 63.00 | 2.28 | 3315.79 | 51315.79 | 7500.00 |
| **120** | **2** | **2050** | **290** | **290** | **2630** | **131.50** | **65.75** | **2.40** | **3287.50** | **51250.00** | **7250.00** |
| 100 | 2 | 1700 | 240 | 240 | 2180 | 109.00 | 65.40 | 2.00 | 3270.00 | 51000.00 | 7200.00 |
| **100** | **2.1** | **1770** | **260** | **260** | **2290** | **114.50** | **68.70** | **2.10** | **3271.43** | **50571.43** | **7428.57** |
| 100 | 2.2 | 1870 | 270 | 270 | 2410 | 120.50 | 72.30 | 2.20 | 3286.36 | 51000.00 | 7363.64 |
| **100** | **2.3** | **1960** | **285** | **285** | **2530** | **126.50** | **75.90** | **2.30** | **3300.00** | **51130.43** | **7434.78** |
| 100 | 2.4 | 2050 | 300 | 300 | 2650 | 132.50 | 79.50 | 2.40 | 3312.50 | 51250.00 | 7500.00 |
| **100** | **2.5** | **2130** | **320** | **320** | **2770** | **138.50** | **83.10** | **2.50** | **3324.00** | **51120.00** | **7680.00** |
| | | | | | | | **Average** | | 3302.20 | 51171.60 | 7436.22 |

(*) Hostafloat (5 % by weight) is prepared by mixing with water

### 1.3. Improvement and Automation Studies on Kure Concentrator

Kure concentrator has been put on operation for the purpose of production of 450.000 tons of pyrite concentrate containing 46% S, 60.000 tons of Cu concentrate with 15% Cu content and processing of 920.000 tpy raw ore containing 1.71% Cu and 37% S. The proved reserve is 12 Mt.

Kure orebody consists of 47 % impregnant copper (0.7 - 1.2 % Cu), 45 % copperous pyrite (1.2 - 1.7 % Cu) and 8 % massive copper (1.7 - 2.5 % Cu). The grade of raw ore fed to concentrator is 1.71 % Cu. Ore which is to be fed to plant, is a mixture of three different types of copper ore that are mentioned above. The rate of liberation changes between 1 to 74 microns.

Prior to improvement work of Kure concentrator: cyclone overflow had been subjected to thickening with reactives for 20 minutes in four conditioners which have a total volume of 100 $m^3$. Pulp that were thickened in conditioners are first subjected to rougher circuit which is formed of three unit having a total volume of 192 $m^3$ and the flotation time was 40 minutes. The product taken from rough flotation was send to ball mill to regrind and tailing was brought to pyrite flotation . Having left the cyclones, rough concentration product that was grinded by small ball mill (BM3) enters 1st. Cleaning flotation. The volume of first cleaning circuit is 192 $m^3$ and flotation time is 45 minutes. First cleaning product is sent to 2nd. Cleaning and 3rd. Cleaning flotation respectively, thus after 40 minutes of additional cleaning time final product is taken from 3rd. Cleaning. Return product of 2nd. And 3rd cleaning runs as closed circuit while 1st. Cleaning` remaining go to waste.

As a result of observations it was understood that cleaning circuits were insufficient. For this reason 3rd. Pyrite flotation unit (this unit is formed 4 cell, 64 $m^3$) is pulled apart and added to copper flotation circuit as 4th. Cleaning unit. Thus flotation extension time was 20 minutes. Because of getting high grade concentrate from rougher circuit 1st. Unit, product was directly sent to 2nd. Cleaning without regrinding and first cleaning. The product that was produced from1. Cleaning was re-send to grinding unit to be re-grinding. It was assumed that tailing of 1st. Cleaning was considered as semi-product and instead of sending to waste dam, it was re-fed to grinding circuit (see Appendix V).

Another important change in the flotation unit is the automatic feed of utilized reactives in different points. Hostafloat used as collector is previously used to be fed with only two points. Feeding points are tested, with increasing numbers and promising result were obtained. Thus collector feeding points in copper flotation were increased to 5 and they were started to be fed according to entrance grade and amount.

Dowfroth is used as foaming material for both copper and pyrite and KAX feeding used as collector in pyrite circuit are carried out on computer situated on control board. A control loop circuit was established in order to balance the pH of pyrite circuit. pH is read from pyrite conditioner and acid feeding is automatically done according to required pH values. All motors in flotation circuit are connected to signalization line. In addition pulp levels and air flows are also controlled by control loops.

### 1.4. The Automation System Used in Kure Concentrator

To be able to observe all machines that work in the plant at one center, to bring the number of breakdowns to a minimum and to lesser the human factor on every stage of the process, it was decided to automate the system in 1992 and initial works commence in July 1993 (Arikan, A. T., 1992 - 1995).

It can be said that PROSCON 2100 is a system in which process is controlled, methodized and data are collected. Proscon 2100 is connected to the process with Application Control Station (AC-Station). This station uses measurements, digital signals and control data. Beside AC-station collect the informations that comes from automatically working analysors. In the plant grade analysis apparatus (Courier 30AP) and particle sizer (PSI) are directly connected to AC-Station.

#### 1.4.1. Features of PROSCON 2100 System

Proscon 2100 is formed of 4 main station. Function of this system is shared by these stations:

**- Application Control Station (AC-Station) :**

In the Proscon 2100 system application control station connects the process to control system, AC-Station controls input-output, measurements, data signals and control outputs. AC-Station performs the followings :

- Obtaining analogous measurements
- Obtaining analogous outputs
- Controls circuit functions
- Fast calculations
- Locking and aligning controls
- Communications with other stations by Genious LAN
- Carrying out the control programs adaptable to the process.

**- Process Management Station (PM-Station) :**

Proscon 2100 system performs the followings :

- It enables the operators to control the process on colored screens
- To collect the process data, calculated values and alarms as data as data base for later analyse and report works
- Sending the reports to printers and discs
- It is used in system configurations
- Let the process operators know about alarm situations
- Communications with other Proscon stations
- Updating AC-Station parameters
- Contains software protection, configuration and graphic editors functions.

**- Plant Operator Station (PO-Station)**

Proscon 2100 PO-Station performs the followings :

- Editing and saving the control programs
- It is used for configuration of AC-Station
- It is possible that the configuration of PM-Station can be done by this station
- Contains plant's maintenance and daily bulletin programs.

**- Engineering Workstation (EW-Station)**

Proscon 2100 EW-Station performs the followings

- Edit and save control programs
- Used the configuration of AC-Station
- Configuration of PM-Station can also be done by this station
- Can be used other all engineering studies.

### 1.4.2. Application Examples of PROSCON 2100 In Kure Concentrator

Figure 1.1 shows main menu of process control system. Moreover both section contains indows. Figure 1.2 shows an opened windows that was prepared to control of copper flotation cells air consumptions. These window enables process operator to observe and change set point values. Figure 1.3 shows some windows and pop-up window of which are calculate solid content in RM in %. M48 shows measured value of process water flow to RM. Input comes to M48, from FM-2104 (flow-meter). Measured value is used in C21 and then solid content is calculated as percentage. In C21, formula is written by engineers. In control loop FIC-2104, measured value (M48) is compared with the set point of process water flow to RM and then a signal is sent to FV-2103 (plug valve) for opening or closing the valve to obtain desired value for water flow.

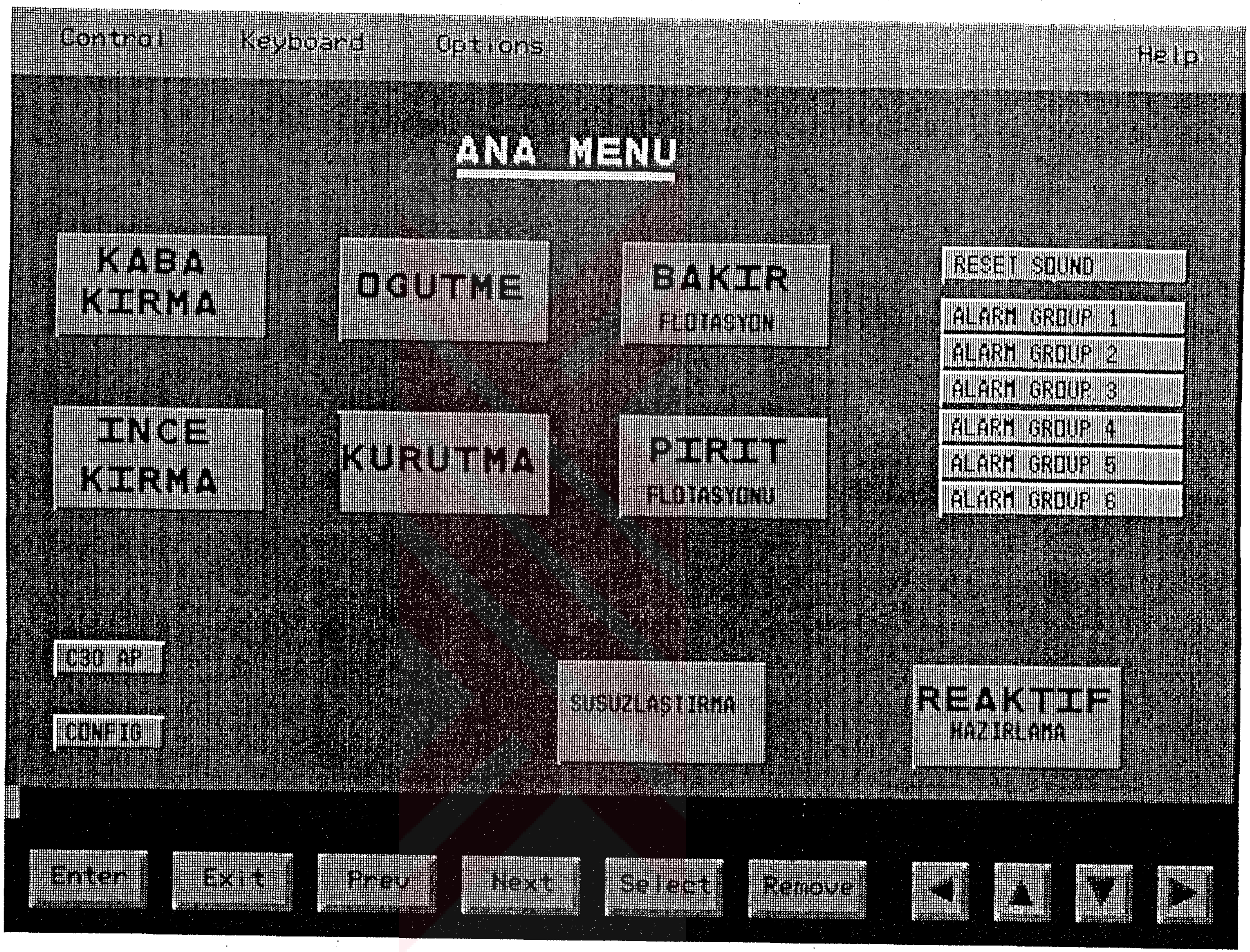



Figure 1.1 Main menu.

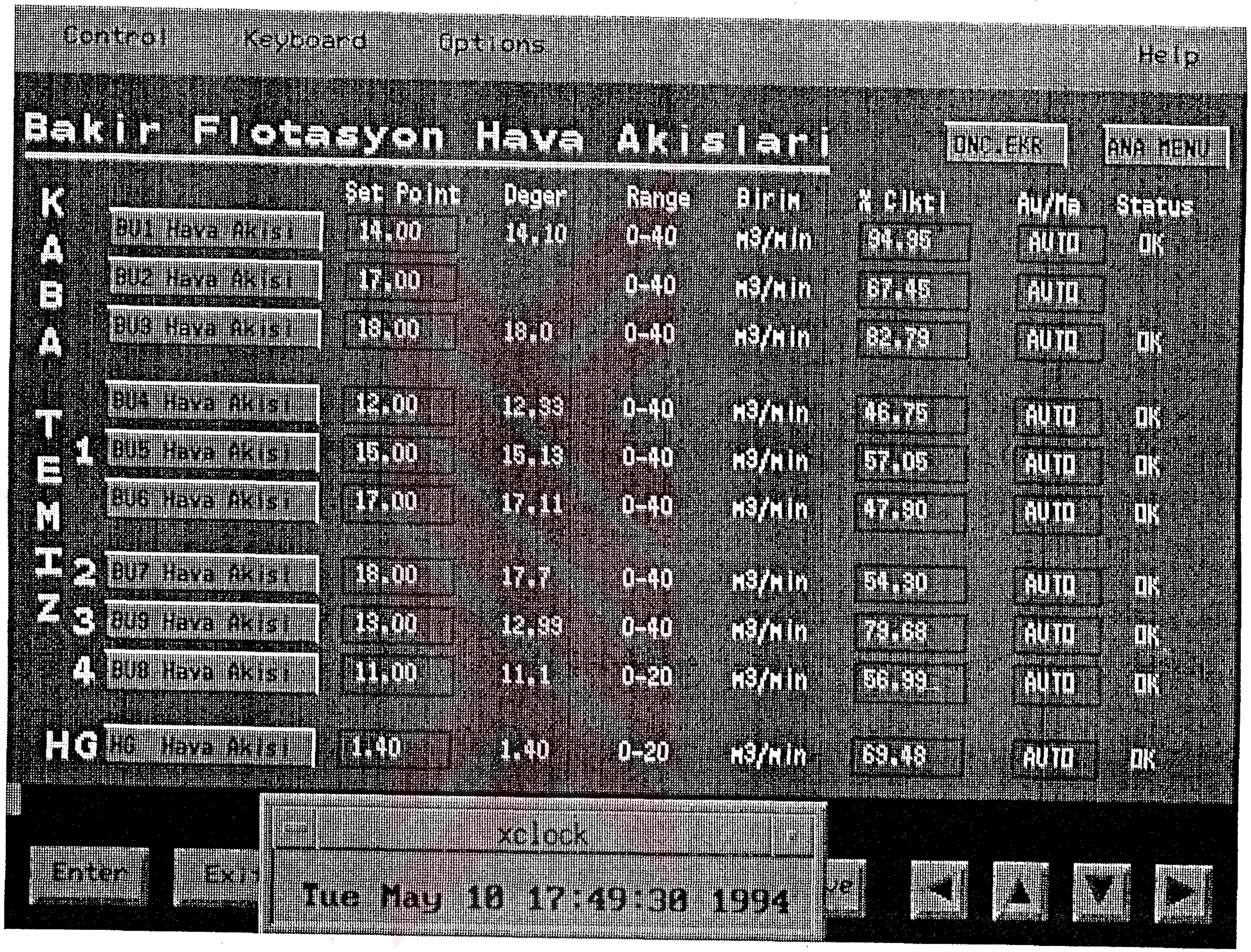



**Figure 1.2 Air consumption of copper circuits.**

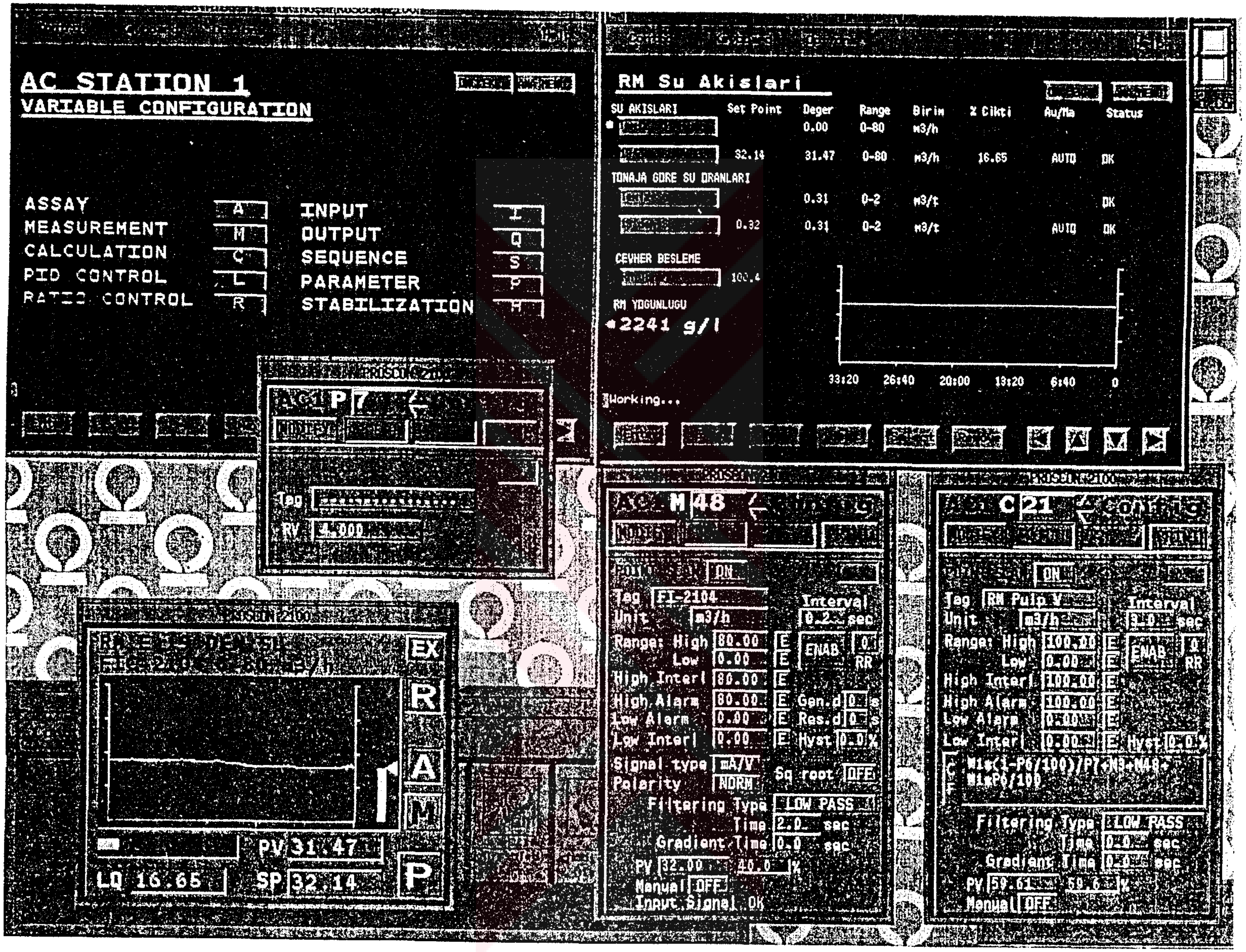



Figure 1.3 Controlling of process water flow to RM.

### 1.5. Troubles Encountered in Kure Concentrator Before Automation

1 - Problem caused by copper being found mineralogically in pyrite crystal structure and very fine ( - 38 μ ) liberation size: a lot of problems were faced before automation in grinding circuit which is the first step in preparation of flotation conditions. Not being able to stabilize cyclone feed density, in most cases not being able to determine correctly whether the milling product is in liberation particle size which counts on operator's experience and as a result of bad or excess grinding formation of slime.

2- Fluctuation in copper concentrate grade has brought the effectiveness of the plant into discussion. Maximum concentrate grade which was reached before automation was 15 % Cu. It is known that it sometimes went down to 12 % Cu.

3- Increase in labor expense of production cost as time passes by.

4- It is very hard and painstaking to establish optimum flotation conditions and hold these conditions stable. To get rid off the inconvenience happening at flotation conditions, a struggle takes place against time from grinding circuit to final concentrate cell. In this process when optimum conditions are provided for any grade, a different ore can be fed to the circuit because of inhomogenity.

5- Since it is impossible to estimate possible future problems, dimensions of the area increase and result in time and money waste. For examples Kure concentrator is a three story plant and pulp overflows in tanks which are situated on the bottom floor, used to be realized after minutes due to lack of attention and as a result of this event hundred of $m^3$ of pulp used to flow to tailing disposal.

6- Control of flotation was in the initiative of the personnel, interference of each operator was different, and obtained results used to be different from shift to shift. For example during the shift of an attentive operator, concentrate grade can be 15 % Cu, while it can be 12 % Cu in another shift.

7- Since flotation is a continuos process, grade of concentrate and tailing should be known simultaneously. In conventional method, laboratory results used to come to plant after 5 - 6 hours. Interference to the process during this period was almost impossible. Because measures, which were to be taken, used to be decided after this results had come. Before automation laboratory expenses used to be high and time used to be wasted for analysis.

8- Production reports of each shift used to be past to the following shift. As there was no efficient and fast recording system, collection of datas which is the basis of backward researches to be done, did not used to be reliable. Particularly during night shifts, concentrate grade used to decrease and during the day engineers who seek for the reason, have difficulties in finding the cause.

