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Sammendrag, innholdsfortegnelse eller innholdsbeskrivelse

Kopi av en rapport som ble bestilt av Outokumpu,mining service.
Den konkluderer med at fortynningen av malmen fra starten og fram til nå har ligget på 10-15 %, men at den nå har tendens til å øke grunnet mer komplekse gruveforhold. I hovedsak skyldes dette malmkropp/-linsers uregelmessighet.
Økonomisk vil en reduksjon i fortynningen øke inntekten med 1-1,5 %. Det bør være mulig å redusere fortynningen med 5 %.
Det gis forslag til tema som kan bedre fortynningsfaktoren.

OUTOKUMPU

MINING SERVICE

ANALYSIS OF ROCK DILUTION IN GRONG GRUBER

BY

SONG XIAOTIAN

1989

GRONG GRUBER A/S

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Introduction

The purpose of this research is to carry out numerical analysis of rock dilution of Grong Gruber A/S and to study the realistic possibilities to get it down,to control ore grade and in this way to improve the economy of the mine.
The Grong Gruber is mining the Joma massive sulphide deposit situated in the north of the central Norwegian Caledonides.The strike,dip and thickness of the orebody is variable.The mineralized zone consists of pyrite,chalcopyrite and sphalerite.The surrounding rocks are mainly greenstones and green scists.The orebody is 1500 m long,with a known depth of 1000 m.
The annual production is about 500 kt with average grades of 1.35% Cu and 1.70% Zn.

1. Rock dilution problem

The mining method is normally by sublevel stoping and R & P mining. The yearly production consists of about 30 % from development drifts, 30% from R & P systems and 40% from long hole sublevel stoping.The Joma orebody is not regular and has a variations in thickness,ore/ waste boundary, grades and mineral types.The orebody conditions with irregular boundary and uneven distribution of ore grades at Grong Gruber are major nature factors influencing the rock dilution.The orebody conditions,mining methods and the choice of equipments determine the mine ore production,the rock dilution level and production costs. The factors influencing the rock dilution can be illustrated as in fig.1. Estimating rock dilution in Grong Gruber.

In general no direct figures of rock dilution were available at the mine. The data concerning the rock dilution calculation at Grong Gruber is shown in App.1.Based on the data of ore reserve in situ for 1972-1987 and ore mined for corresponding time period,the rock dilution can be estimated approximately.

According to formula presented as follows(see App.2):

$$d = (a(0) - a)/a(0) * 100 \qquad (1)$$

Where: d-rock dilution(grade percentage),%,

a(0)-ore grade in situ,%;

a- ore grade mined,%.

On the basis of estimating grades of produced reserves in situ for 1972-1986(see App.1-1,1-2)and calculating grades of ore mined for the same period(see App. 1-3),the rock dilution in grade percentage can be estimated according to formula (1).The results of calculation are shown in table 1.

From table 1 can be seen that rock dilution for 1972-1986 long time period averages 12-13%.The same level of rock dilution can also be estimated for 1972-1984 time period(see App.1-4,1-5).The rock dilution is ranged 13-14%.See table 2.

Table 1 rock dilution estimation(1972-1986)

(1)Total ore reserves in situ

Year	tonnage (Mt)	ore grade,% Cu	Zn	Remark
1972	16.424	1.27	1.70	App.1-1
1972-1987	4.500	1.40	1.58	
Total	20.924	1.30	1.67	

(2)Current remaining reserves

1986	14.976	1.10	1.81	App.1-2

(3)Produced reserves in situ

1972-1986	5.948	1.80	1.32	(3)=(1)-(2)

(4)Rock dilution calculation

Ore reserves in situ			Ore mined(App.1-3)			Rock dilution,%	
Tonnage (Mt)	Ore grade,% Cu	Zn	Tonnage (Mt)	Ore grade,% Cu	Zn	Cu	Zn
5.948	1.80	1.32	5.457	1.56	1.16	13	12

Table 2 Rock dilution estimation(1972-1984)

Ore reserves in situ			Ore mined			Rock dilution,%	
Tonnage (Mt)	Ore grade,% Cu	Zn	Tonnage (Mt)	Ore grade,% Cu	Zn	Cu	Zn
20.924-15.661=5.263	1.84	1.28	4.525	1.59	1.10	13.6	14.0

In real operating situation the rock dilution depending on specific conditions and mining methods may vary widely, from 5% to 20% and more.See App.1-6,and 1-7.The variations in rock dilution in dependence with various conditions in planning stage are shown in table 3.

Table 3 Rock dilution variation

year and series	ore reserves in situ tonnage (Kt)	ore reserves in situ Cu grade,%	ore mined tonnage (Kt)	ore mined Cu-grade %	rock dilution %
According to plan					
1982-1985	1844	1.50	1747	1.44	4
According to major production blocks					
1972-1984	5670	1.83	4525	1.59	13
According to series of orebody					
X-series	3522	1.55	2976	1.33	14
Y-series	392	2.19	311	1.80	18
X+Y+B1+B2 series	5396	1.73	4427	1.47	15

The data presented in tables 1, 2, and 3 should give an idea of general situation of rock dilution level at Grong Gruber. Due to the employed mining methods with limited dilution and owing to stability of unmined pillars and hanging wall and also in accordance with complicated conditions of Joma orebody,the estimated dilution figures can be considered realistic and reasonable.Based on the above calculation and discussion the average dilution level at Grong Gruber should be ranged 10-15%,but in future it will probable be difficulty to control rock dilution within this level owing to mining of south-wester thin orebody and extraction of existing pillars.

2. Numerical analysis of rock dilution

In order to study how the rock dilution can be controlled, the irregularity and complexity of Joma orebody and its influence on rock dilution should be analyzed.