9- To solve the break-downs and to stabilize the process conditions engineers used to spend too much time. They did not used to have time for future researches.

10- As a result of frequent changes of chemical and physical properties of ore coming from mines, time wasting used to occur by manual control of the process conditions.

**1.6. Results Obtained After Automation of Kure Concentrator**

1- Particle size of grounded ore can be determined instantly and reliably by particle size indicator (PSI). Hence an effective grinding can be achieved and slime formation is suppressed. Measurement of cyclone feeding density is carried out by density-meter and to hold it at a desired value, water addition to tanks is done by automatic valves. Operator sees these values on computer screen and when necessary he interference by using only keyboard.

2- Concentrate grade is now between 16 - 18 % Cu and it can meet the world desired level. Especially copper grade in tailing is stabilized and hold between 0.2 - 0.3 % Cu. Before automation this value was 0.5 - 0.7 % Cu.

3- Process conditions are improved by automation and since interference are done in computer media, time is saved and costs are lowered.

4- An economic level is reached by reducing the number of workers working in production and by employing these workers in repair and maintenance works, both trouble shouting has become effective and protective care system is introduced. Before automation, trouble shouting used to take much longer time because of insufficent staff and no time was available for protective care.

5- Technological failures can be seen on computer screen and a warning sound is also available for instant interference.

6- As the control of process is performed by instruments, Human originated faults are minimized.

7- Copper rate in concentrate and tailing can be detected immediately by x-ray analyzer and it is more reliable comparing to laboratory, analysing.

8- Every single step of the process is recorded by the computer. Hence previous results are easily available and mistakes done in the past can be scrutinized by engineers.

9- By reducing break downs and stops, plant engineers had more time for research and development.

10- Changes in physical and chemical properties of ore is determined instantly and feeding rates are adjusted automatically.

### 1.7. Examples of Some Automated Copper Concentrators

In Automation of copper flotation, Outokumpu Oy. is a well known company. In that case some of the concentrators of Outokumpu Oy. is being examined in this section.

### 1.7.1. Pyhäsalmi Concentrator

Pyhäsalmi Concentrator is situated in Pyhäsalmi in Finland and processes about 1 Mt/y of ore containing mixed sulphides (see Figure 1.4). This company is a member of Outokumpu Group.

The following data illustrates some ore characteristics essential to know for the concentration process (Arikan, A.T., 1983) :

- The average composition of the mill feed is:

| | |
|---|---|
| Pyrite | 60 - 80 % |
| Sphalerite | 2 - 7 % |
| Chalcopyrite | 2 - 3 % |
| Barite | 3 - 8 % |
| Others | 10 - 30 % |

- Sericite quartzite, cordierite gneiss, felcit volcanics and amphibolite are the main gangue rocks.

- Pyrrhotite, galena, arsenopyrite, magnetit, electrum metallic gold and certain sulphosalts are among the accessory minerals.

- The average feed is 0.8 % Cu, 2.2 % Zn, 38 % S, 0.45 gr/t Au, 15 gr/t Ag.

- The specific gravity of ore varies between 4.0 - 4.5.

- Pyrite occurs as coarse grains and chalcopyrite and sphalerite form a fine grained matrix between pyrite crystals.

- Chalcopyrite is partly disseminated in country rock fragments.

- The ore is partly brittle and partly tough: the brittle material fractures easily down to pyrite crystals.

- In general the ore is very abrasive and hard to grind.

- Most of the copper and zinc in the surface part of orebody occurred as secondary copper sulphides and iron-copper-zinc sulphates.

- Some copper dissolves easily into the mine water, which then activates sphalerite surface.

- There are sudden variation in the ore grade, in the degree of alteration and in the grindabilty.

- The upper part of the orebody was mined from an open pit; nowadays the ore is mined underground.

Production started in 1962. About 1 Mt/y of ore are processed. A separate circuit is used in the concentrator for processing the ores from small orebodies in the vicinity. The average production of copper concentrate is 30.000 tpy, of zinc concentrate 40.000 tpy and of pyrite concentrate 600.000 - 700.000 tpy.

The flotation arrangement is direct, selective flotation consisting of three sequential flotation circuits, one each for copper, zinc and pyrite. The copper circuit includes roughing, cleaning and scavenging stages. In the zinc circuit, the first stage is open circuit roughing and scavenging. The scavenger concentrates and cleaner tailings are pumped to another similar bank of cells, i.e. the middling flotation. Rougher concentrates from both banks are pumped to cleaning, which has four stages. The pyrite circuit consists of open roughing and cleaning operations. Dewatering of sphalerite flotation tailings before acid circuit pyrite flotation is essential (Miettunen, J., 1993).

The greatest problem during copper flotation is the sphalerite activates by the copper-bearing mine water. Although cyanide and zinc sulphide are used as depressants, the zinc concent of the copper concentrate is still quite high. A flotation time of 20 minutes is required for satisfactory Cu-recovery.

In the original zinc circuit the circulating loads tented to be excessively large. The present flowsheet with open circuit roughing compared to the copper flowsheet improved the control of the circulating loads. Pyrrhotite, which occasionally appears in the ore, tends to float with sphalerite, causing a reduction in the grade of the zinc concentrate.

Over the years it has been found that the concentratability of the Pyhäsalmi ore varies considerably. Grinding to a certain fineness does not, for example guarantee constant metallurgical results. What is even more important is that constant metallurgical result are also guarantee of maximum profit. As a Consequence of this, process control at the highest level is carried out today according to economic criteria.

In the Outokumpu company the Pyhäsalmi Concentrator has often been the test site and the first to utilize new techiques. Computer control was started as early as 1969 after the tests of the first on-stream analyser were successful. Today the process is equipped with modern devices and versatile instruments with two Courier 300 analysers (28 streams) and a Proscon 20/200 control system. There are about 600 measuring points and 200 control points supervised by Proscon.

Since 1970 the computer has been used continuously for the control of grinding and the copper and zinc flotation circuits. Computer control was subsequently introduced into the pyrite circuit in 1982.

The stabilizing controls of copper and zinc flotation are based on Courier analyses and three froth level measuremens. The controllability of zinc flotation improved significantly after froth level measurements were started at the beginning of 1987.

A system for controlling the economy of the whole mine has recently been developed. The economic indicators needed for this are obtained by transforming the metallurgical results into economic results, and by measuring these and the cost of whole operation continuously.

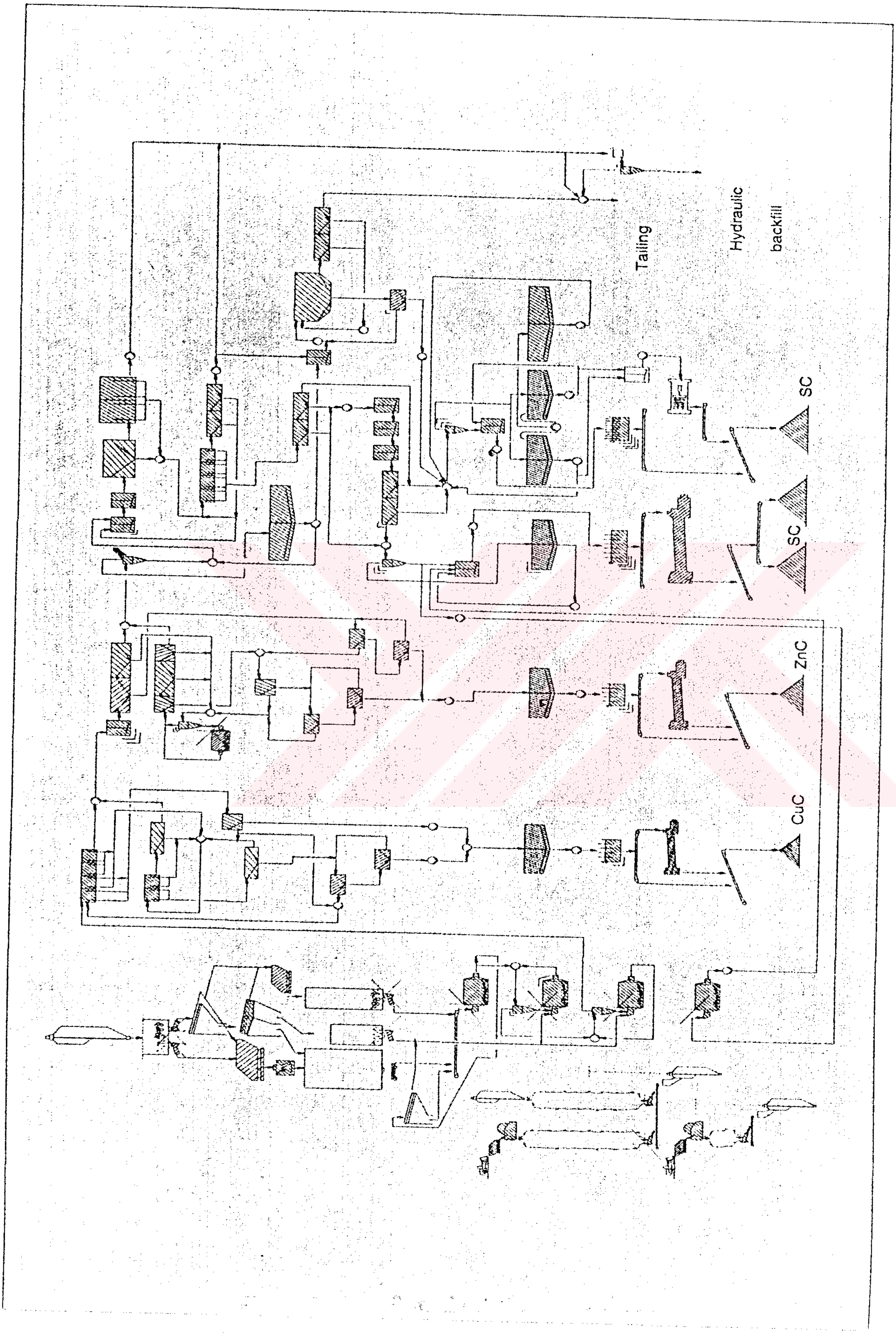



**Figure 1.4. The flow sheet of Pyhäsalmi concentrator**

### 1.7.2. Vuonos Concentrator

A computer control system was installed at Vuonos in 1973 for start-up operations. Vuonos produced nickel and copper concentrates from 6.000 tpd ore. Stabilizing control was applied to hold the combined flow rate of copper cleaner tailings and middlings (from scavenger flotation) in balance. Control variables are cell aeration rate to each of rougher, scavenger, and cleaner cells. Manual setpoints of combined cleaner tealings-middlings flow, and also rougher concentrate flow, are maintained by cascade loops. A manual setpoint on copper concentrate grade is maintained by lime and NaCN feed control in both the cleaner and rougher circuits.

The principals distrubance to flotation metallurgy is a large and varying pyrrhotite feed causing excessive circulating loads to develop if not controlled. The result of stabilizing circulation loads to balance flotation conditions which, augmented with reagents controls, has led to reduction in copper assays from 0.105 to 0.076 %, representing a copper recovery improvement of 1.5 % ( SME,Volume 1, p 5- 101, 1985).

### 1.7.3. Keretti Concentrator

In 1989, after 76 years in production, the ore reserves were exhaused and the mine was closed. Total ore production of the mine was 28.35 Mt of ore, from which 952 kt copper, 246 kt zinc, 63 kt cobalt, 18,380 kg gold and 241,915 kg silver were produced (see Figure 1.5).

The ore was dominantly pyrrhotite with Co-bearing pentlandite. It was found in a rock assemblage called Outokumpu association in which serpantinite predominated. The host rock and footwall rock was mostly quartzite. The average grade of the ore was 3.35 % Cu, 0.86 % Zn, 0.22 % Co, 0.6 gr/t Au and 8.5 gr/t Ag.

Feed is 1,500 tpd new feed and up to 3,000 tpd reclaimed tailings feed. Stabilizing flotation control employs assay data in the loop direcly, because mass flow measurements on concentrates are unreliable due to froth conditions. A combination of cell level and aeration rate control loops in the copper flotation circuit attains stabilized middlings and cleaner tailings circulating loads. Off-line simulation studies with plant data led to establishing a combination of ore feed/rougher concentrate/final concentrate assay ration which maximized recovery. Substantial reduction in reagent consumption also was achieved. Improved economics for the Keretti Concentrator were reported in 1975 as $ 425,000 annualy, from which the $ 48,000 annual operating cost for the control system is deducted. An eight-year annual amortization cost of $ 75,000 (at 10 % interest) is used for depreciating the computer system. X-ray analyzer, and other instrumentation investment totaling $ 406,000, yielding $ 283,000 annual profit derived from the control system ( SME,Volume 1, p 5- 102, 1985).

### 1.7.4. Kotalahti Concentrator

Kotalahti produced nickel and copper concentrate in ore feed of about 2,200 tpd(see Figure 1.6). The results of stabilizing control led to increasing nicel and copper recovery by 2 %. Optimizing nickel recovery was initiated in 1975 through pH control, using the economic values of concentrates as primary control criteria ( SME,Volume 1, p 5- 102, 1985).

### 1.7.5. Vihanti Concentrator

Mining activity began in 1951 and production started in 1954. The mine was completely underground and closed in 1992. Total ore production reached 27.9 Mt (see Figure 1.7).

There were three types orebodies comprising zinc ores, pyritic ores, and disseminated chalcopyrite ores. The average grades were 5.2 % Zn, 0.47 % Cu, 8.59 % S, 0.37 % Pb, 8.63 % Fe, 21.8 gr/t Ag and 0.39 gr/t Au.

Over time the main metal concent of the ores had decrase from 10 % Zn in 1960`s to 2.7 % Zn in 1990`s. However automation, mechanization and rational administration guaranteed the contiuation of economical production (Outokumpu , 1985).

Stabilized operation is attained by setpoint adjustment of cascade control loops from assays, with scavenger product and cleaner tailings flows controlled by cell level. Zinc losses in the tailing were reduced with recovery gains approaching 2 % and added benefits from control of zinc concentrate grade.

Datas that are taken from Vihanti Cu - Pb - Zn concentrator in Finland are given below.

Table 1.2. Comparison of datas of Vihanti concentrator (Yazan, H.A., Akar, A., Ozmerih, L.,1974)

| | Ore grade | Cu Concentrate | | | Productivity |
|---|---|---|---|---|---|
| | % Cu | % Cu | % Zn | % Pb | % |
| Before automation | 0.53 | 22.09 | 4.71 | 2.13 | 76.9 |
| After Automation | 0.57 | 24.12 | 3.24 | 1.91 | 79.4 |

Pb and Zn grades in copper concentrate is decrease by automatic control of Vihanti concentrator. On the other hand concentrate grades of Pb and Zn are increased. Also copper productivity is increased by 2.5 % (Yazan, H.A., Akar, A., Ozmerih, L.,1974).

In Canada, Kidd Creek Cu - Pb - Zn flotation plant is also controlled by on-stream x-ray analyzer and a computer. After the system had been installed, grade of concentrates and productivity were increased, consumption of reactives were decreased by 12 %, increase in copper and zinc productivity were 0.8 % and 1.6 % respectively(Yazan, H.A., Akar, A., Ozmerih, L.,1974).

Table 1.3. Outokumpu Oy. concentrators using computer control ( SME,Volume 1, p 5-101, 1985).

| Concentrator | Capacity | Concentrates | Mode of Controls | Results of automation |
|---|---|---|---|---|
| Pyhälsalmi | 2,500 tpd | Cu, Zn, Fe | Stabilizing control 1969and optimizing control 1972 | 2.5 % increase metallurgi. values from optimizing control ($450,000 annual gain) |
| Vuonos | 1,500 tpd copper circuit; 4,500 tpd nickel circuit | Cu, Ni | Direct digital control in 1972 for operation of crushing and materials handling; stabilization control in 1974 | 1.5 % increase copper recovery |
| Katolahti | 2,200 tpd | Ni, Cu | Proscon 103 stabilizing control in 1975 | 2.5 % increase nickel and copper recovery |
| Vihanti | 2.500 tpd | Zn, Pb, Cu | Proscon 103 stabilizing control in 1975 | 2 % increase zinc recovery |
| Keretti | 1,500 tpd new feed; 3,000 tpd reclaimed tails feed | Cu, Zn, Co | Proscon 103 stabilizing control in 1975 | About $200,000 net annual gain in reagents savings, improved recovery |

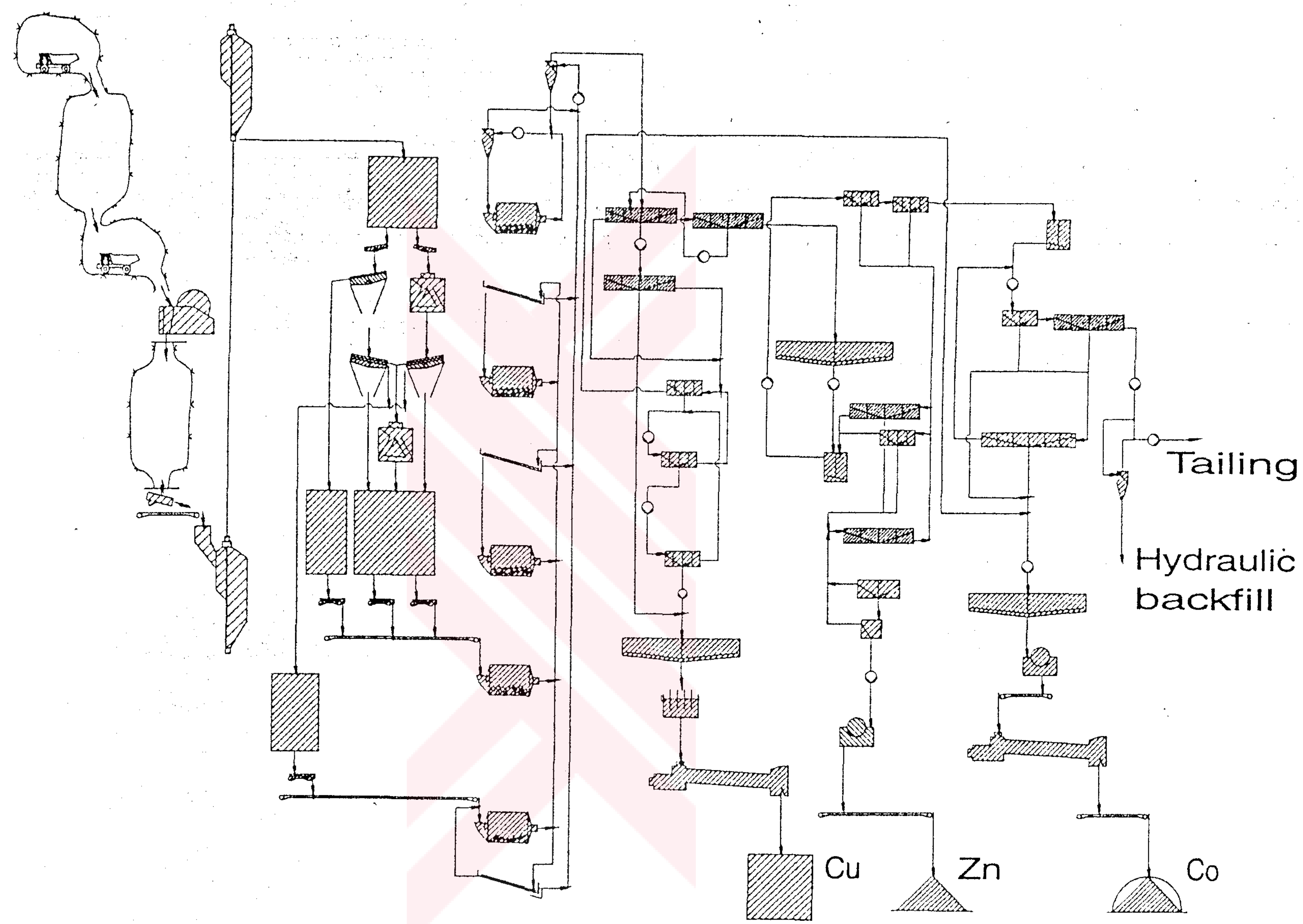



**Figure 1.4. The flow sheet of Keretti concentrator**

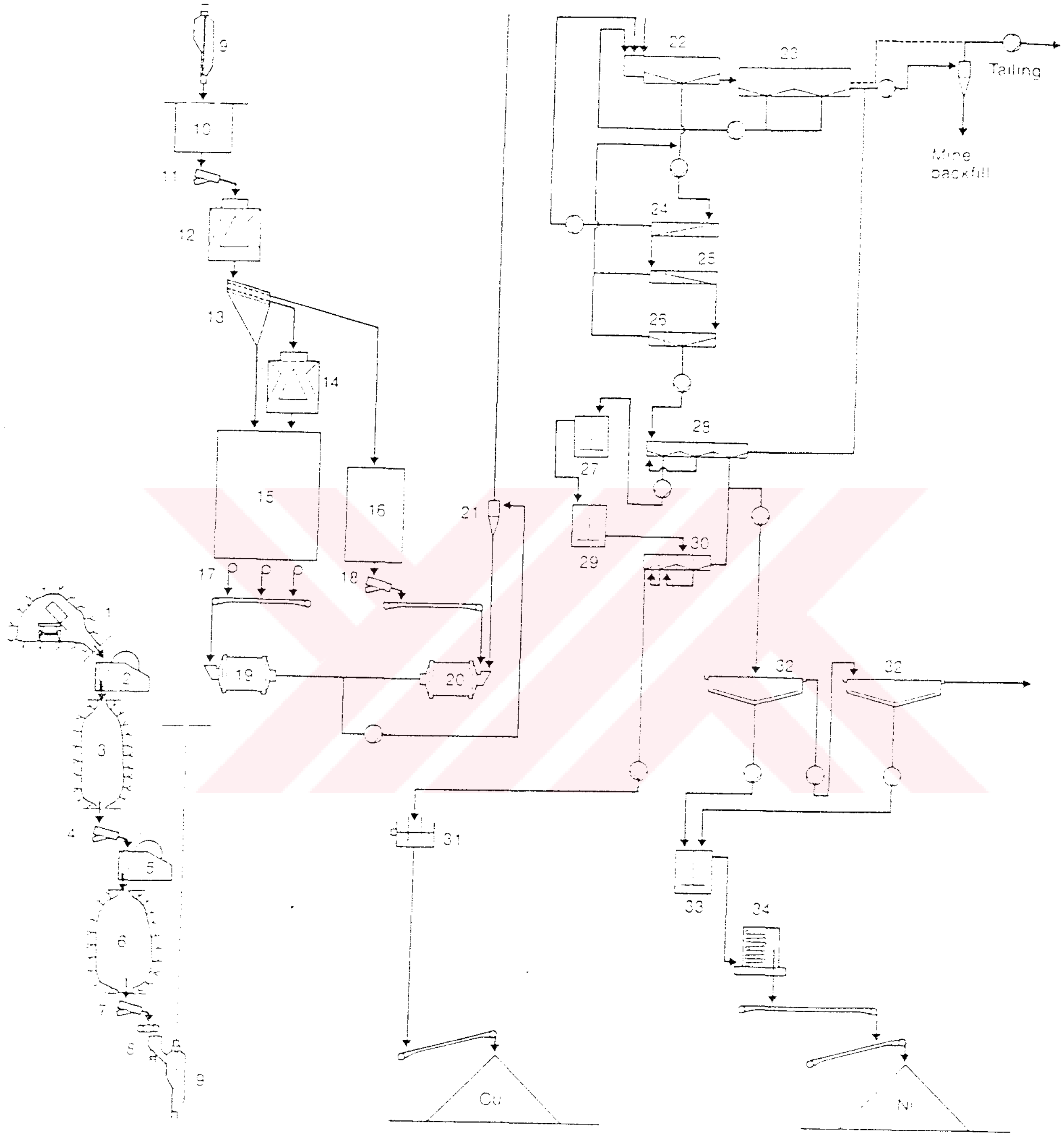



**Figure 1.6 The flow sheet of Kotalahti concentrator**

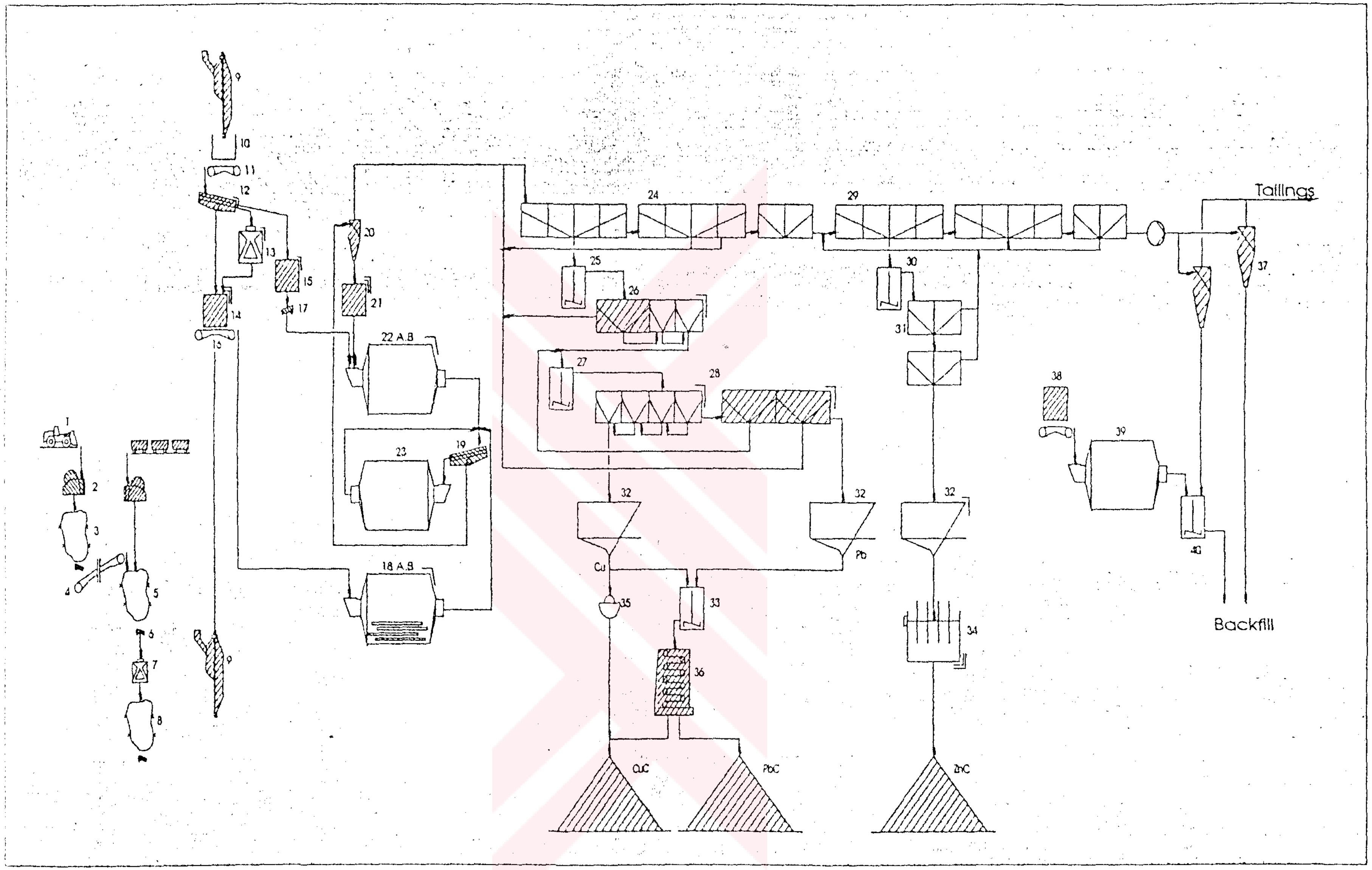



**Figure 1.7 The flow sheet of Vihanti concentrator**

CHAPTER TWO

# CONTROL LOOPS

## 2. General

Important advances have been made since the early1970s in the field of automatic control of mineral processing operations, particularly in grinding, and to a lesser extent in flotation. The main reason for this rapid development are:

**(i)** The development of reliable instrumentation for process control system. On-Line sensors such as flowmeters, density gauges, and chemical composition analysers are of greatest importance, and on-line particle size analysers have been successfully utilised in flotation circuit control.

Other important sensors are pH meters, and level and pressure transducers, all of which provide a signal relating to the measurement of the particular process variable. This allows the final control elements, such as servo valves, variable speed motors and pumps, to manipulate the process variable based on a signal from the controller. These sensors and final control elements are used in many industries beside the minerals industry.

**(ii)** The availability of sophisticated digital computers at very low cost.

During the 1970s the real cost of computing virtually halved each year, and the development of the microprocessor allowed very powerful computer hardware to be housed in increasingly smaller units. The development of high-level languages allowed relatively easy access to software, providing a more flexible approach to changes in control strategy within a particular circuit.

**(iii)** A more thorough knowledge of process behaviour, which has led to more reliable mathematical models of various important unit processes being developed. Many of mathematical models which have been developed theoretically, or “off-line” have had limited value in automatic control, the most successful models having been developed “on-line” by empirical means. Often the improved knowledge of the process gained during the development of the model has led to improved techniques for the control of system.

**(iv)** The increasing use of very large flotation cells has facilitated control, and reduced the amount of instrumentation required.

Financial models have been developed for the calculation of costs and benefits of the installation of automatic control systems, and benefits reported include significant energy savings, increased metallurgical efficiency and throughput, and decreased consumption of reagents, as well as increased process stability.

The concept, terminology, and practice of process control in mineral processing have been comprehensively reviewed by Ulsoy and Sastry; in relation to crushing and grinding by Lynch, and in relation to flotation by Lynch and Wills (Wills, B.A., 1985).

### 2.1. Control Loops

The basic function of a control system is to stabilise the process performance at a desired level, by preventing or compensating for disturbances to the system. Ultimately the objective is not only stabilisation but also optimisation of process performance based on economic considerations.

In order to achieve these objectives, there are various "levels" of control used to make up a "distributed hierarchial computer system". At the lowest level-regulatory control-the process

variables are controlled at defined "set-points" by the various individual control loops.

#### 2.1.1. Feed - Back Control Loop

A simple feed-back control loop, which can be used to control the level of slurry in a sump is shown in figure 2.1 The signal from the level transmitter is fed to the controller where the measured value is compared with the desired value. The output signal is controlled by a standard control algorithm which is most commonly a proportional plus integral plus derivative (PID) algorithm, although computers have made possible the implementation of more complex algorithms (Wills, B.A., 1985).

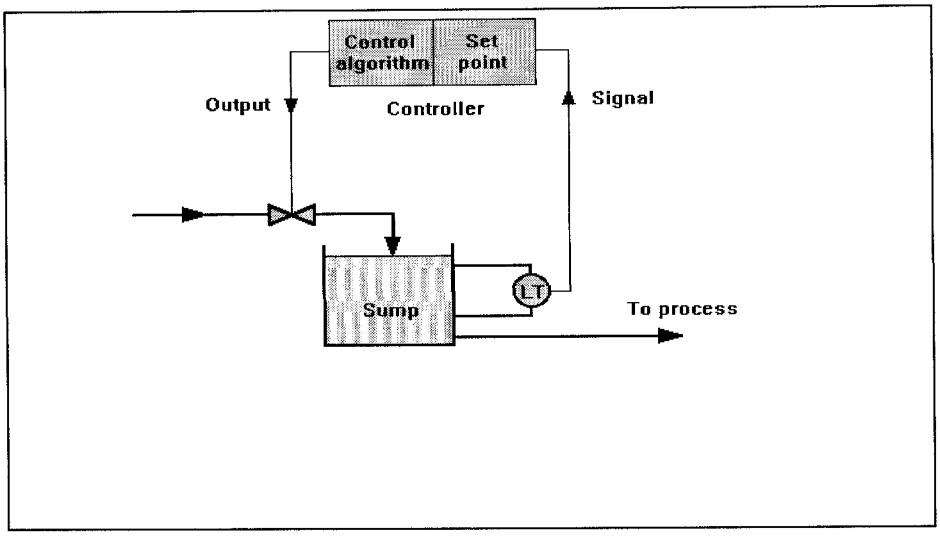



**Figure 2.1 Simple feed - back control loop.**

Although the various process may be controlled by separate analogue controllers, it is now more common to control the variables at their set-points by a process computer. The requirement of such direct digital control is that the analogue signals from the measuring devices are converted into digital pulses by an analogue-digital converter, and that the digital signals from the computer are converted to analogue form and transmitted to the transducers which convert the analogue signals to mechanical form in order to operate the control devices (Wills, B.A., 1985).

Although set-points may be adjusted manually, it is preferable to control them automatically by other computers in the hierarchy. Such supervisory, or cascade control may use feed-back loops, such as in the regulation of reagent feed to a flotation process in response to metal content of the flotation tailing ( see Figure 2.2).

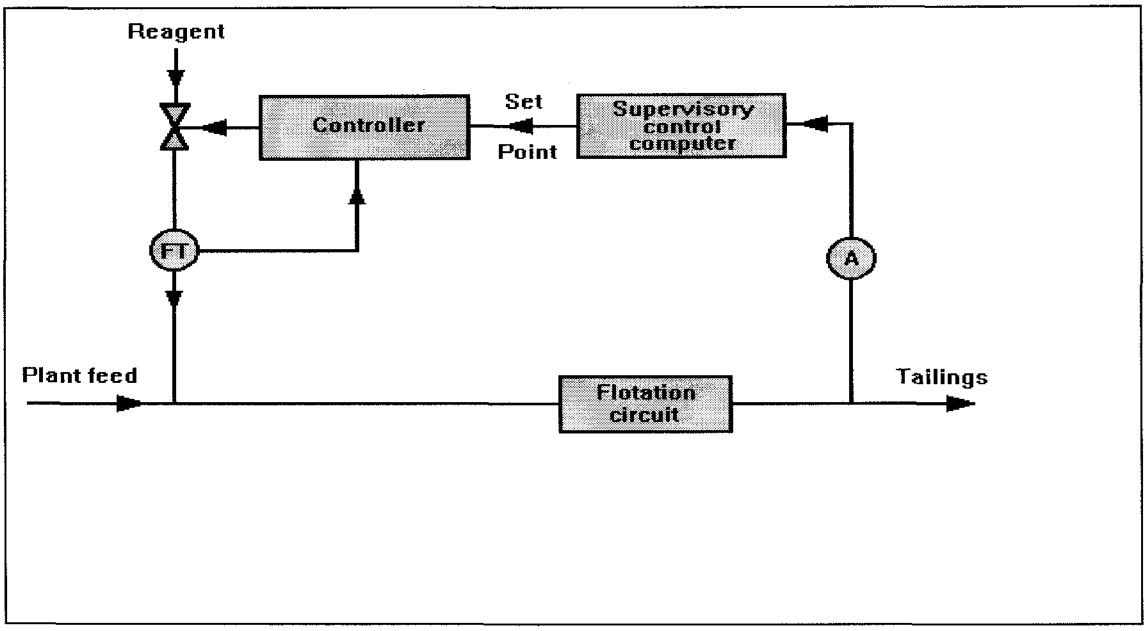



**Figure 2.2 Feed - back loop with supervisory control.**

A change in the metal content of the tailing is acted upon by the supervisory computer which, by means of a process algorithm, modifies the set point used by the process computer in the regulatory control loop. The greatest problem found, which has never been fully overcome in flotation control, is in developing algorithms which accommodate change in ore type and which can define flexible limits to the maximum and minimum amounts of reagent added (Wills, B.A., 1985).

### 2.1.2. Feed - Forward Control

The disadvantage of feed-back control loops is that they compensate for disturbances in the system only after the disturbances have occurred, and the effect of the control system on the manipulated variables is not observed until after a time delay roughly equivalent to the residence time of the process flow between the control device and the measuring device (Wills, B.A., 1985).

Feed-forward control loops do not suffer from this disadvantage, and they are sometimes used in flotation processes to control the addition of a specific reagent to the process feed. Such a control loop is shown in Figure 2.3.

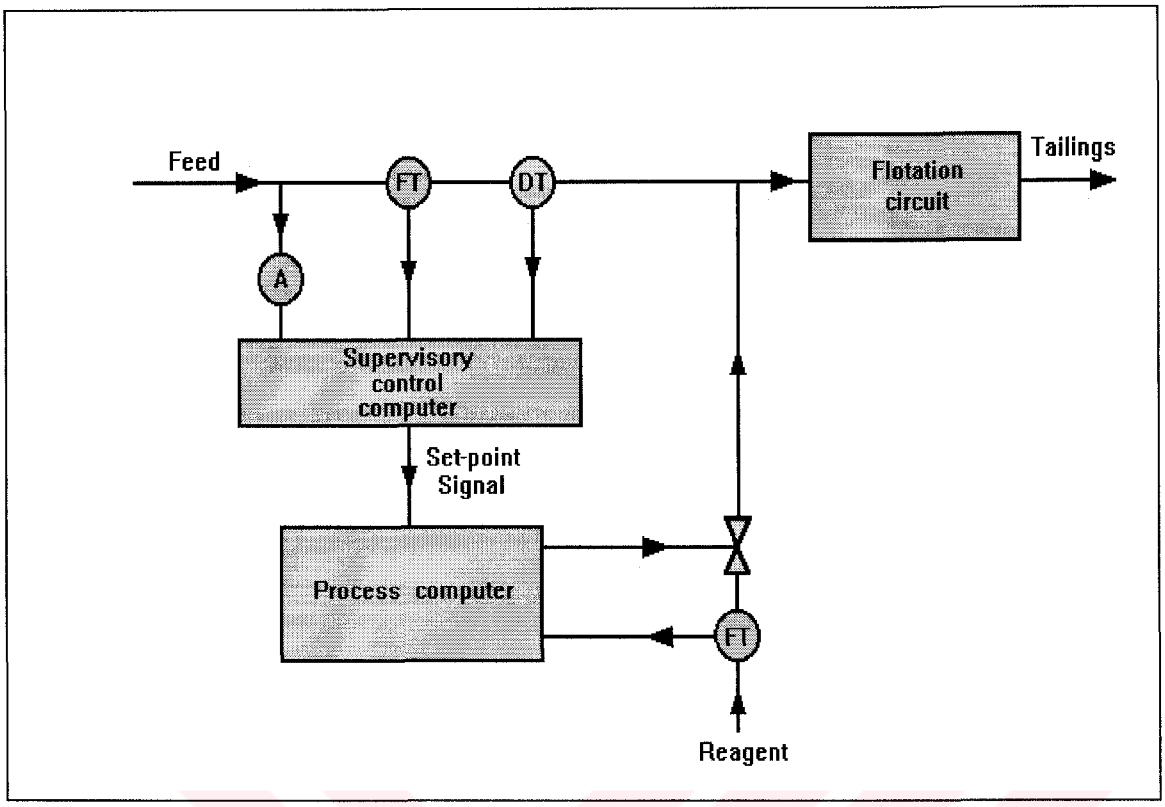



**Figure 2.3 Feed - forward control loop.**

The flowrate of slurry, and slurry density are continuously measured by means of magnetic flowmeter and density gauge respectively, the two signals being computed to mass flowrate of dry solids. The chemical analysis signal is incorporated to obtain the mass flowrate of valuable metal to the process, the resultant value being used to calculate the required reagent flowrate in order to maintain constant reagent addition per unit weight of metal. The new set-point is then compared with the measured value of reagent flowrate and adjustment made as before. The success of such control loops depends upon a consistent and predictable relationship between the controlled variable i.e. the reagent flow, and the measured variable (the metal flowrate to the plant), and they often fail when significant changes occur in the nature of the plant feed. Since it is really mineral content which controls reagent requirements, and this can only be inferred from the metal content, changes in mineralogy or concentration of recycled reagents can cause unmeasured disturbances, which cannot be compensated for in such pure feed-forward loops, and hence some form of feed-back trim is sometimes incorporated, often only with limited success.

### 2.1.3 Feed - Forward / Feed - Back Control

Figure 2.4 shows a simple **feed-forward/feed-back** system which has been used to control the reagent addition to maintain an optimum tailing assay.