2.1 Ore grade distribution

The ore grade distribution of Joma orebody in situ is usually distributed uneven. The ore grade distribution of the drill holes and the places and blocks in situ is shown in App.3-1, table 4 and fig. 2.

Table 4 Ore grade distribution in situ

Place	Mean(a),%		Stand. deviation(s)		Coeff. of variation(v) %	
	Cu	Zn	Cu	Zn	Cu	Zn
Along X series	1.15	1.86	0.307	0.23	27	12
Perpendicular X series	1.16	1.87	0.76	1.09	65	58
Total blocks	1.17	1.76	0.76	0.70	65	40
Drill hole1028	1.70	1.22	1.52	1.23	89	100
Drill hole 1029	0.71	1.88	0.85	1.92	119	102

Table 4 shows that the coeff. of variation of ore grade of Joma orebody in blocks is ranged between 40-65% and along the X series have less coeff. of variation, i.e. 12-27% and v in drill holes varies 100% or so.

According to classification of unevenness of distribution(App.4), the Joma orebody is related to gradational distribution (v=20-100%). The degree of unevenness gives us an idea where exists possibilities to change ore grade and to improve ore grade.

The ore grade mined is also distributed uneven. The ore grade fluctuation during monthly production(April 1989) is shown in App.5 and fig.3. It shows that the coeff. of variation of ore grade mined varies 23-27%.

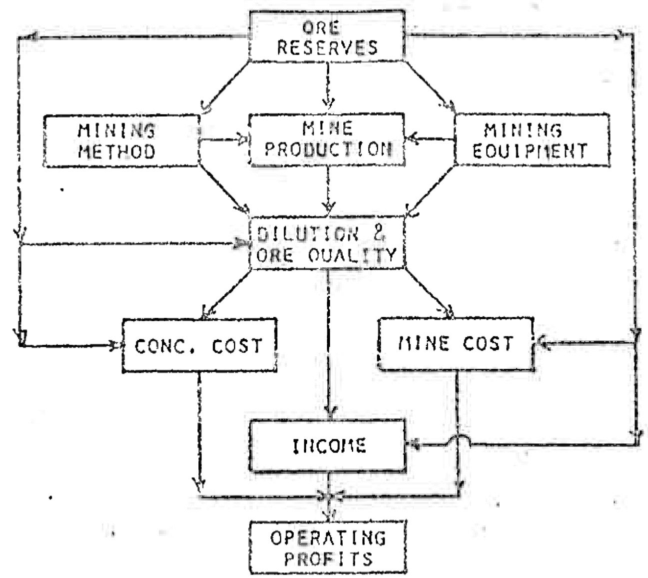


Fig. 1 Factors influencing rock dilution, costs, ravenues and profits.

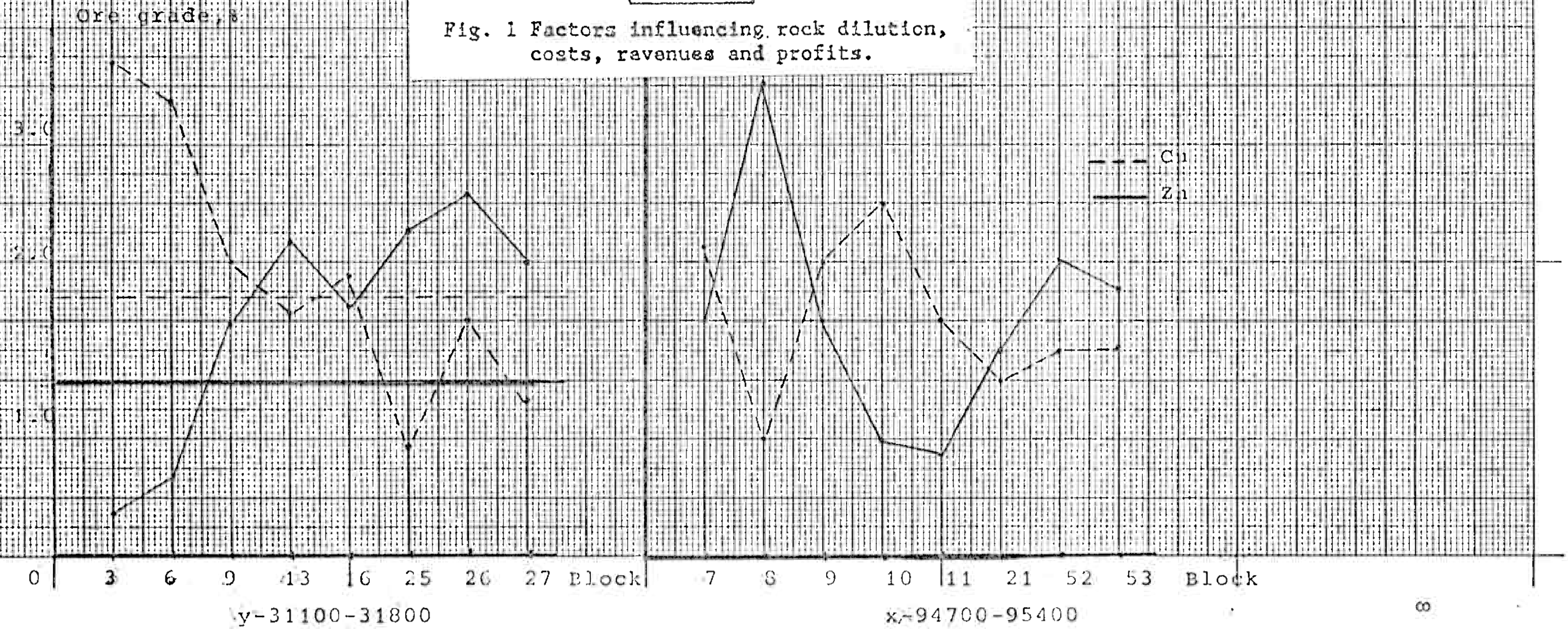


Fig.2

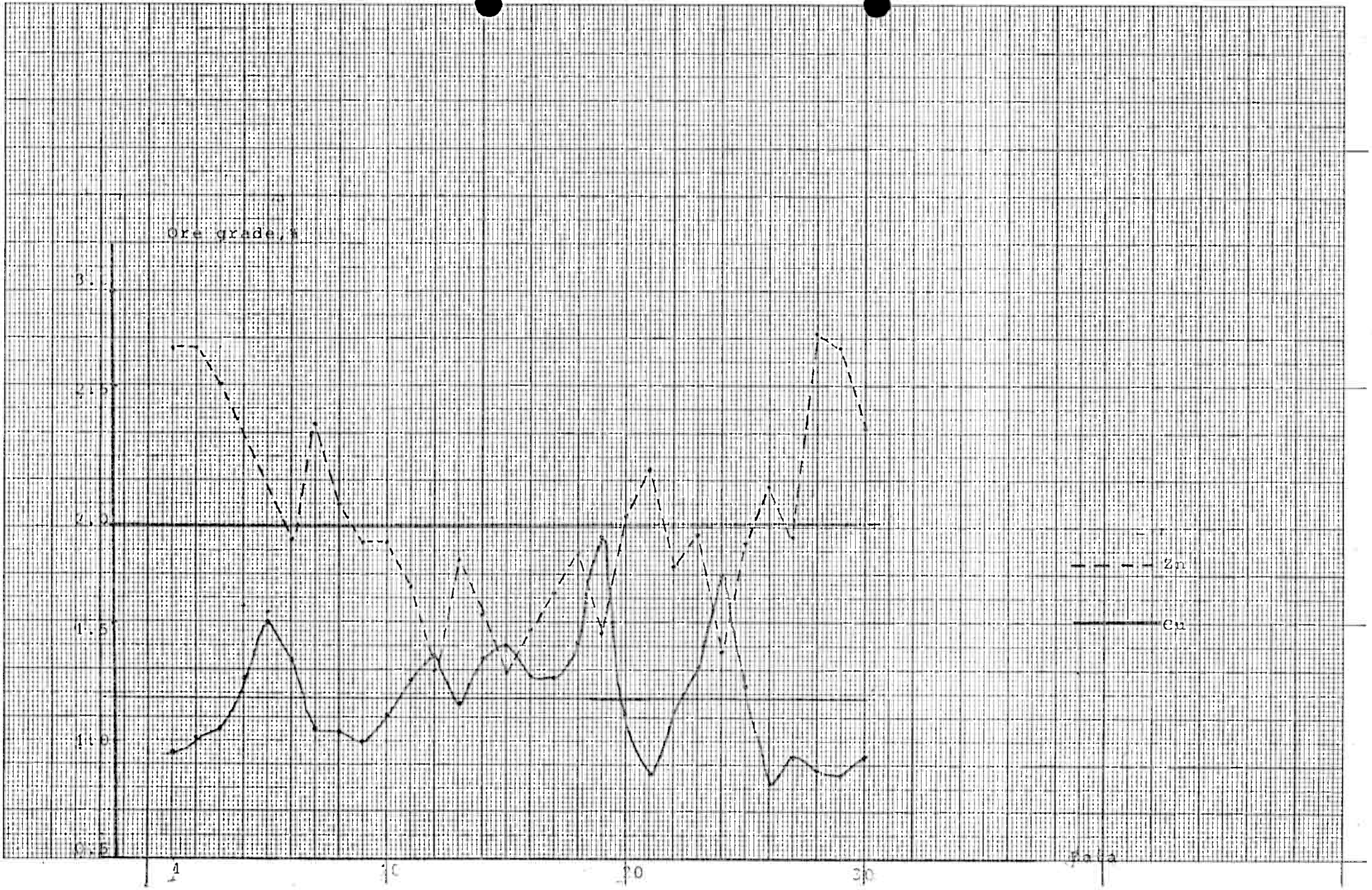


Fig.3

2.2 Influence of irregular hanging wall on dilution

The irregular part of Joma orebody has a unstable boundary on hanging wall,which is often taken undulating pattern.In this case the rock dilution occurs due to irregular ore/ waste contacts on hanging wall,see App.6.The influence of irregular hanging wall on dilution is shown in table 5.

Table 5 Rock dilution on the boundary of hanging wall (429 BSK-1,X=95140,Y=31600)

Influencing factor	Ore tonnage,t	Ore grade,% Cu	Zn	Rock dilution,% by grade Cu	Zn	by weig:
Without waste	28086	0.83	3.0	-	-	-
Including waste	28398	0.76	2.86	8	5.´	5

Table 5 shows that the irregular hanging wall influences on dilution of 5-8% and leads to decrease ore grade mined of o,o7-0.14%.

2.3 Influence of flat-dipping footwall on dilution

The Joma orebody sometimes has a plunge of 20-40^{0} on the foot-wall.It is necessary to cut waste rock of footwall in order to outline the stoping boundary.An example of this influence on dilution is shown in App.7 and table 6.

Table 6 Rock dilution on the boundary of footwall(2/x/LH)

Profile		x-95100	x-95120	x-95140	x-95160	Average
Difference in grades,%	Cu	0.02	0.02	0.01	0.04	0.02
	Zn	0.03	0.03	0.01	0.07	0.03
Rock dilution,%	Cu	1.4	2.0	1.0	4.0	2.1
	Zn	1.4	2.0	1.0	4.0	2.1

From table 6 it will be seen that the flat-dipping footwall influences on dilution of 2% and leads to decrease grade of 0.02-0.03%.