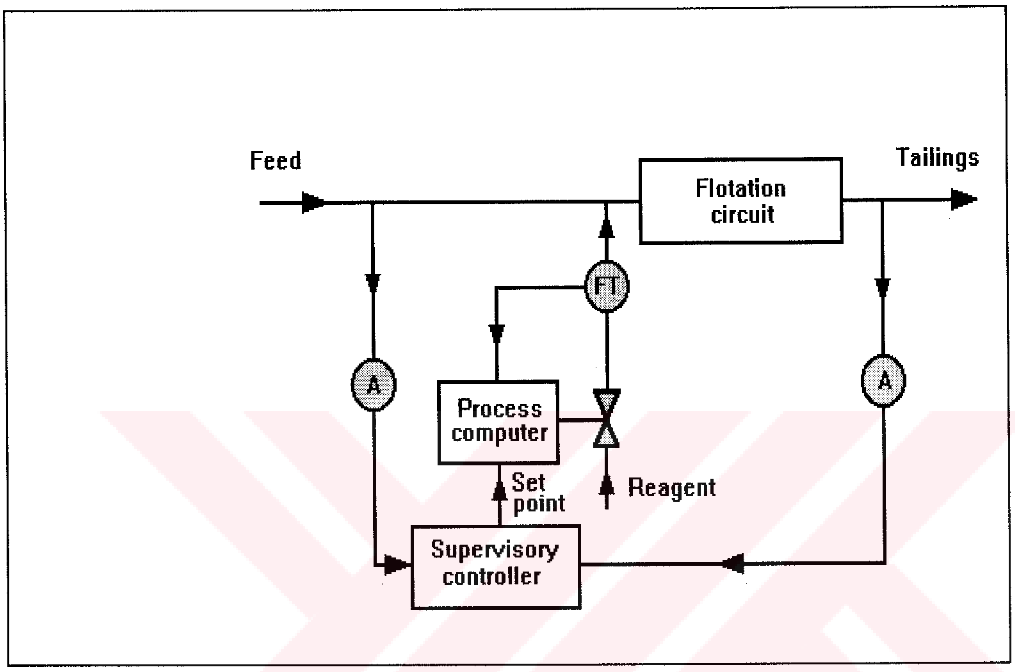



**Figure 2.4 Feed - forward /feed - back) system.**

The optimum tailings assay is calculated using the assay of the feed to the process and a feed-forward algorithm. The reagent addition is controlled at the set-point. Which is adjusted according to the difference between the calculated and actual tailings assays. The disadvantage of this system, as with the simple feed-back loop, is the time lag between changes in head grade and tailings grade response (Wills, B.A., 1985).

The ultimate aim of supervisory control is not only to stabilise the process, but also to optimise the process performance and hence increase the economic efficiency. This higher level of control has been attempted in a few concentrators with somewhat limited success, as optimisation can only be achieved when reliable supervisory stabilisation of the plant has been fully effective. The advantage of the "evolutionary optimisation method" is that no mathematical model of the process is required. The efficiency of the process is determined by a suitable economic criterion which in the case of concentrator can be the economic recovery attained. The controlled variables are altered according to a predetermined strategy, and the effect on the process efficiency is computed on-line. If the efficiency is increased as a result of the change, then the next change is made in the same direction otherwise the direction is reversed; eventually the efficiency should converge to the optimum(Wills, B.A., 1985).

.The logic flow diagram for the control of copper flotation circuit is shown in figure 2.5 The system incorporates an alarm to alert the process operator when the computer's efforts fail to halt a falling efficiency. Action can then be initiated to adjust process variables not influenced by the control strategy (Arikan, A.T., 1993-1995).

In addition to the process and supervisory control computers, a general purpose computer may be incorporated in the hierarchy. This computer is generally housed in the control room and coordinates the activities of the supervisory computers, as well as performing such tasks as logging and plant data, preparation and printing of shift, daily and monthly reports, and supervision of shut-down and start-up. The computer can allow the operator to input information such as changes in metal price, smelter terms, reagent cost, etc., which can aid optimisation of the supervisory controller (Arikan, A.T., 1993-1995).

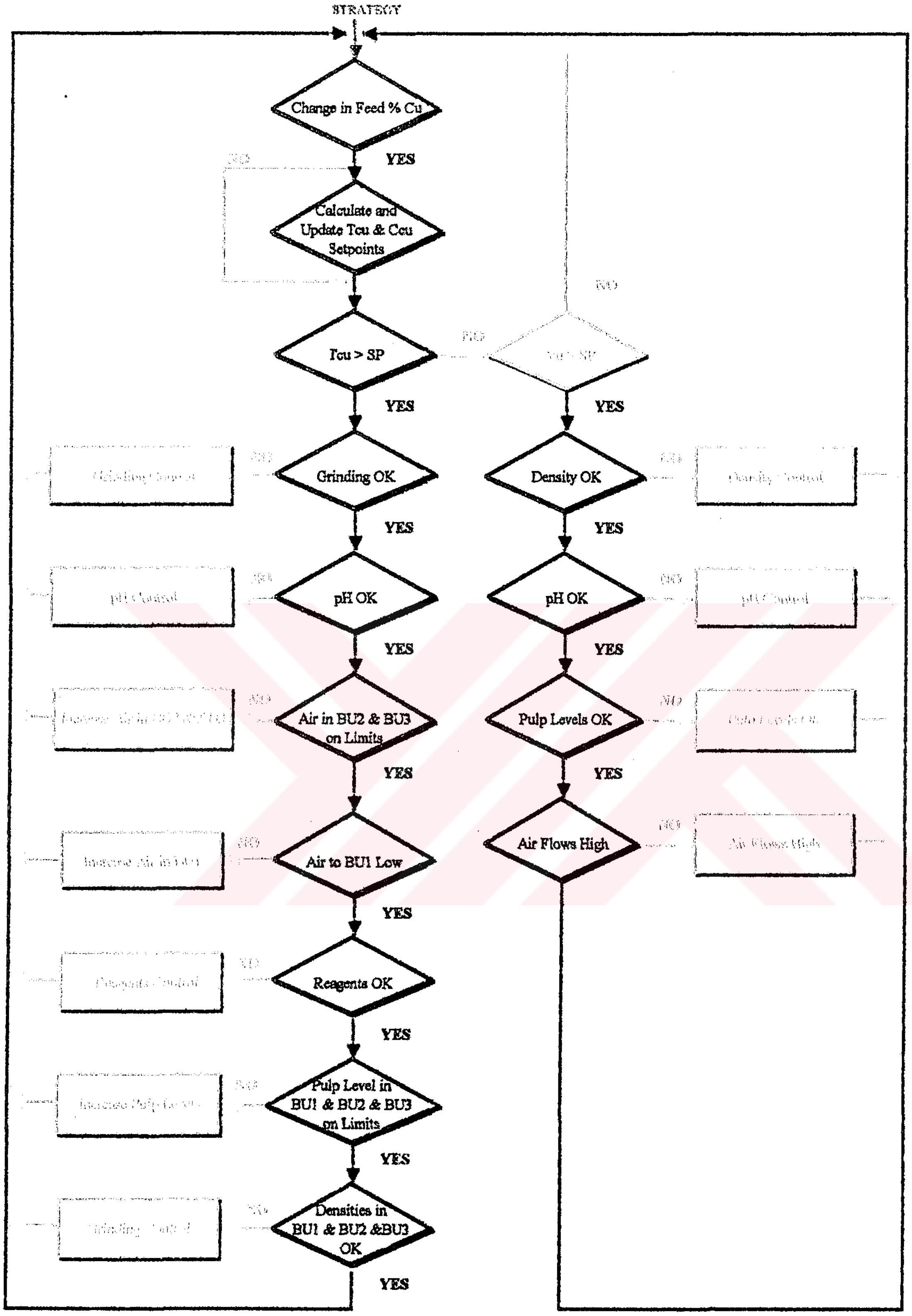



Figure 2.5 Logical control of final concentrate grade and tailing grade.

CHAPTER THREE

# PROCESS CONTROL

**3. General**

Instruments and their properties are very important part of effective automatic control. Variables that are controlled by instruments at Kure concentrator are presented in this chapter.

**3.1. Particle Size Control**

On-line measurement of the particle size distribution of a mineral slurry is commonly considered a difficult job. However, in both grinding and flotation circuits it is very useful to know the particle size. If a dynamic process is to be controlled efficiently, a real-time particle size information enables continuous monitoring of the performance of key process sections. Operators can base their decision on facts, not guess work.

Efficient grinding with a continuous particle size indication enable to eliminate over- and undergrinding in a cost effective way, thus resulting in increase revenue.

On flotation processes the measurement of particle size of rougher concentrate will help to reach and control the optimum flotation conditions.

A particle size indicator ( **PSI** )should be efficient, need low cost, minimal maintenance solution for better grinding and flotation control.

PSI should mimics a simple technique which actually has been in manual use at concentrators in the day when there was no idea of automatic process control. The flotation operators felt the particle size by taking a handful of slurry and rubbing it with their hands. In PSI the same method is used, just with mechanical fingers and precision distance measurement to give a quantitative indication of the particle size distribution (Outokumpu Oy., 1992).

Particle size indicators are designed as the basic instrument for on-stream measurement of the particle size distribution in a slurry. PSI will measure particle size from one sample line and it calculate two fraction, selected by the user (Outokumpu Oy., 1992).

Particle size indicators are easy to install and easy to use and not sensitive of air bubbles or slurry density. Calculated fraction percentage information can be transferred to the other system by 4-20 mA current loop (Outokumpu Oy., 1992).

Typical measurement points are the cyclone overflow product of a grinding circuit and concentrate stream of flotation circuits (see Figure 3.1).

### 3.1.1. Description of Operation

Accurate and continuos particle size measurement are very important for a successful grinding and flotation process. Efficient grinding with particle size analyzer helps diminish high and low size grinding. This results in optimal grinding conditions, higher grinding efficiency and decrease in process cost.

PSI 200 has four main parts as shown in figure 3.2.

- Flow Stabilization Tank Assembly (PFSA)
- Calibration Sampler
- Particle Size Measuring Probe
- Electronic Control Cabin Set

PSI is designed for continuous measurement of grinded materials grain size and mostly used for the measurement of cyclone overflow product this apparatus measures the particle size of known sample and calculates two size fraction (- 38μ and - 74μ) that is chosen by engineers. It imitates manual testing of particle size which takes place in plant where there is no automatic process control. Flotation operators are used to try to estimate particle size by their fingers. In PSI 200 the same method is carried out mechanically. From the process line a representative sample is taken into analyzor`s local sample system. A small amount of pulp sample (10 lt/min) is put into measurement probe. Measurement unit periodically catches particles between two flat surface and the distance of this surface is also measured. Statistics of following measurements (about 120 measurements) are gathered and desired size fractions are calculated with respect to average and standard deviations of these measurements. PSI gives correct results with 1 - 2 % error.

The calibration sample collection procedure is activated from a push button. The sample that is measured by PSI is automatically diverted by the calibration sampler for laboratory analysis. Data from up to 30 calibration measurements are automatically stored in memory. Calibration and screen analysis data are entered through the keyboard. Calibration models are determined using Lotus 1-2-3 or an equivalent statistical package.

In normal operation a representative sample is taken from the process stream to the local sampling system of the analyser. A small slurry stream (about 10 Lt/min) is led through the measurement unit (Arikan, A.T., 1993-1995) (see Figure 3.2).

The sampling system and the status information of the PSI is continuously controlled by the program. In case of fault an alarm is generated and the reason will be shown on computer screen.

The sampling system is flushed and zero check measurement is done automatically on regular time intervals to maintain the stability of the measurement (see Figure 3.3).

For the size measurement, representative primary sampling is crucial. To obtain a good sample from process stream a fairly large (about 50 - 150 l/min) primary sample is taken to the analyser and returned back to the process (Arikan, A.T., 1993-1995).

The electronics are housed in a compact enclosure equipped with an LED display.The particle size measurement, which is shown as percent passing the selected mesh, is continuosly updated and displayed on the LED and as a 4-20 mA signal.The LED is also used to present calibration measurement and parameter information and for displaying alarm messages if the measurement is interupted or other different problem.Calibration sampler is

Integrated to the PSI system for getting a representative sample for laboratory. This sampler can be programmed for daily sample operation as well.

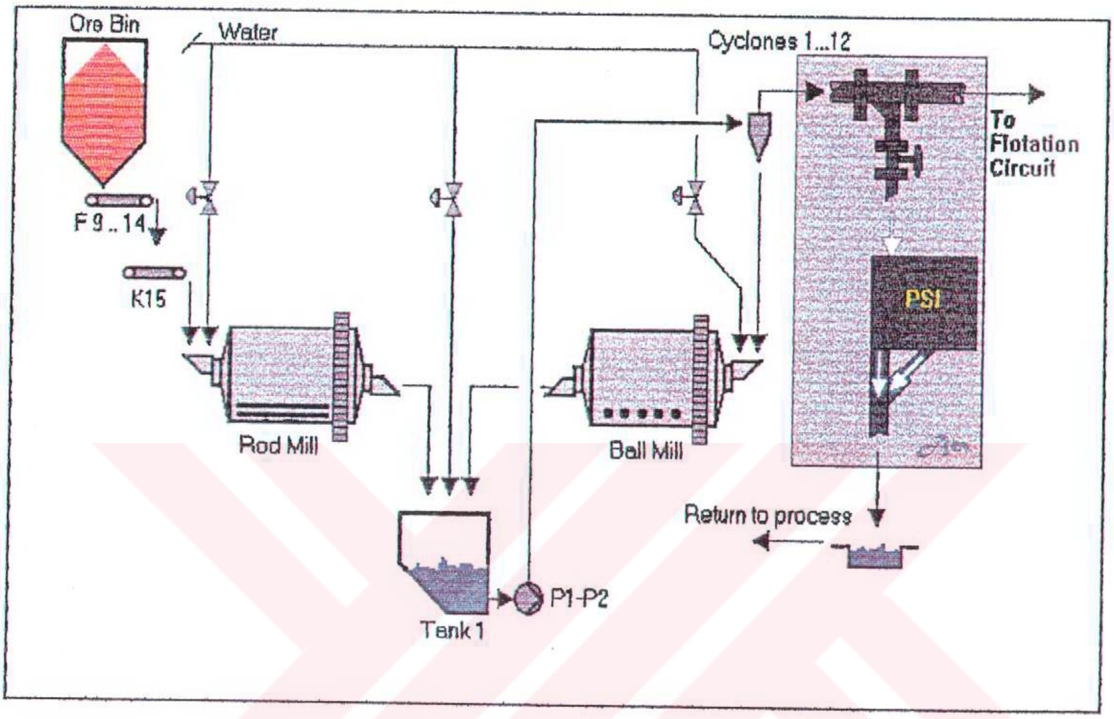



**Figure 3.1 Installation of particle sizer.**

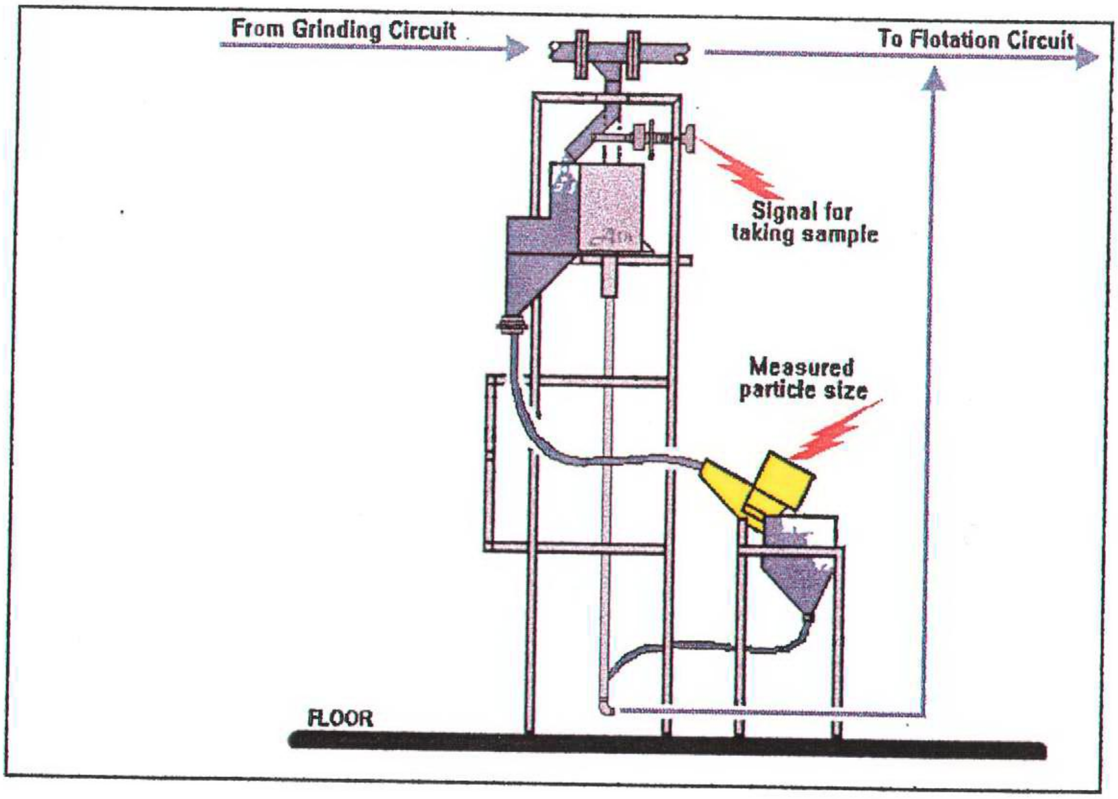



**Figure 3.2 Measuring of particle size.**

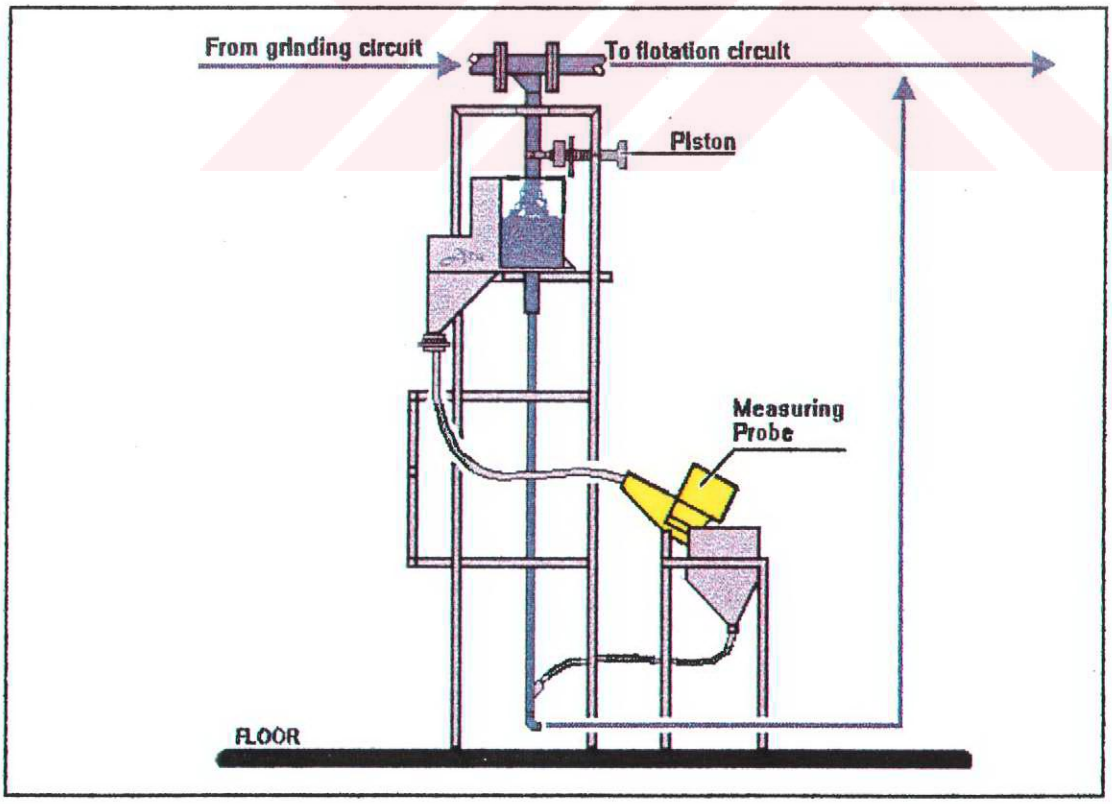



**Figure 3.3 Particle sizer (no measurement).**

### 3.1.2 Calibration

For getting the PSI to show the correct percentage value it must be calibrated against sieve analysis or some other reliable method (see Figure 3.4).

Around 20 calibration sample covering the normal range of process condition are usually sufficient.

The calibration process produces the parameter values used when calculating real-time particle size percentage values of the measured slurry.

When enough sample have been collected from PSI, the average values (AVE) and standard deviations (SD) measured by the PSI are fitted to the laboratory analysis by simple multilinear regression.

To correct the nonlinearity of the distribution, the inverses of the average values (1/AVE) can be used as variables as well. From the regression analysis, coefficients are obtained for calibration model expression.

% fraction = A0 + A1*AVE + A2*SD + A3*1/AVE

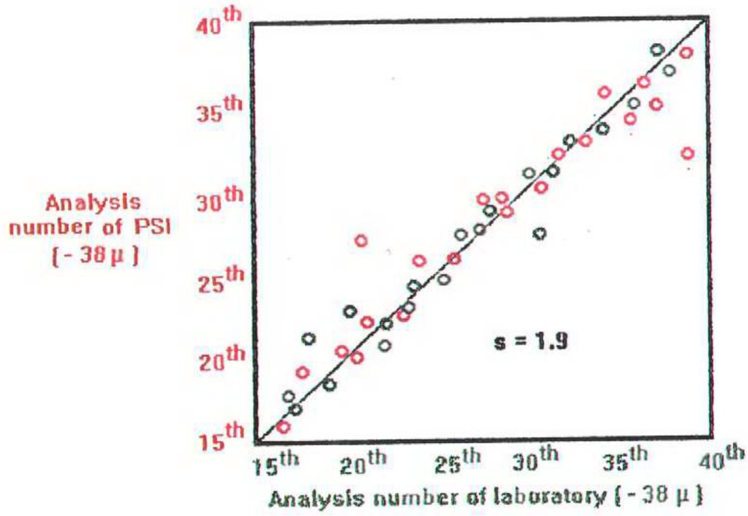



**Figure 3.4 Calibration Curve.**

### 3.2 On - Stream Analyzing

On-stream analyser systems designed for the measuring element concentrations in mineral slurries in mining and other process industries. On-stream analyser systems include all necessary equipment and controlling devices from primary sampling to producing the analysis result.

System is based on the Wavelength Dispersive X-ray Fluorescence method. The X-ray Wavelength Dispersive spectrometers have been developed to give the highest precision and sensitivity in elemental analysis.

A separate operator station (i.e. computer) is available to be directly interfaced with the on-stream analyser for process analysis and analyser status data logging, further calculations, analyser status display, process diagram, trend and history displays and reporting. The operator station can be expanded into a plant wide control system. The operator station is based on PC with the following features (Arikan, A.T., 1993-1995) :

- 80486 CPU
- 24 MB RAM, 240 HB H.D.D.

- 15" VGA display, keyboard and printer
- UNIX operating system
- Operator station application software

The operator station software should be have :

- Alarm and event handling
- Data logging
- Report generation
- Process diagram, trend and history display (see Figure 3.5 and 3.6)

Analysis results and analyser system status data is stored into the PLC (provides MODBUS RTU (Binary) protocol for communication) memory. MODBUS RTU is one of the most widely used PLC data transfer protocol in process automation system (Rantala, A., 1993).

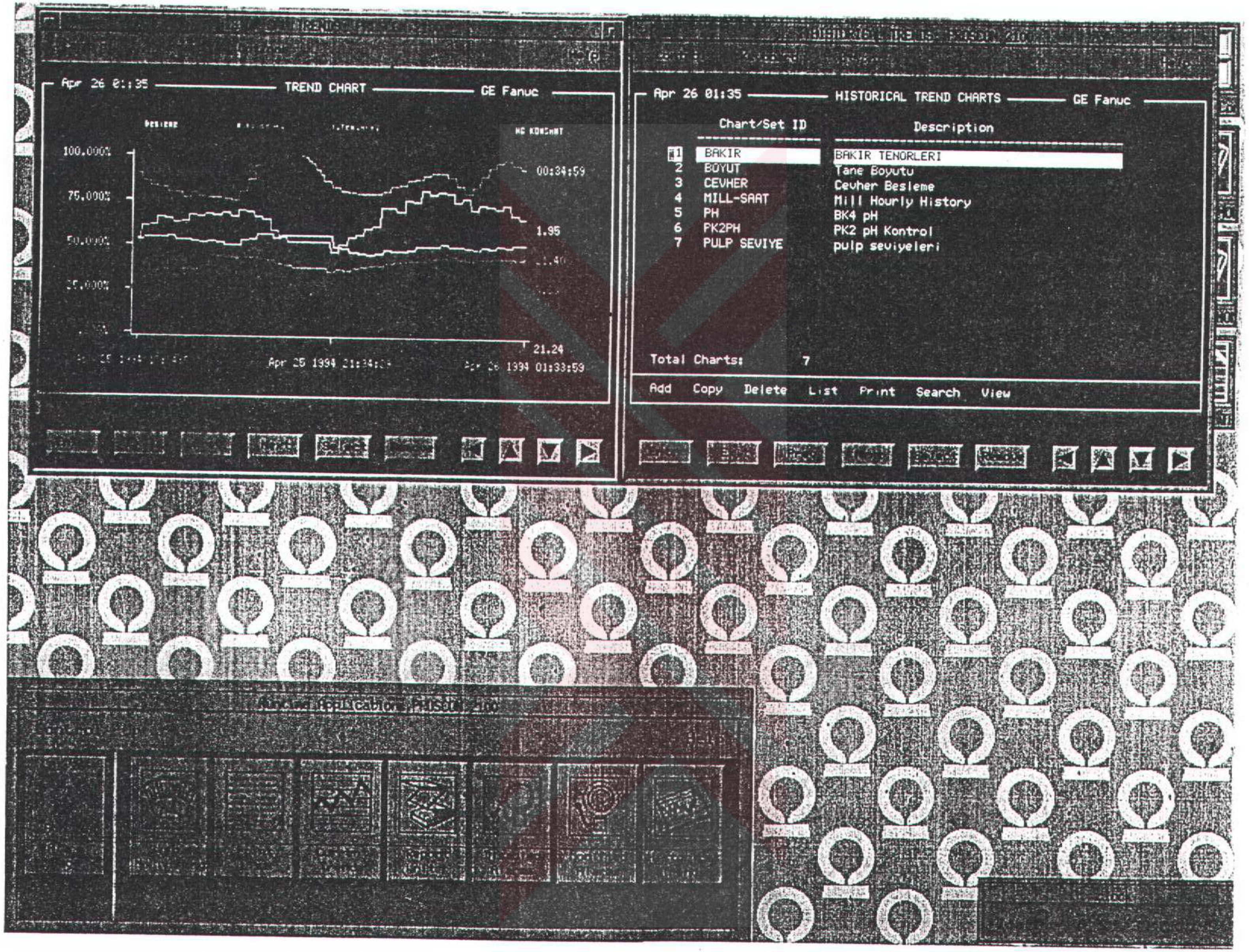



Figure 3.5 Historical trend chard of copper grades.

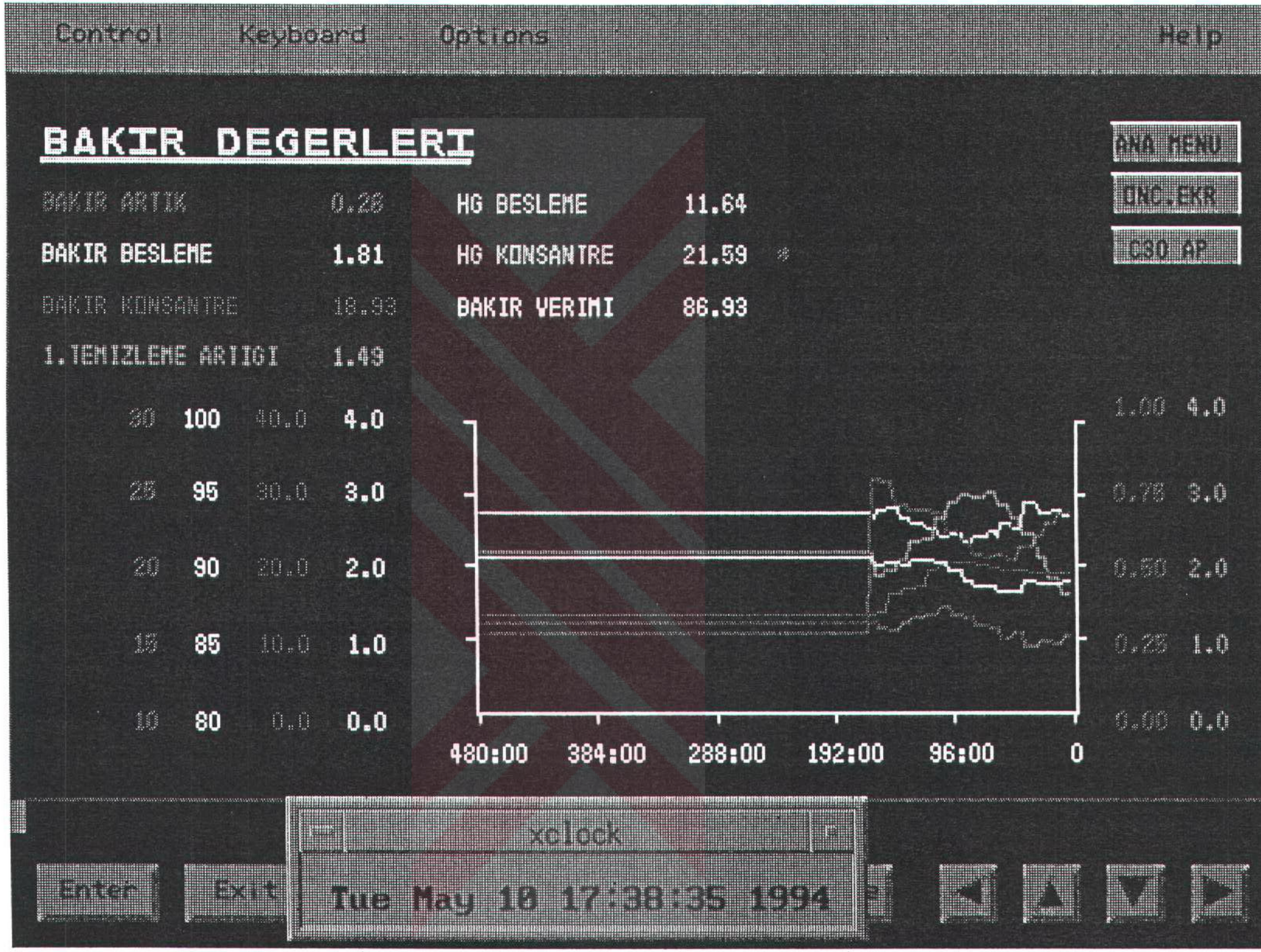



Figure 3.6 Historical trend chard of copper assays.

### 3.2.1 Description of Operation

On-line stream analysers include all functions needed for sample presentation control, instrument stability, process sample measurement, assay calculation, calibration and system setup (Outokumpu, Oy., 1992).

The operator station is available for analyser status and process data logging, alarm generation, further calculations, process diagram, trend, history display and reporting on different time levels (see Figure 3.11).

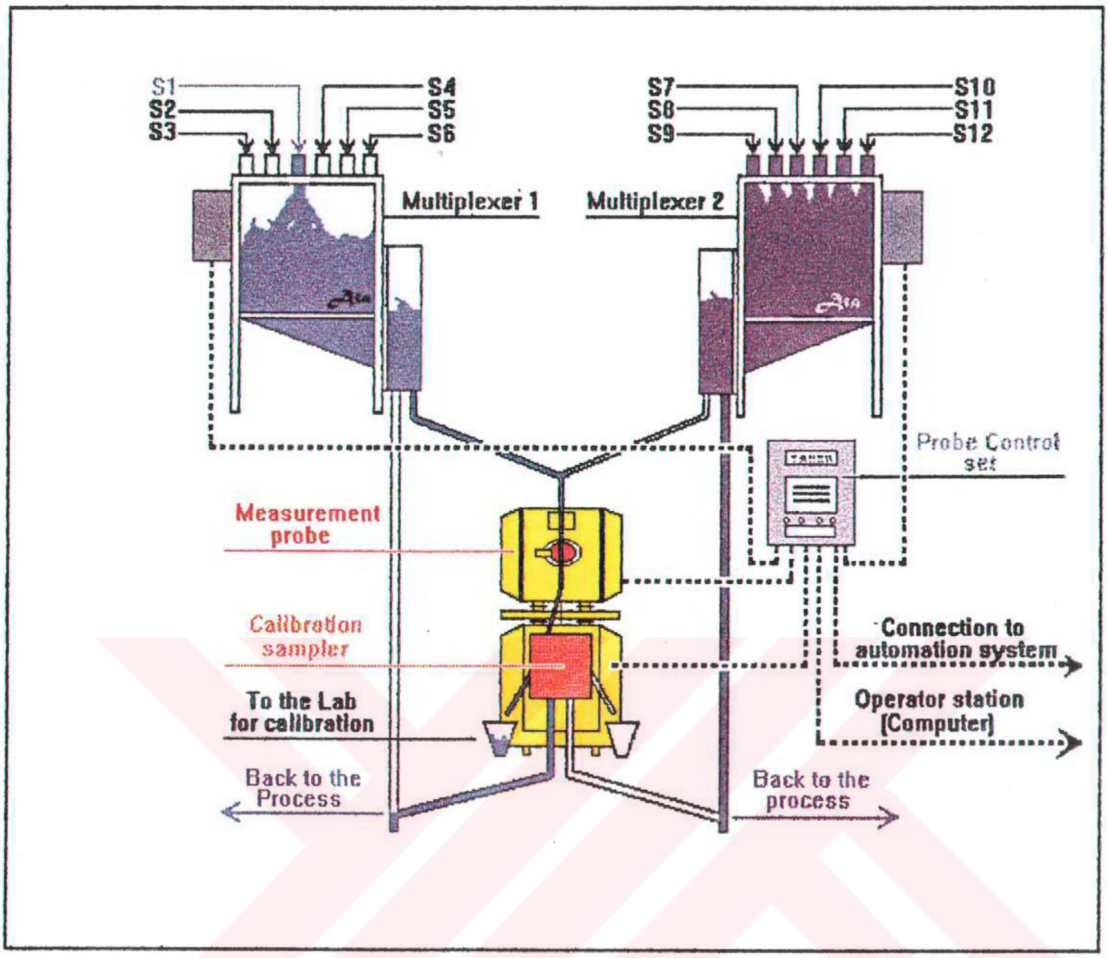
Systems are designed to produce accurate analyses from a large number of streams at short intervals. This is achieved by efficient use of the measuring probe and the multiplexers.

The sequencing of sample measurement, water flushing and sample change in the analyser probe is organized to maximize the effective time available for process sample measurement.

While one sample is being measured from one multiplexer the next sample is already directed into the other multiplexer. the sample changeover then only requires flushing of the measurement cell and related slurry pipes. the first multiplexer is drained, flushed and refilled with new sample while measuring sample from the second multiplexer (see Figures 3.7 and 3.9).

The sample changeover time is efficiently utilized for internal reference measurement. Thus the measurement probe stability can be maintained without extending the actual process sample measurement interval.

Sample presentation control monitors the performance of sampling equipment in many ways to ensure proper presentation of the process sample to the analyser. A malfunction will interrupt the measurement and alarm is shown on the display of the probe control set (PCS). Alarm status flags are also available for operator station or third party automation system (see Figures 3.8 and 3.10).




**S1** : Rough Tailing

**S2** : 2 nd. Cleaning Concentrate

**S3** : 3 rd. Cleaning Concentrate

**S4** : 1 st. Cleaning tailing

**S5** : Flotation Feeding

**S6** : Final Concentrate

**S7** : High Grade Cell (HG) Feeding

**S8** : HG Concentrate

**S9** : Spare

**S10** : Spare

**S11** : Spare

**S12** : Spare

**Figure 3.7 Schematic view of x - ray analyser.**

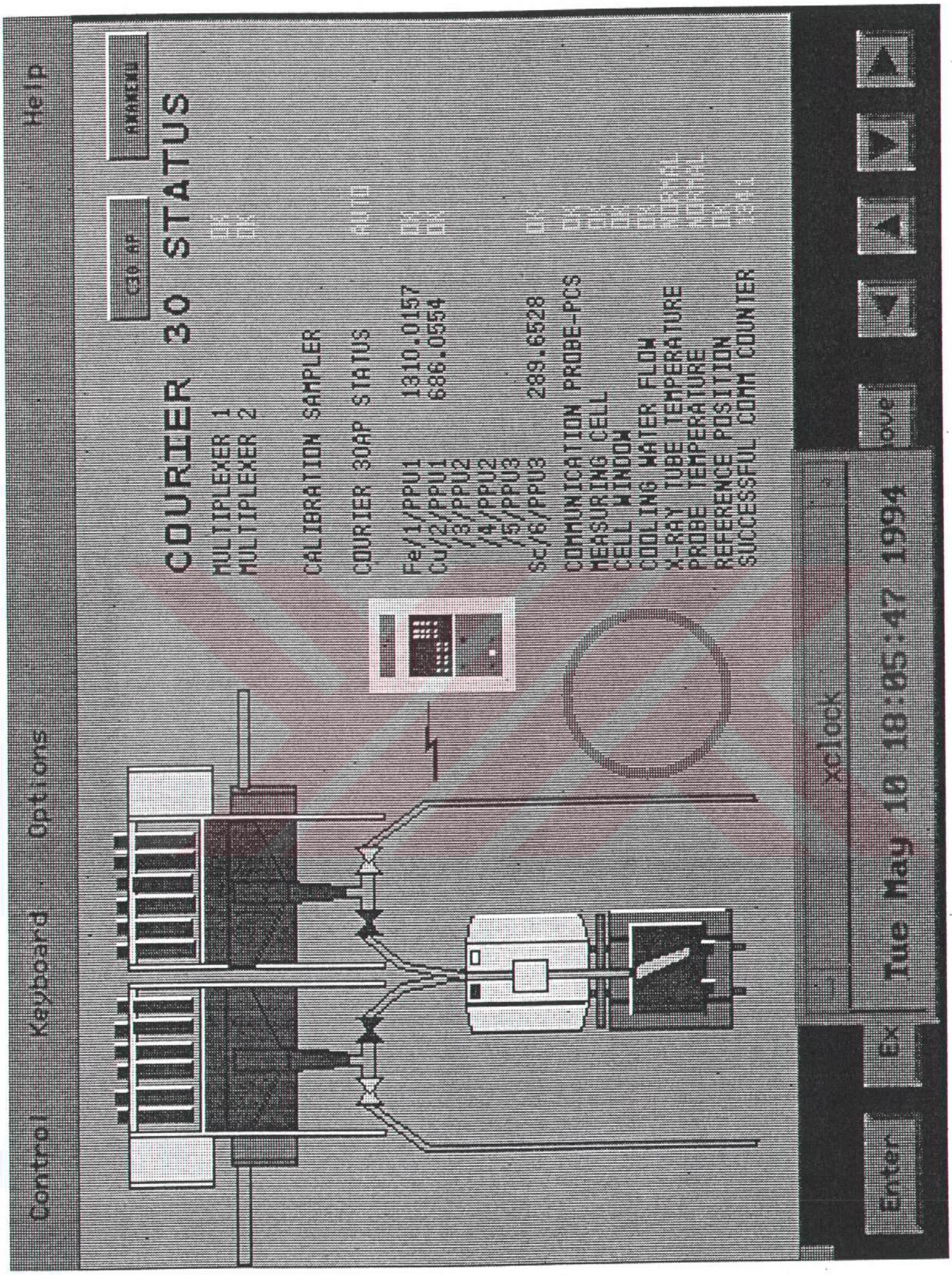



Figure 2.8 Status display of X - ray analyser.