2.4 Influence of waste layer inside orebody

In case of presence of waste layer which can be mined selectively, the influence of waste layer may be considered. An example of that influence is shown in App.8 and table 7.

Table 7 Influence of waste layer on dilution(416 BSK)

Profile	Rock dilution,% by grade Cu	Zn	by weight	Change in grade % Cu	Zn
X-95120	16	15.6	16	0.20	0.28
X-95100	8.8	9.1	8.9	0.16	0.16
X-95080	7.2	7.5	7.4	0.11	0.13

Table 7 indicates that the influence of waste layer on dilution in giving example is strong. Rock dilution reaches to 7-16% and ore grade decreases to 0.11-0.28%.

2.5 Influence of thickness of orebody on dilution

In case of narrow vein orebody the influence of waste mixture on dilution is obvious. The dilution caused by waste mixture as an example is shown in App.9 and table 8.

Table 8 Rock dilution in case of orebody with thickness less 3m.

	Tonnage,t	Ore grade,% Cu	Zn	Rock dilution,% Cu	Zn
Ore	12494	o.37	3.7		
Wasee	1960	0	0		
Ore+waste	14454	0.26	3.2	13.3	13.5

From table 8 it can be found that the dilution can be reached to 13% and ore grade is decreased by 011 -0.5% in case of orebody with thickness less 3 m.

2.6 Influence of stoping boundary on dilution

In order to illustrate how the stoping boundary influences on dilution,the block 2/x/LH(416 BSK,x-95100) may be used as an example.The various alternatives of stoping boundary correspond to different dilution.Theresults calculated by comparising stoping boundary with orebody boundary are shown in App.10 and able 9.

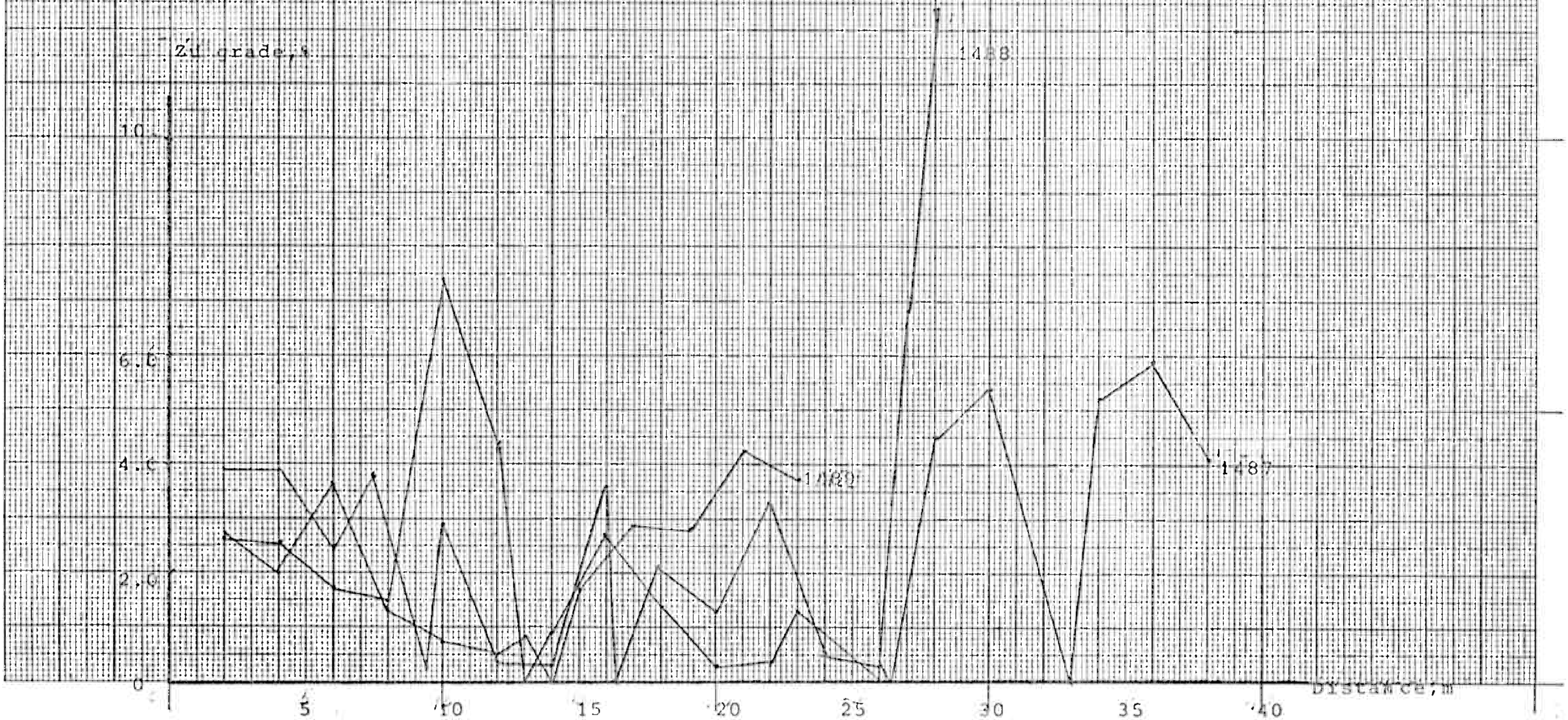
Table 9 Influence of stoping boundary on dilution

Stoping boundary	Mining area m^2	Ore grade,% Cu	Ore grade,% Zn	Change in grade,% Cu	Change in grade,% Zn	Change in dilution % Cu	Change in dilution % Zn
Alt1(original)	943	1.39	2.05	0	0	0	[illegible]
Alt.2	815	1.54	2.17	+0.15	+0.12	-11	-
Alt.3	1153	1.37	2.10	-0.02	+0.05	+1	-

Table 9 shows that the Alt. 2 with less dilution is better than Alt.1and 3.The rock dilution is reduced by 6-11% and corresponding ore grade is increased by 0.12-0.15%comparising with Alt.1,the original design.The example also indicates that design of stoping boundary is essential for rock dilution control in case of complicated orebody.The suitable stoping boundary in planning stage must always be adapted to the actual mining conditions.