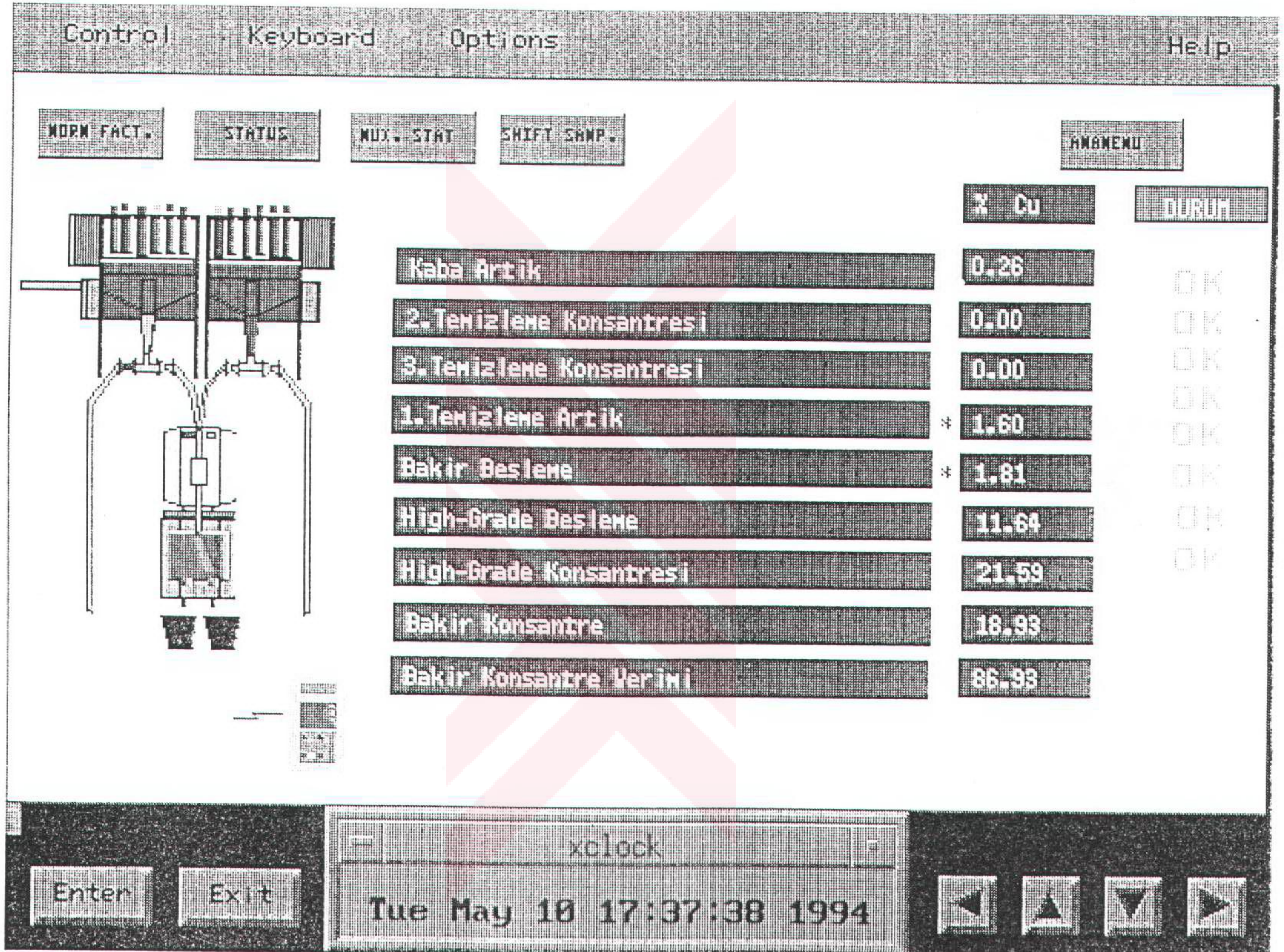



Figure 3.9 View of copper assays display.

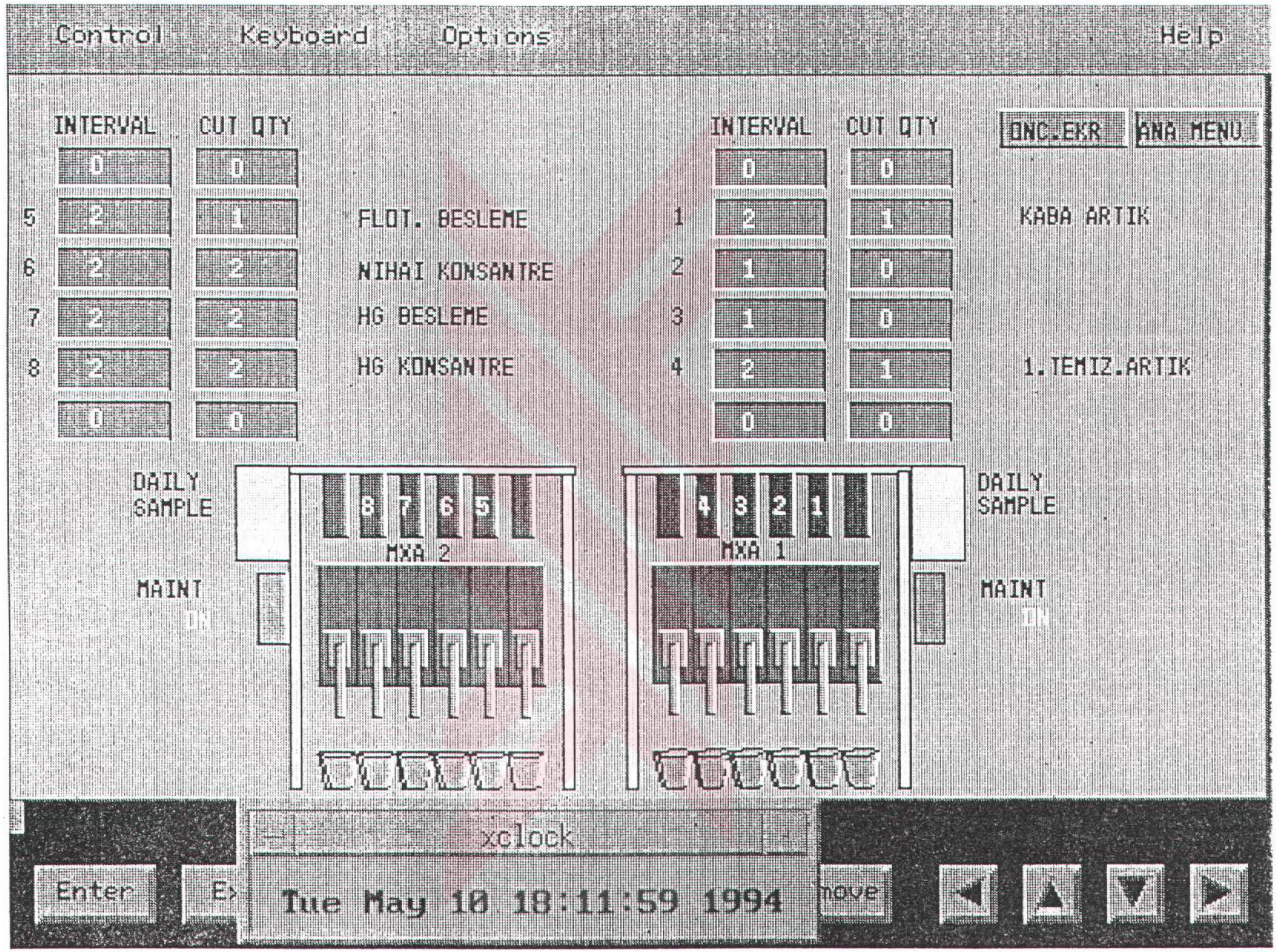



Figure 3.10 Interval times and quantities of samples for laboratory.

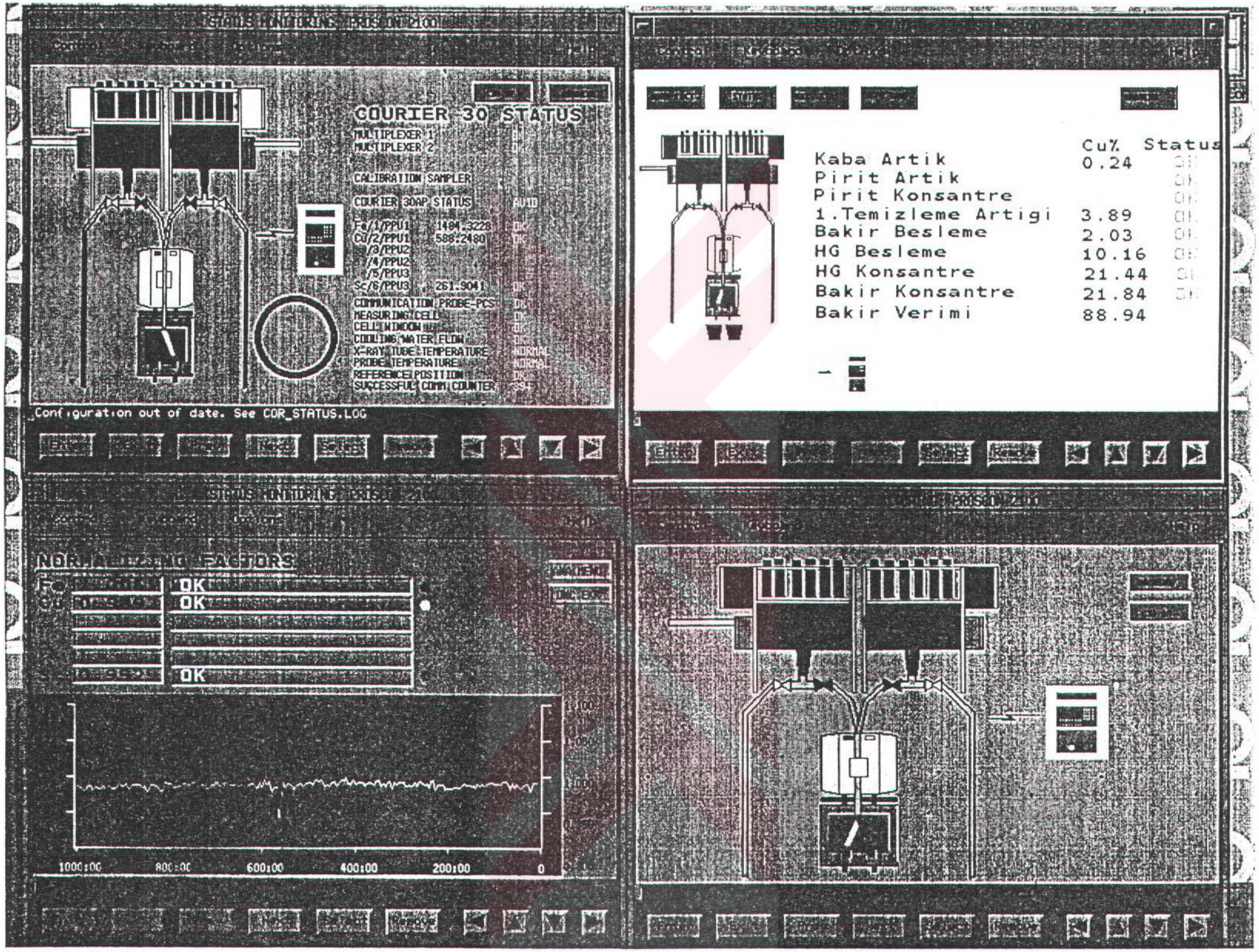



Figure 3.11 View of x - ray analyser displays.

### 3.3 pH and Reagents Control

Control of slurry pH is a very important parameter in many selective flotation circuits, the control loop often being independent of the others (see Figure 3.12), although in some cases the set-point is varied according to changes in flotation characteristics. For automatic control of lime or acid it is important that time delays in the control loop are minimised, which requires reagent addition as close as possible to the point of pH measurement. Since lime is often added to the grinding mills in order to minimise corrosion and to precipitate heavy metal ions from solution (see Figures 3.13 and 3.17), there is a considerable and unacceptable distance-velocity lag if the pH probe is situated in the rougher cells. In such cases, excess lime may be added to mills and final pH control made by acid addition at the a point close to the roughers, such as into the conditioning tank (see Figure 3.14). To prevent a considerable excess of lime being added to the mills, with a subsequent high acid requirement at the roughers, supervisory control can be used to minimise the acid consumption. In the circuit shown in Figure 3.21, the computer monitors the acid consumption, and if this amount is excessive, changes the lime addition set-point to reduce the input to the grinding mills. This can be adjusted so as to be very slow thus minimising instability due to the long distance-velocity lag in the feed-back loop (Ketola, S., 1993).

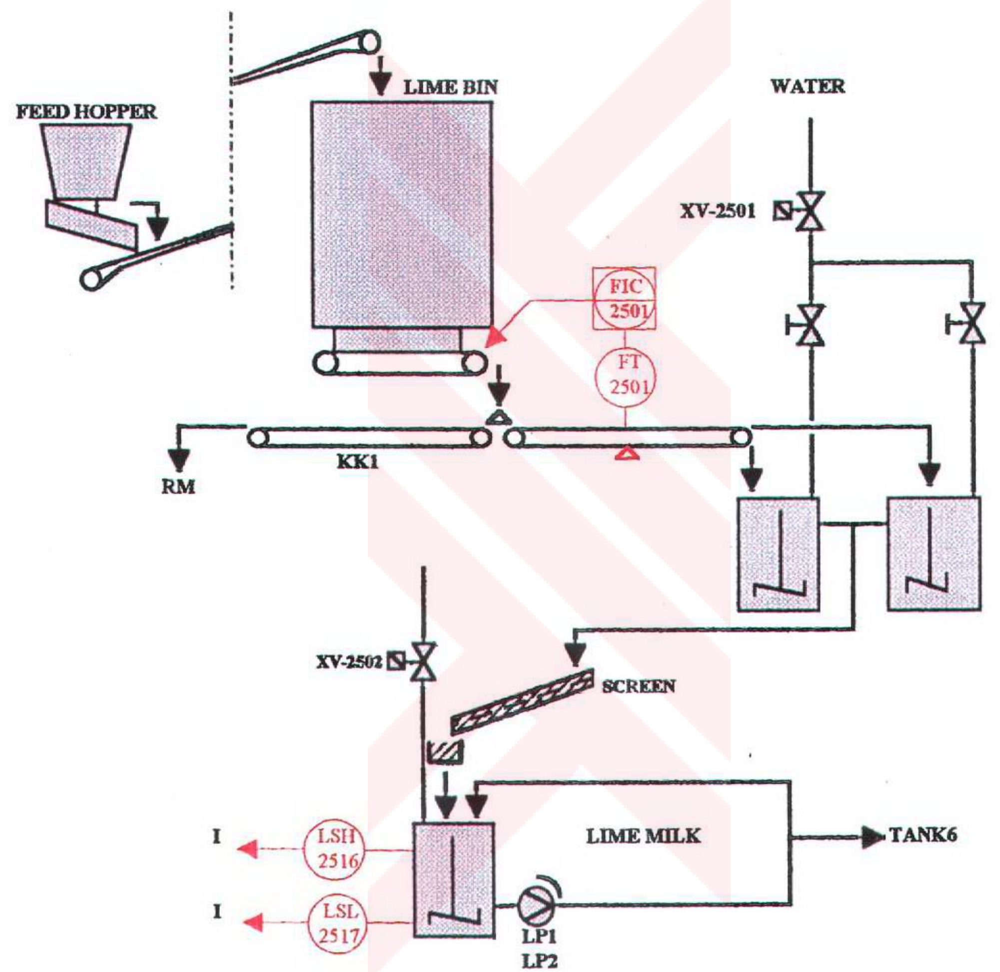



Figure 3.12 Lime preparation before flotation.