2.7 Influence of ore grade distribution on dilution

The ore grade distribution of drill holes 1487,1488 and 1489 through the orebody is shown in fig.4.The ore grades are usually distributed uneven and occasionally with high grades in the boundary.It is necessary to pay more attention to choice the stoping boundary in case of presence of high grade area.



Zn grade, %
10
6.0
4.0
2.0
0
5
10
15
20
25
30
35
40
Distance, m
1488
1482
1487

Fig.4

So as to illustrate how the rock dilution is affected by ore grade distribution three alternatives of stoping boundary can be considered for instance:

N1-Alt1 with waste dilution and without ore losses of high grade area;

N2-Alt. with ore losses of high grade area and without waste dilution;

N3-Alt.in between Alt. 1 and 2.

The results calculated for Alt. 1,2 and 3 are shown in App. 11 and table 10.

Table 10 Difference in ore grades for various alternatives

Alt.	Difference in ore grades mined,%					
	Cu for drill holes			Zn for drill holes		
	1487	1488	1489	1487	1488	1489
N1	-0.14	-0.11	-0.09	+0.06	+0.25	-0.01
N2	0	0	0	0	0	0
N3	-0.06	-0.04	+0.01	+0.13	+0.28	+0.22

Table 10 shows that the Cu grade of Alt.1 and 3 is decreased in comparision with Alt.2 in case of absence of Cu high grade area on the boundary.On the contrary,the Zn grade of Alt. 1 and 3 is increased in regard to Alt. 2 if there is Zn high grade area on the boundary.The data in table 10 also indicates that the Alt.3 with increasing in Zn grade of 0.13-0.28 is nearing to optimum than Alt.1 and 2.The optimum stoping boundary can be outlined in each specific case of presence of high grade area.

3. Economical analysis of rock dilution

The purpose of an economical analysis of rock dilution is to formulate a mathematical model providing a means for analyzing the effects of changes in dilution and other variables.The approch is to establish the interrelations between variables connected with dilution.Mining conditions differ in many ways and have unique features so that model of rock dilution have to be tailored to fit the specific mining conditions of Grong Gruber.

3.1 Relationship between dilution and profit

The relationship between dilution and income,costs and profit in mine operations can be expressed by:

$$P=\left(I_1\frac{r_1}{g_1}a_1+I_2\frac{r_2}{g_2}a_2\right)(1-d)-C \tag{2}$$

Where:

- P- specific profit,Kr/t;
- I_1,I_2-Cu and Zn concentrate value,respectively,Kr/t;
- r_1,r_2-Cu and concentrate recovery,part;
- a_1,a_2-Cu and Zn ore grade in situ,part;
- g_1,g_2-Cu and Zn ore grade in concentrate,part;
- d- rock dilution,part;
- C-specific total cost,Kr/t.

According to data of costs and price of concentrates at Grong Gruber shown in App. 12 and table 11,the profit is expressed in terms only of ore grade and rock dilution by substituting in Eq. (2):

$$P=\left(2252\frac{0.86}{0.23}a_1+1901\frac{0.82}{0.45}a_2\right)(1-d)-142$$

$$=(8490\ a_1+3489\ a_2)(1-d)-142 \tag{3}$$

Table 11 Data of costs and price

Operation costs

year		1988	1-8.1989
Mining, Kr/t	fixed	-	30.92
	variable	-	47.29
	total	67.43	78.21
Mill Kr/t		40.33	41.72
Sum.direct Kr/t		107.76	119.93
Indirect Kr/t		34.24	38.92
Total Kr/t		142.00	158.85

Price of concentrates(1989 plan)

Cu concentrate, Kr/t	1986
Cu+Ag concentrate, Kr/t	2252
Zn concentrate, Kr/t	1901

The relationship between profit,costs and rock dilution for each ore grade is shown in fig.5.The profit depending on ore grades would be as follows:

$a_1=1.4\%, a_2=1.3\%$; $P=164(1-d)-142; d=0, P=22$ Kr/t

$a_1=1.6\%, a_2=1.4\%$; $P=185(1-d)-142; d=0, P=43$ Kr/t

$a_1=1.8\%, a_2=1.5\%$; $P=205(1-d)-142; d=0, P=63$ Kr/t

$a_1=2.0\% a_2=1.6\%$; $P=226(1-d)-142; d=0, P=84$ Kr/t

From fig.5 it can be seen that how the income and profit(P) are affected by combination of ore grade(a)and rock dilution (d).The relationship expressed by Eq.(2)presents following discussing:

(1)how will dilution and ore grade influence on income.

The income is expressed by $I=(I_1r_1a_1/g_1+I_2r_2a_2/g_2)(1-d)$

Compare two cases,differing only in dilution.

Thus

$$\frac{I(1)}{I(2)}=\frac{1-d(1)}{1-d(2)}$$

or $$d(1)=1-\frac{I(1)}{I(2)}+\frac{I(1)}{I(2)}d(2) \qquad (4)$$

The changed situation is giving index 1 and the present situation-index 2.

According to Eq.(4),we thus find that the income,I increases by 3.5% in case of reduction of dilution from 13.5%to 10%,see fig.6.The calculation shows that the reduction of dilution of 1% leads to improving income by about 1-1.5% depending on dilution level.