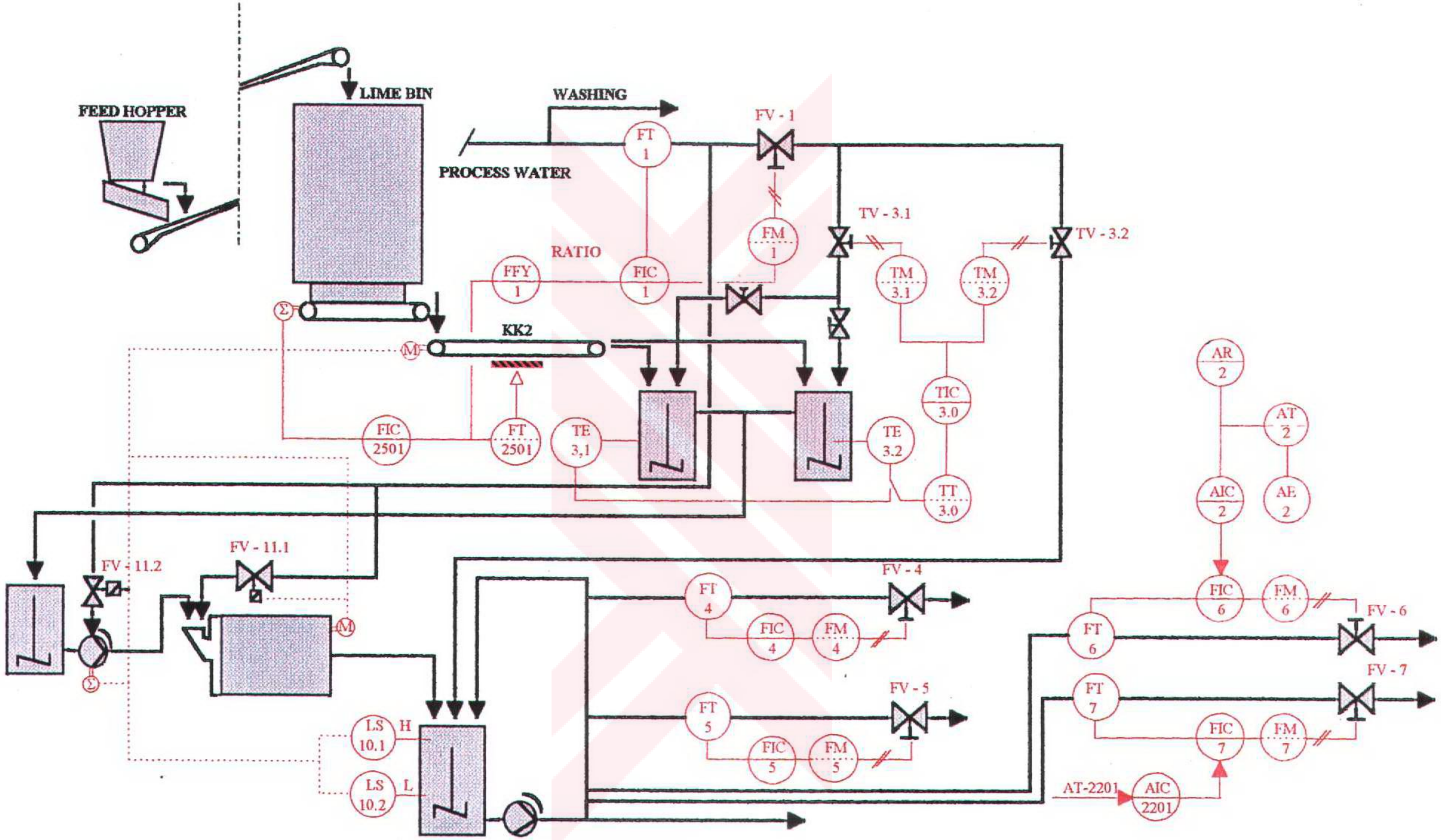



**Figure 3.13 Lime handling and dosage control.**

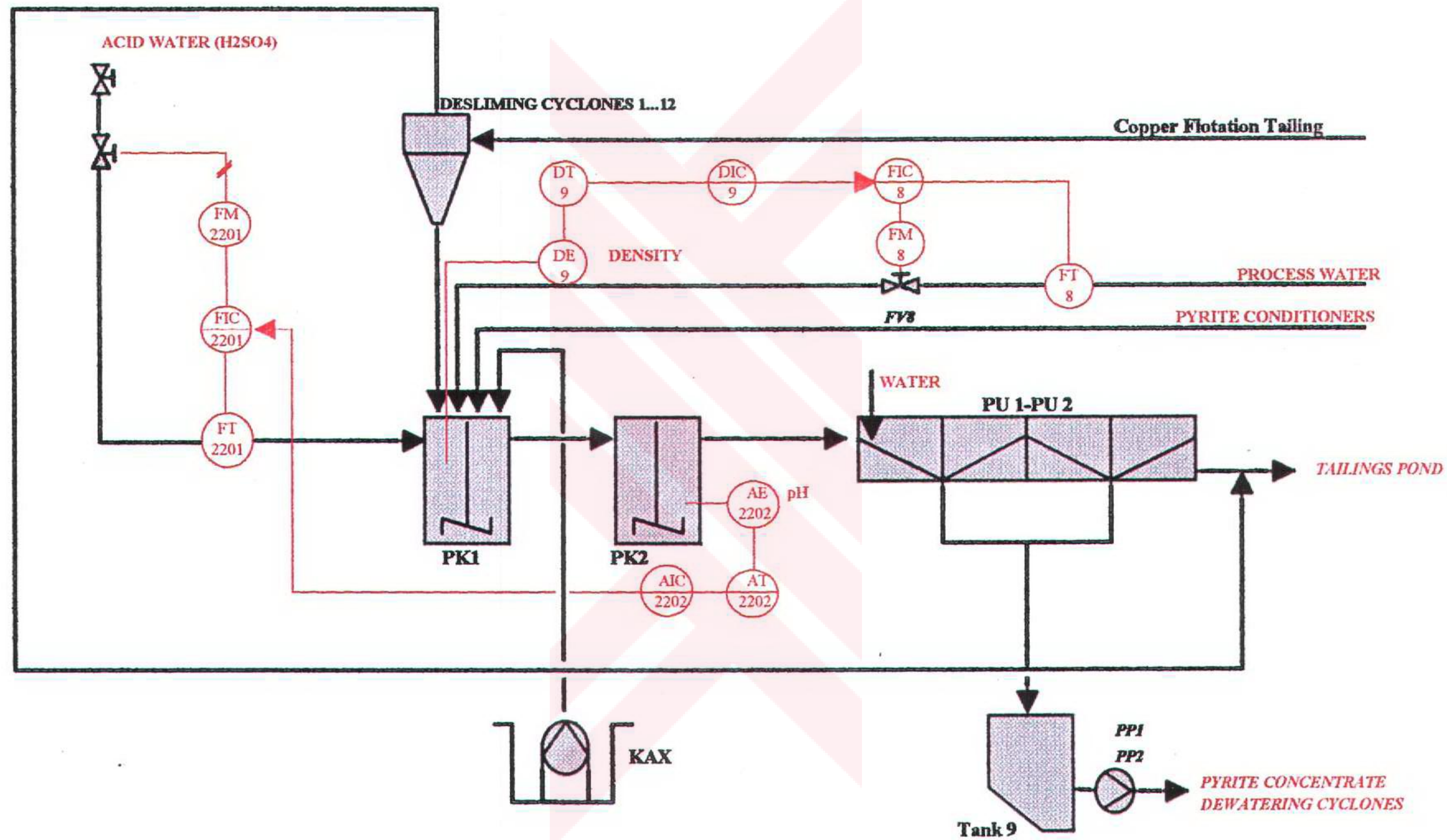



**Figure 3.14 Pyrite flotation pulp density and pH controlling.**

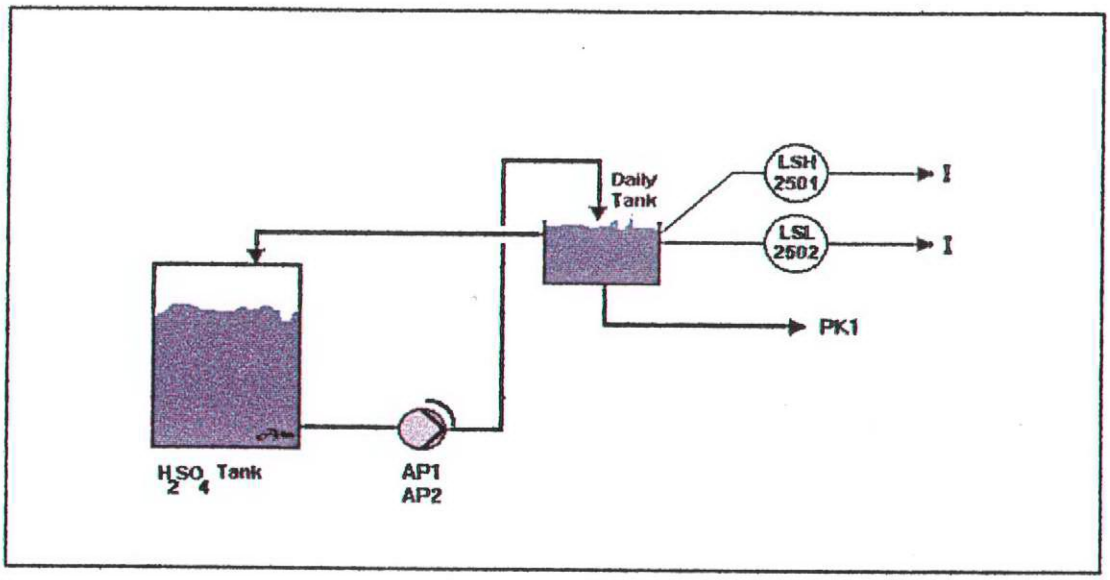



**Figure 3.15 H2SO4 preparation and level controlling.**

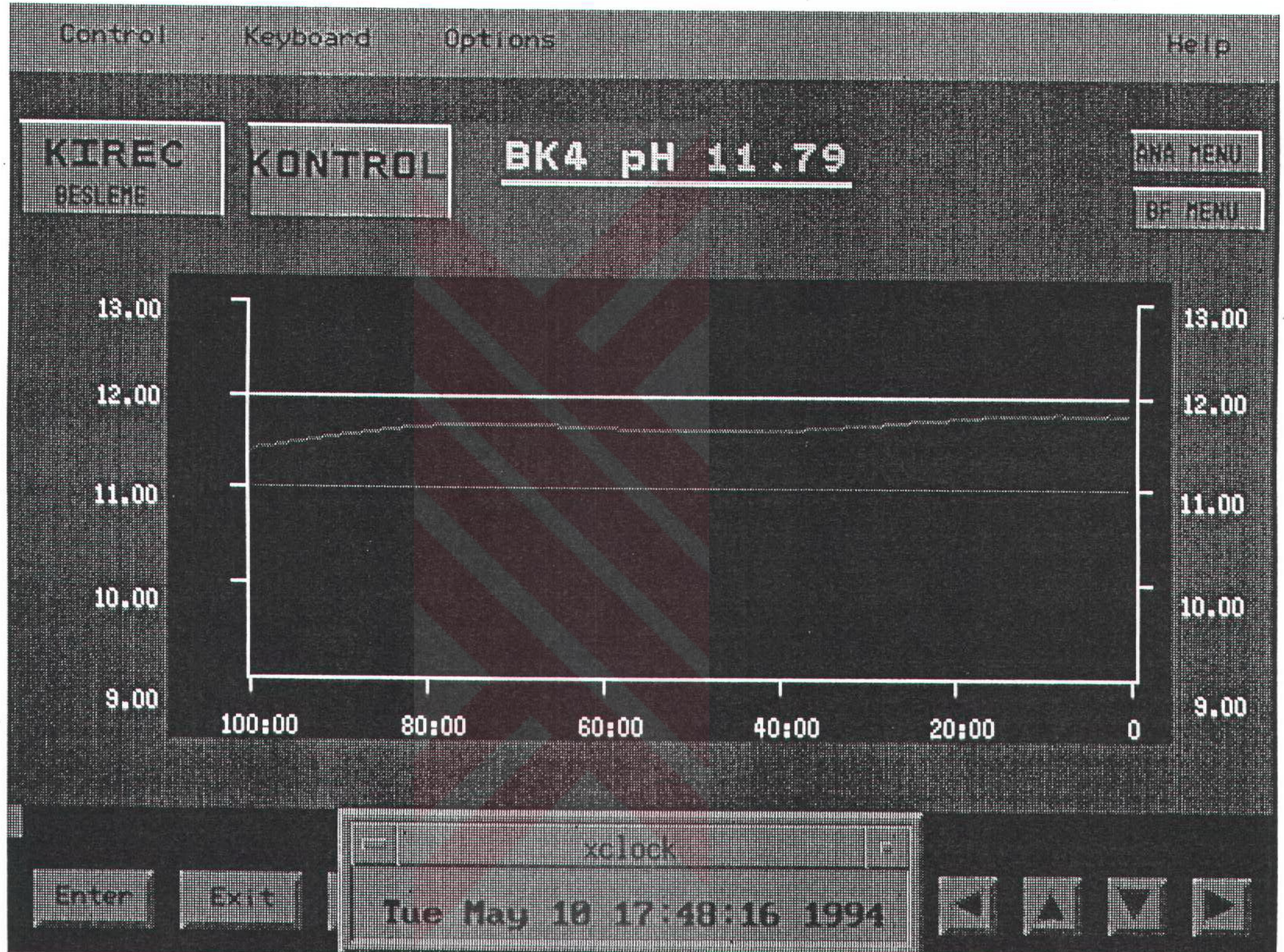



Figure 3.16 View of pH trend display.

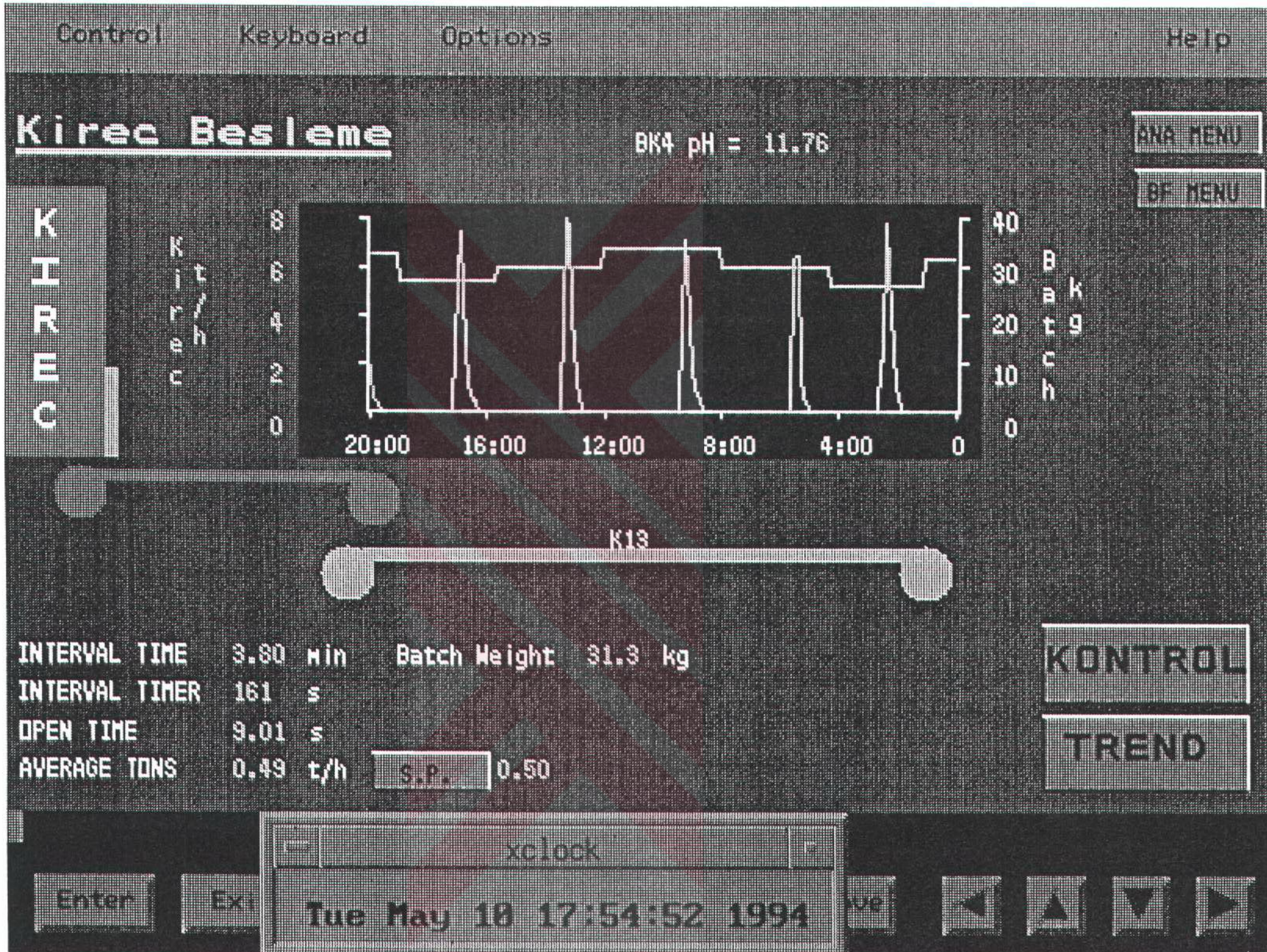



Figure 3.17 View of lime feeding trend display.

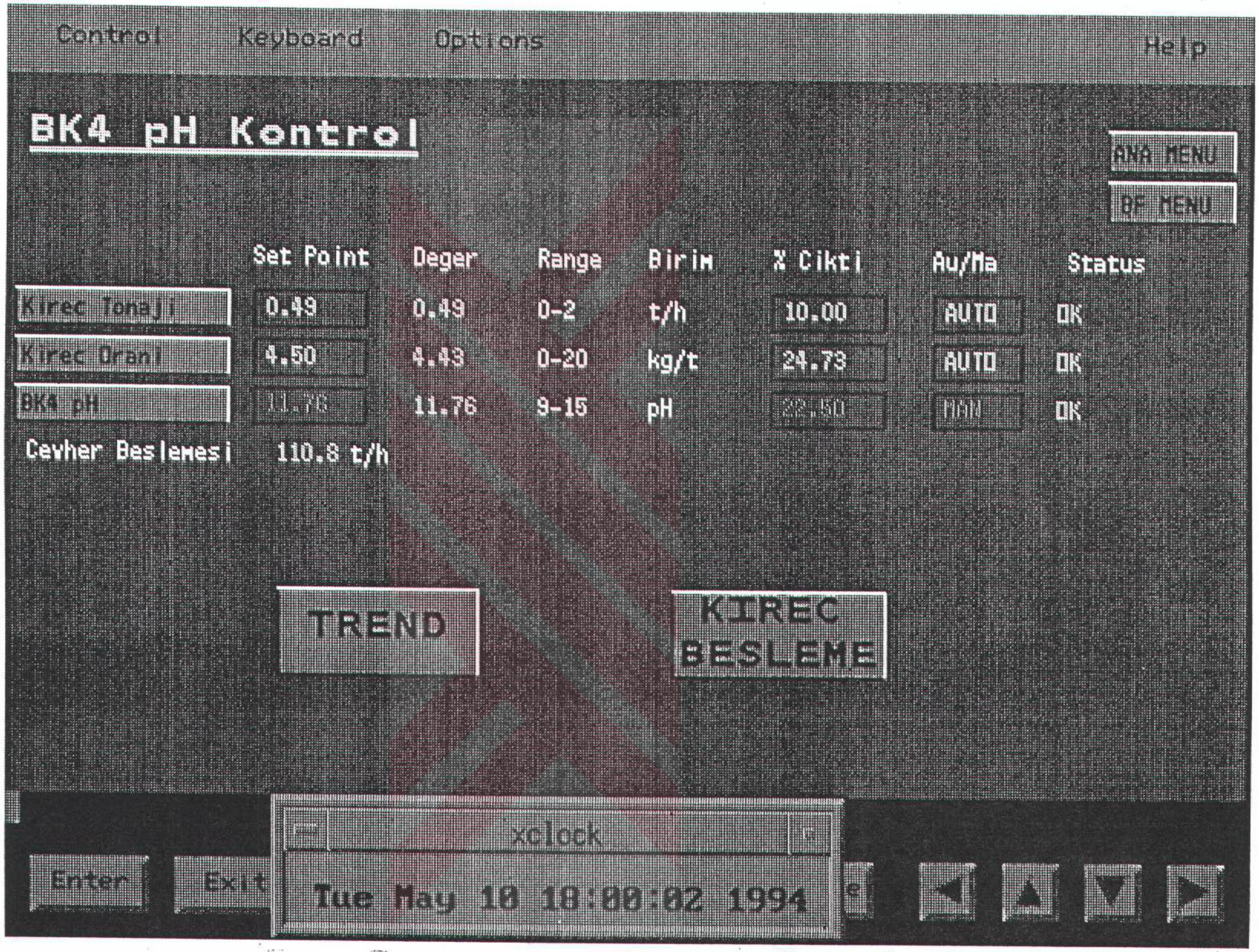



Figure 3.18 View of lime feeding quantity display.

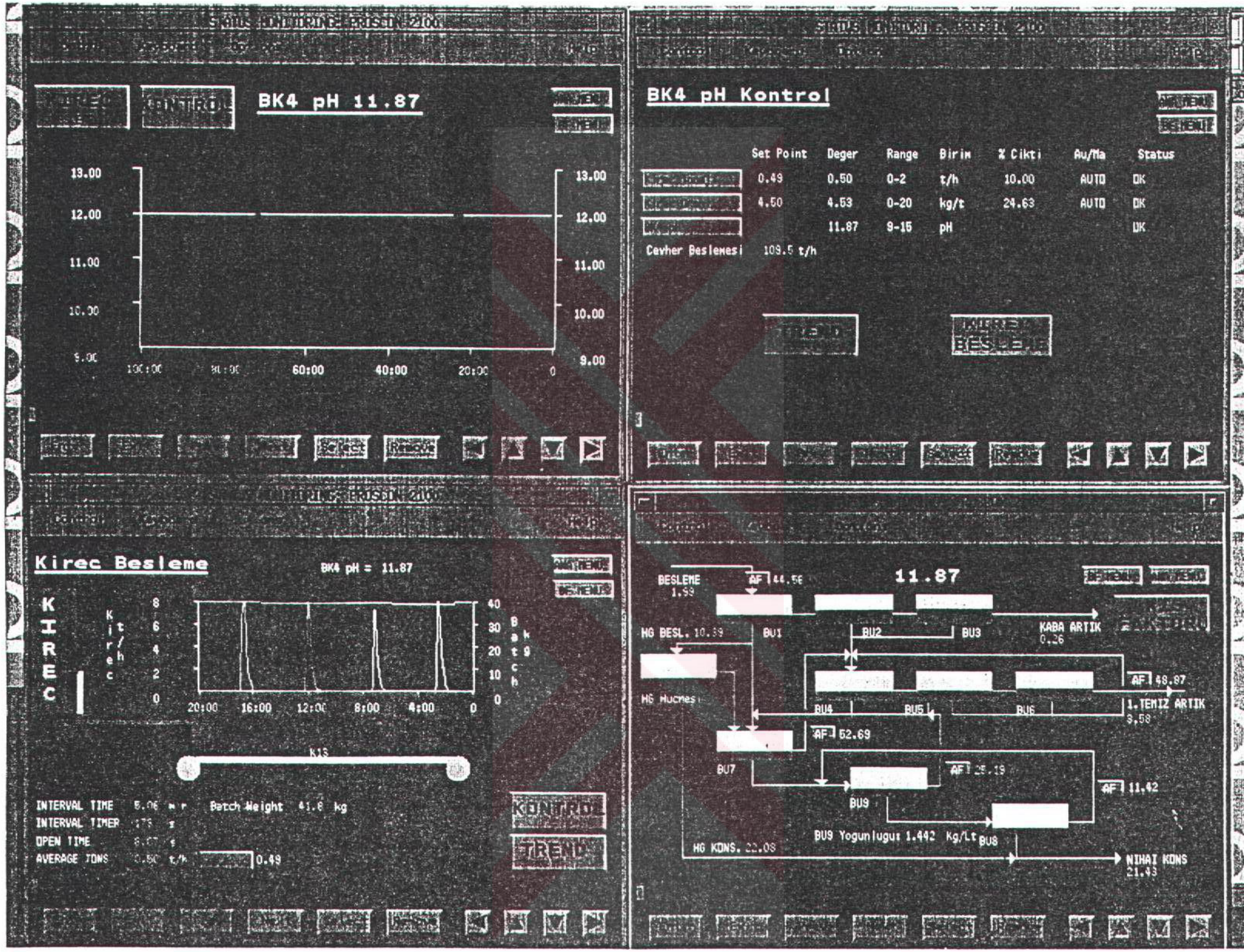



Figure 3.19 View of copper circuit's pH control displays.

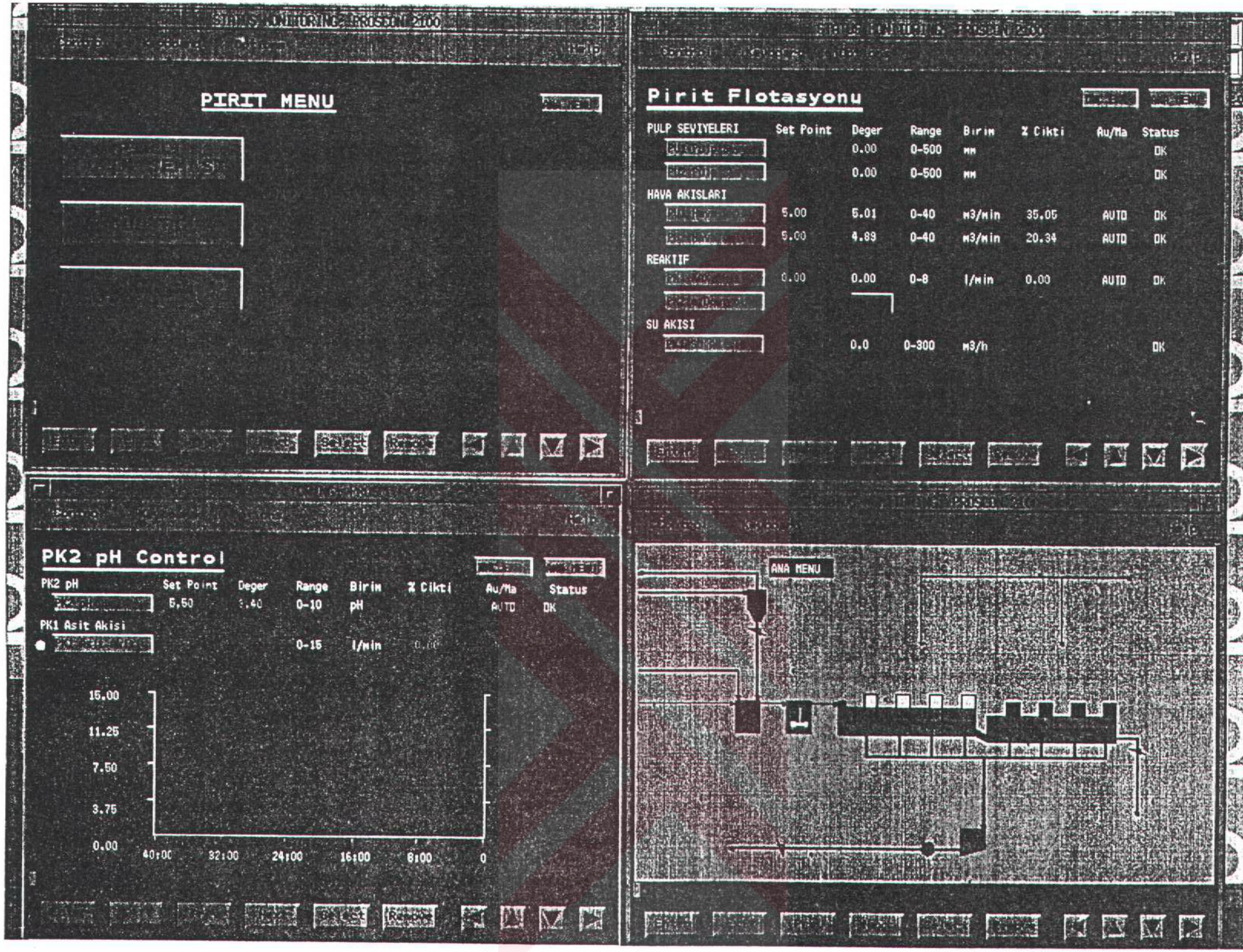



Figure 3.20 View of phyrite circuit's pH control displays.

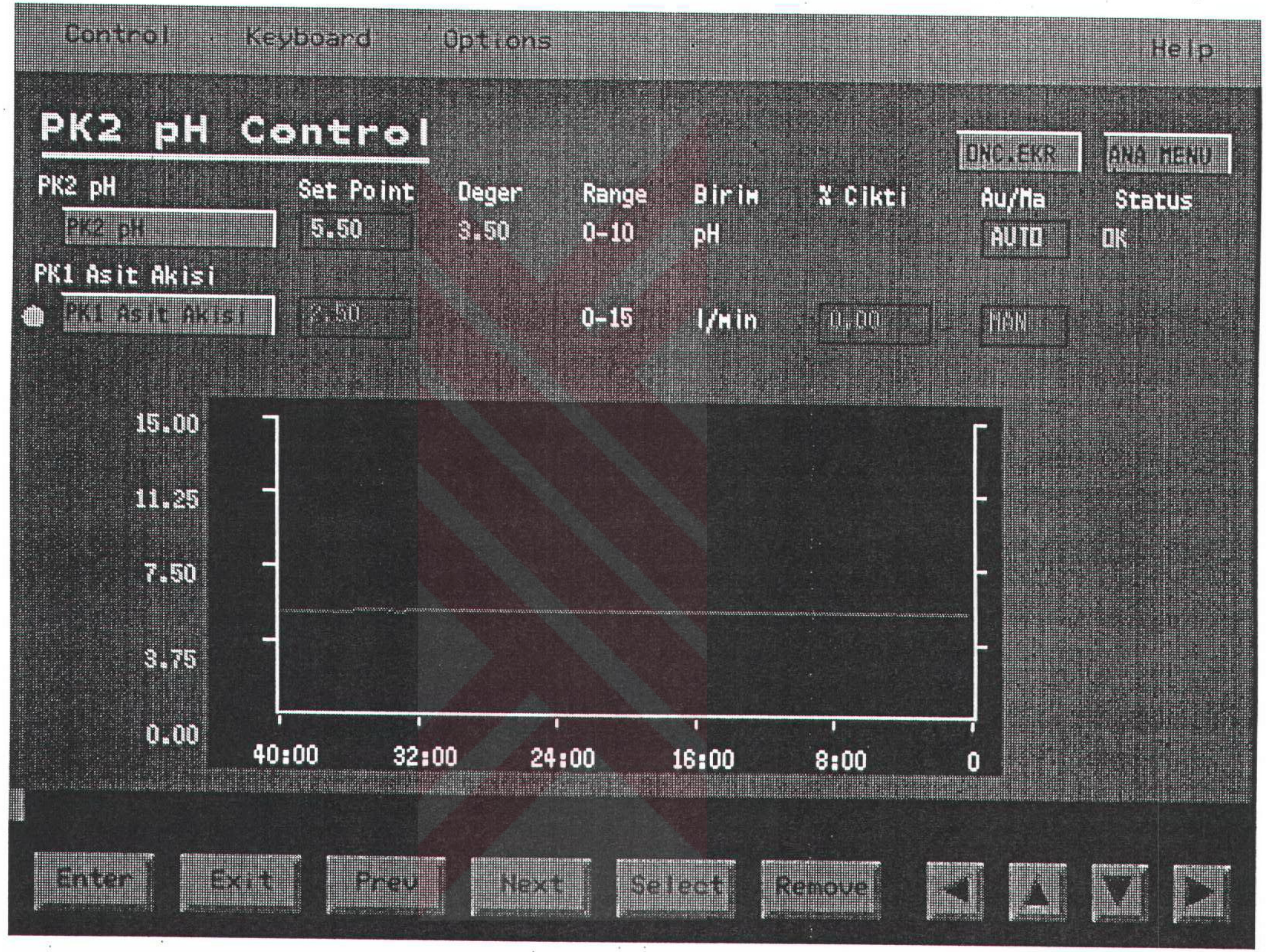



Figure 3.21 View of pH trend display.

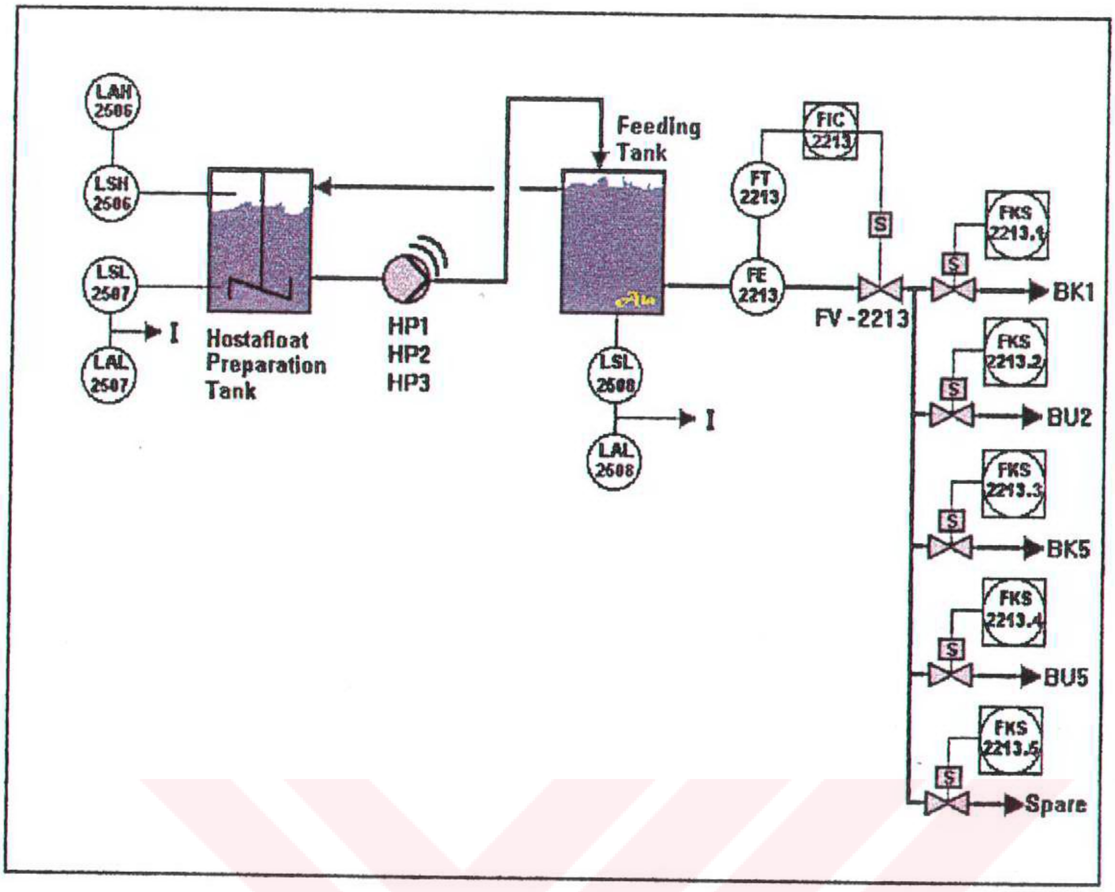



**Figure 3.22 Hostafloat preparation and feeding control diagram.**

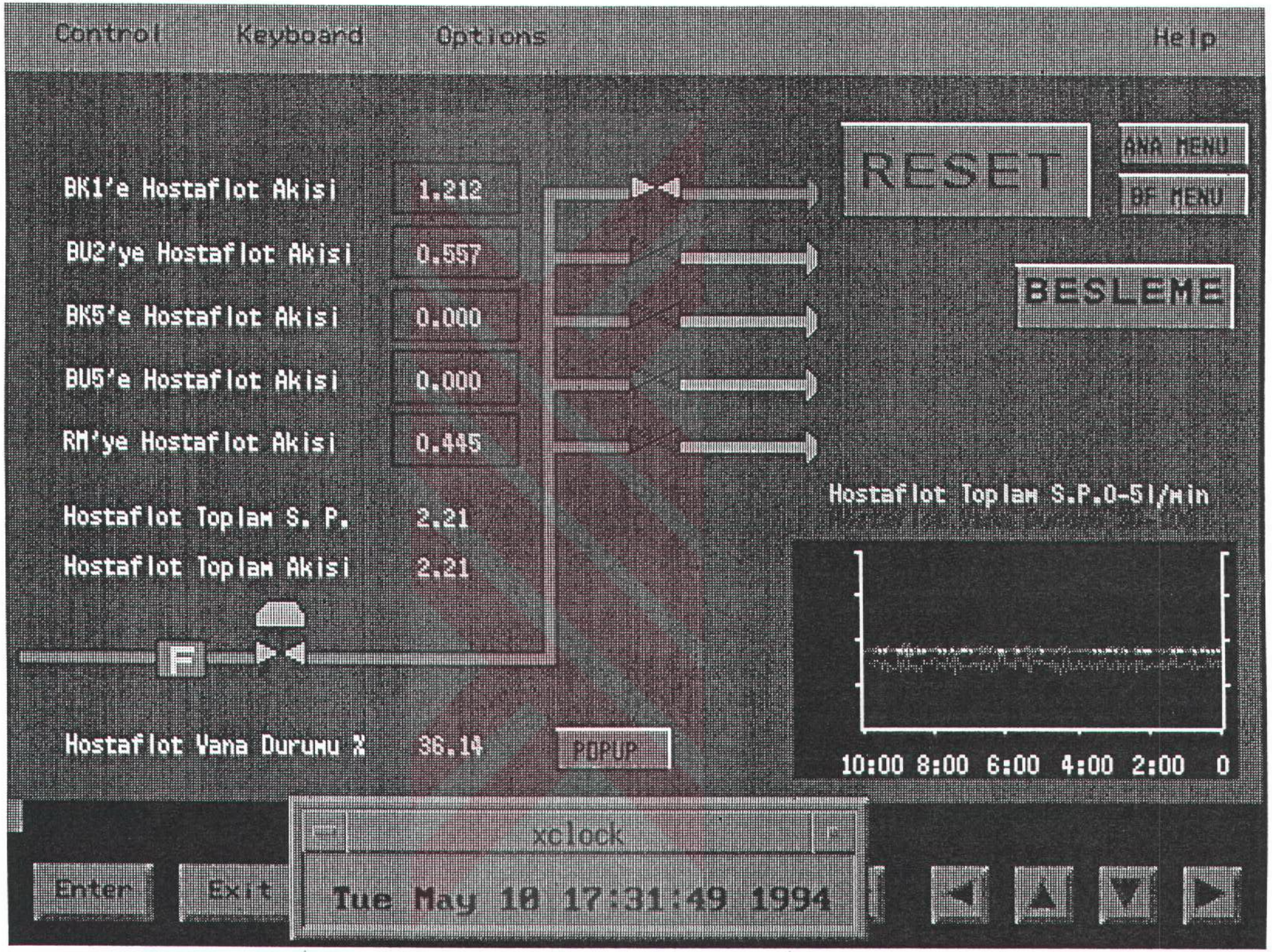



Figure 3.23 View of Hostafloat feeding display.

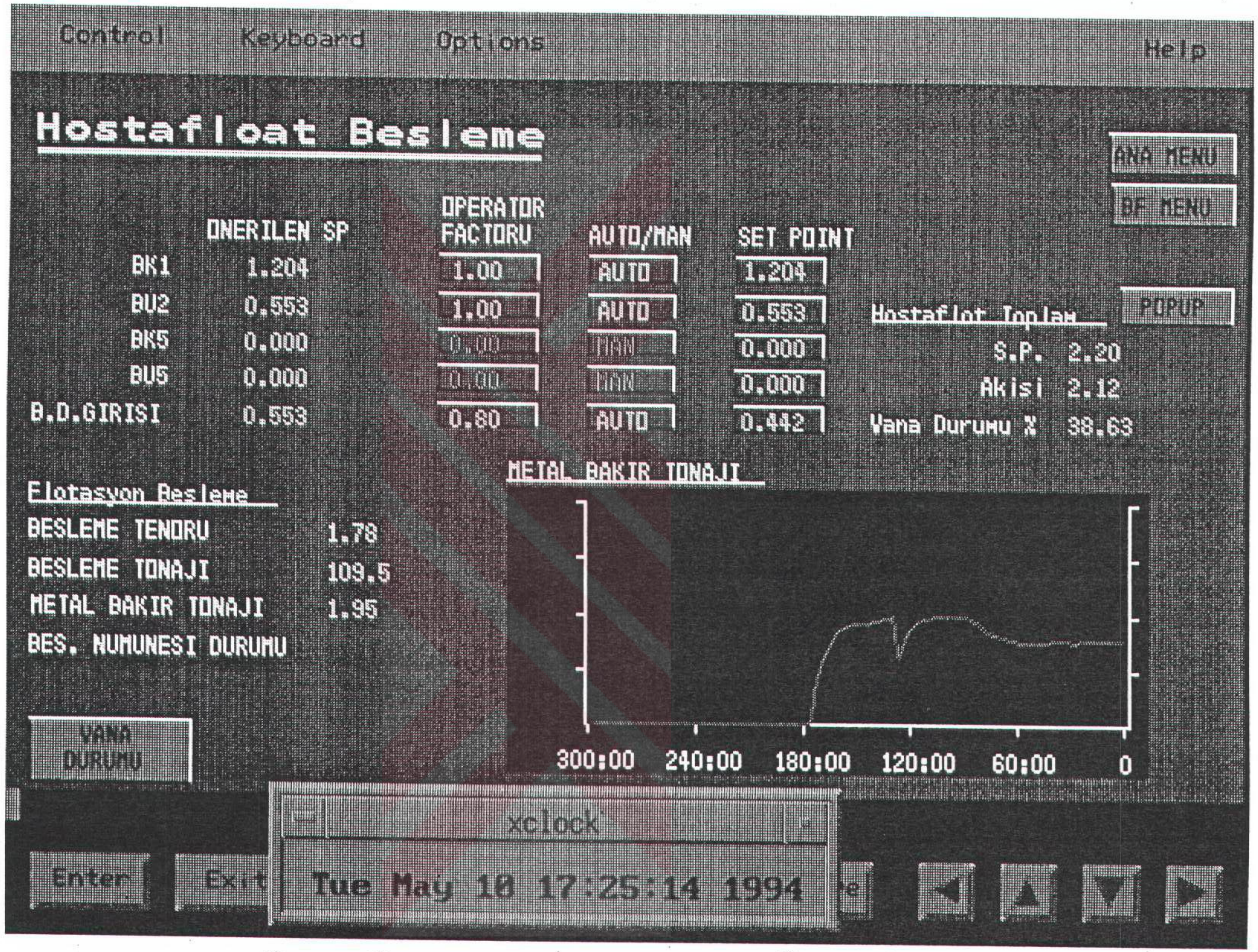



Figure 3.24 View of hostafloat feeding display.

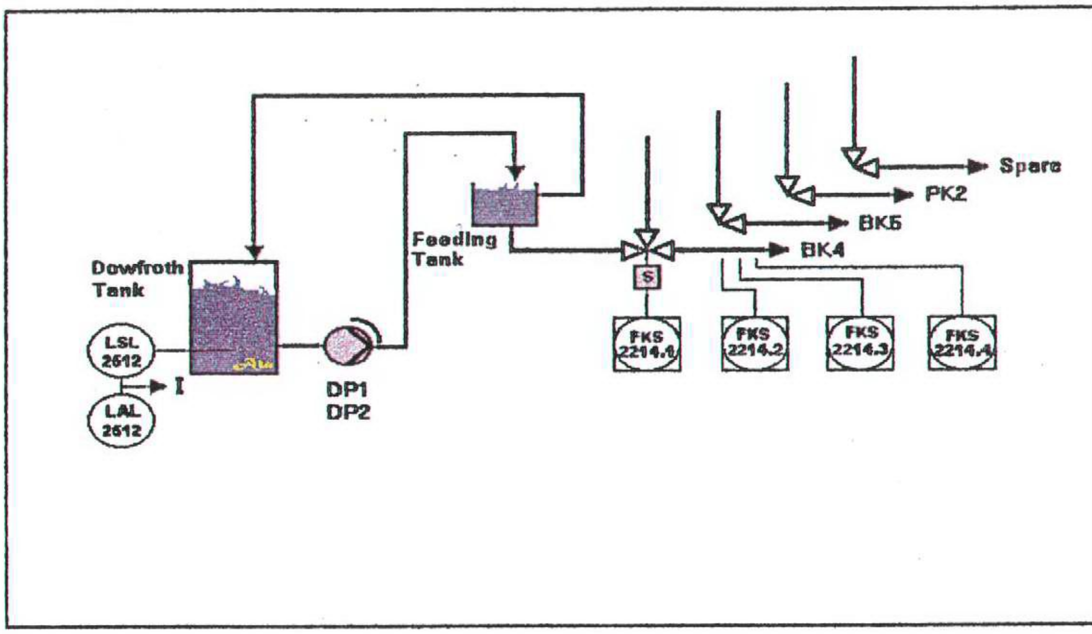
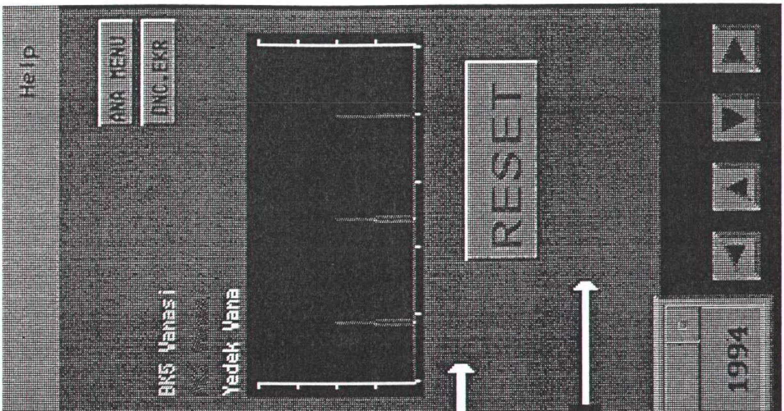



**Figure 3.25 Dowfroth preparation and feeding control diagram.**




isplay.