Example 1. Taking a_1=1.5%,a_2=1.8%,the income will be:

For d=10%,I(1)=(8490*1.5%+3489*1.8%)(1-10%)=171.14 Kr/t

For d= 9%,I(2)=(8490*1.5%+3489*1.8%)(1-9%) =173.04 Kr/t

I(2)-I(1)=2 Kr/t

This means that reduction of 1% dilution will lead to increase income about 2 Kr/t.

Example 2. Taking d=10%,the income varies:

For a_1=1.5%,a_2=1.8% I(1)=171.14 Kr/t

For a_1=1.55%,a_2=1.85% I(2)=176.53 Kr/t

I(2)-I(1)=5 Kr/t

It showed that improving in ore grade of 0.05% leads to increase the income about 5 Kr/t.

As has been mentioned,the influence of orebody boundary on dilution may be reached to 5%.If the rock dilution would be decreased by 5%,from 13%,estimated for 1972-1986,to 8%,the Cu ore grade mined would increase from 1.56% to 1.66%.The difference in income of Cu concentrate would be:

8490(0.0166-0.0156)=8.5 Kr/t!

(2)How will costs be reducted with decreasing ore grade provided profit is not changed.

Calculation:

For a_1=1.5%,a_2=1.4%,d=10%,P=(8490*1.5%+3489*1.4%)(1-10%)-142=17 Kr/t

For a_1=1.45%,a_2=1.35%, P=11 Kr/t

It is obvious that decreasing ore grade of 0.05% caused reductio in profit of 17-11=6 Kr/t.In order to maintian costant profit we have to cut costs from 142 Kr/t to 136 Kr/t,i.e.P=153-136=17.

For giving example it is necessary to cut costs of 142-136/142 =4% in case of reduction of ore grade of 0.05% for providing constant profit level.

(3) It is considered to decrease dilution with decreasing ore grade provided profit and costs not changed at the same time.

Calculation:

The ore grade is decreased from a_1=1.5%,a_2=1.4% to a_1=1.45%, a_2=1.35% and dilution from 10% to 6%,the profit will be the same.

Thus P=(8490*1.5%+3489*1.4%)(1-10%)-142=17 Kr/t

P=(8490*1.45%+3489*1.35%)(1-6%)-142=17 Kr/t

It shows that the reduction of o.05% ore grade is compensited by decreasing dilution of 10% - 6%=4%,providing the profit is not changed.

3.2 Optimum ore grade mined in planning

As has been discussed,the ore grade distribution of Joma orebody in situ is usually uneven.There seemed to exist a optimum ore grade mined especially in case of unevenness of ore grade distribution and existence of high grade area on the boundary. The general model of optimum ore grade mined is shown in App.13. According to optimum model,the ore grade mined can be expressed by formula:

$$a = \frac{m*a_1+(w-m)*a_2-(w-m)^2*a_2/2*n}{w} \qquad (5)$$

Where:

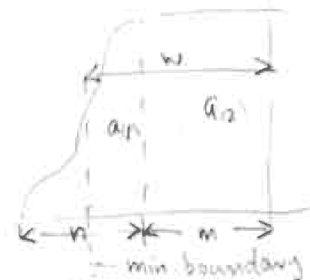
a-Ore grade mined;%;

a_1-High grade in situ,%;

a_2_Low grade in situ,%;

w-Stoping width,m;

m,n-See fig. in App.13.



The relationship between ore grade mined and stoping width for different grade distribution is shown in table 12 and fig.7.

Table 12 Relationship between ore grade mined and stoping width ($m=10$ m, $n=3$ m, $a_1=1\%$ (Cu), $a_1=2\%$ (Zn))

Stoping width, m	Ore grade mined, a, %					
w	Cu			Zn		
	$a_2=1.5\%$	$a_2=2.0\%$	$a_2=2.5\%$	$a_2=3.0\%$	$a_2=3.5\%$	a_2
11.0	1.02	1.06	1.10	2.05	2.07	2.12
11.5	1.01	1.07	1.10	2.03	2.08	2.13
12.0	1.00	1.06	1.11	2.00	2.06	2.11
12.5	-	1.03	1.09	-	2.03	2.07
13.0	-	1.00	1.06	-	2.00	2.03

It should be observed that the maximum ore grade mined corresponding optimum stoping width is increased with increasing the unevenness of grade distribution.

In order to illustrate how the stoping width actually influences on the ore grade mined, the stoping block with high grade in boundary may be used as an example. see App.14. The relationship between stoping width and ore grade mined is shown in table 13 and fig.8.

Table 13 Relationship between ore grade mined and stoping width

Stoping width, m	7	12	17	20
Zn grade mined, %	1.08	1.63	2.51(*)	2.30

In this case the optimum stoping width is about 17 m correspondi the maximum grade mined(*), despite the fact that the optimum stoping boundary has more dilution by weight percentage.

3.3 Sensitivity analysis

The profit function, formula (2) will be used for sensitivity analysis to variables changes in order to study the influence of various factors on the profit.

The formula (2) may be rewritten:

$$P=(I_1'*r_1*a_1+I_2'*r_2*a_2)(1-d)*1000-C \qquad (6)$$

Where:

I_1', I_2'-Price of Cu and Zn contents respectively, Kr/kg;

r_1, a_1, r_2, a_2, d, C-ditto.

The results of sensitivity analysis are shown in table 14 and in the spider diagram as shown in fig.9.

Table 14 Sensitivity analysis
(I_1'=20 Kr/kg, I_2'=12 Kr/kg, r_1=0.87, r_2=0.83
a_1=1.4%, a_2=1.3%, d=10%, C=142 Kr/t)

Influencing factor	Calculation formula	Change in variables %	Change in profit,%
Price,Kr/kg	$P=(12.18*I_1'+10.79*I_2')*0.9-142$	± 10	± 17
Ore grade,%	$P=(17400a_1+9960a_2)*0.9-142$	± 10	± 17
Costs,Kr/t	$P=335.77-C$	± 10	∓ 7
Dilution,%	$P=373.08(1-d)$	± 10	∓ 3

The diagram shows that the price;I' and ore grade,a,are more sensitive as most important factors influencing the profit. Sensitivity analysis also shows that the influence of dilution' on change in profit is dependent upon the level of dilution i.e. the higher dilution level the bigger change in profit at the same change in dilution of 10%.<see table 15.

Table 15 Influence of dilution level on profit change in dilution $\pm$10%

Dilution level,%	10	15	20	30
Change in profit,%	∓ 2	∓ 3	∓ 5	∓ 9

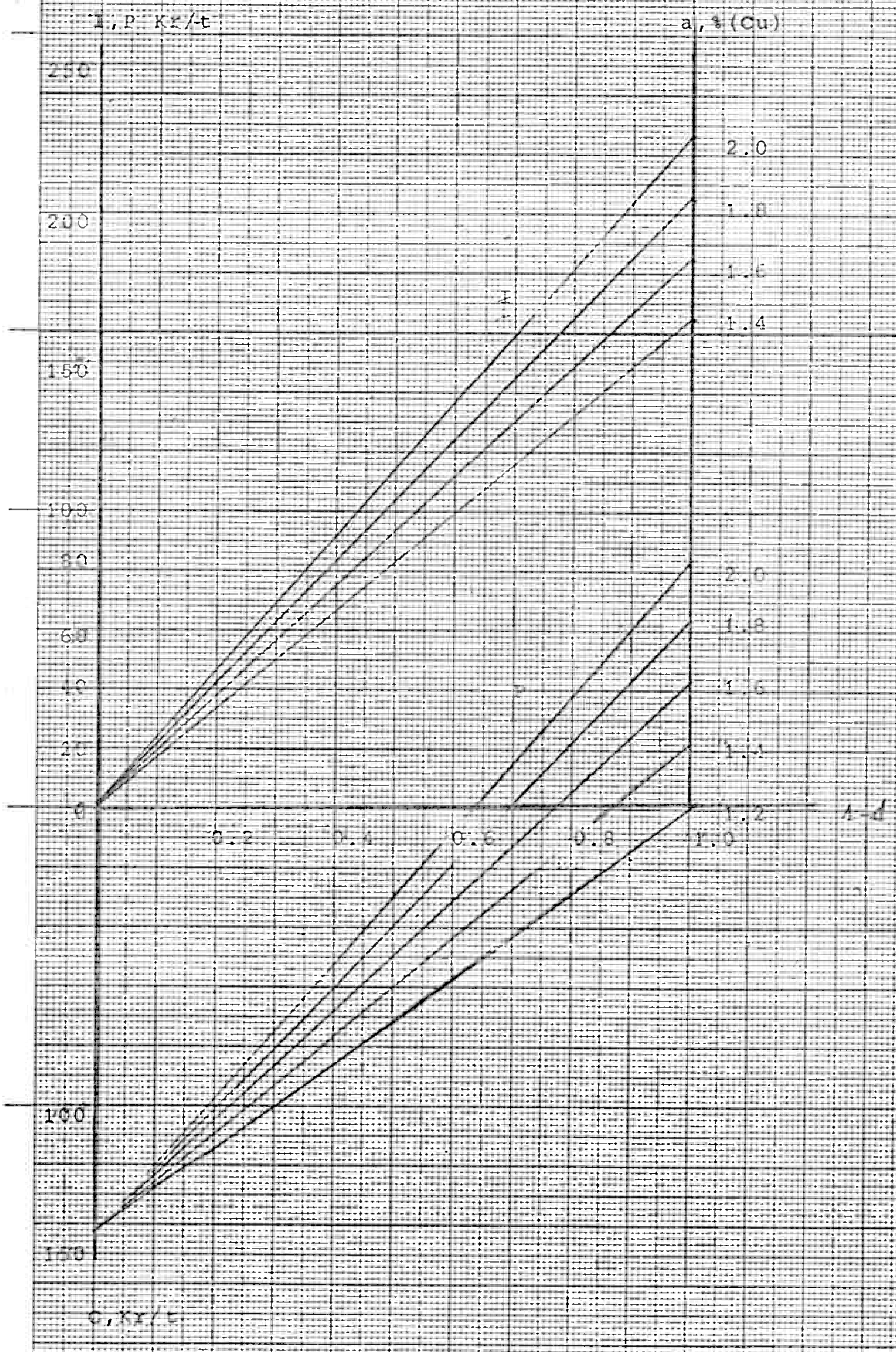

I, P Kr/t
a, % (Cu)
250
200
150
100
80
60
40
20
0
2.0
1.8
1.6
1.4
2.0
1.8
1.6
1.4
1.2
I
0.2
0.4
0.6
0.8
1.0
1-d
100
150
C, Kr/t
n

Fig. 5

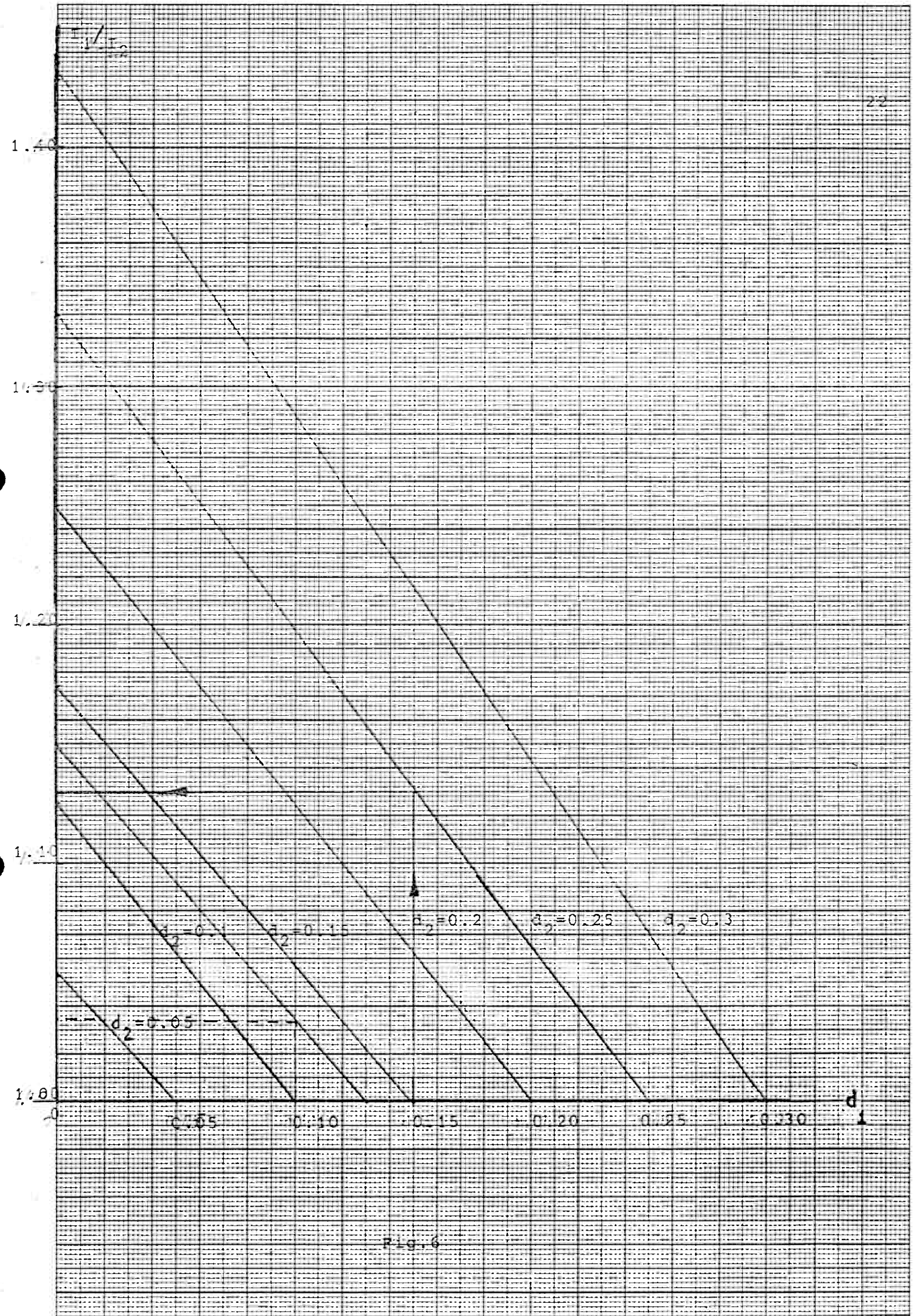

I_1/I_2
1.40
1.30
1.20
1.10
1.00
0
0.05
0.10
0.15
0.20
0.25
0.30
d_1
d_2=0.05
d_2=0.1
d_2=0.15
d_2=0.2
d_2=0.25
d_2=0.3

Fig.6

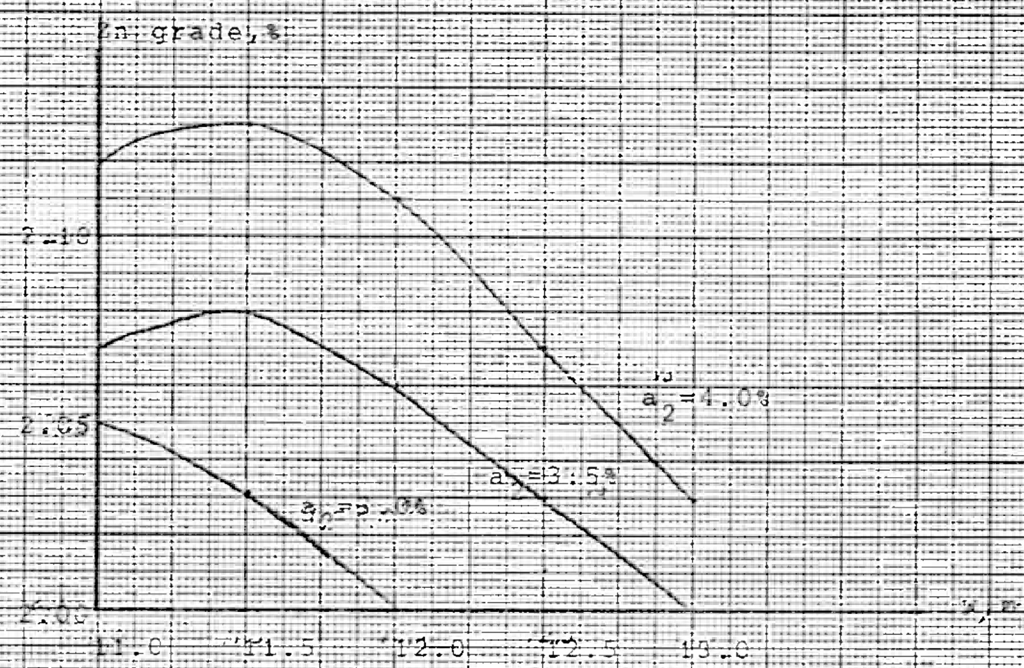

Zn grade,%
2.10
2.05
2.00
11.0
11.5
12.0
12.5
13.0
x, m
$a_2 = 4.0\%$
$a_2 = 3.5\%$
$a_0 = 3.0\%$

Fig.7

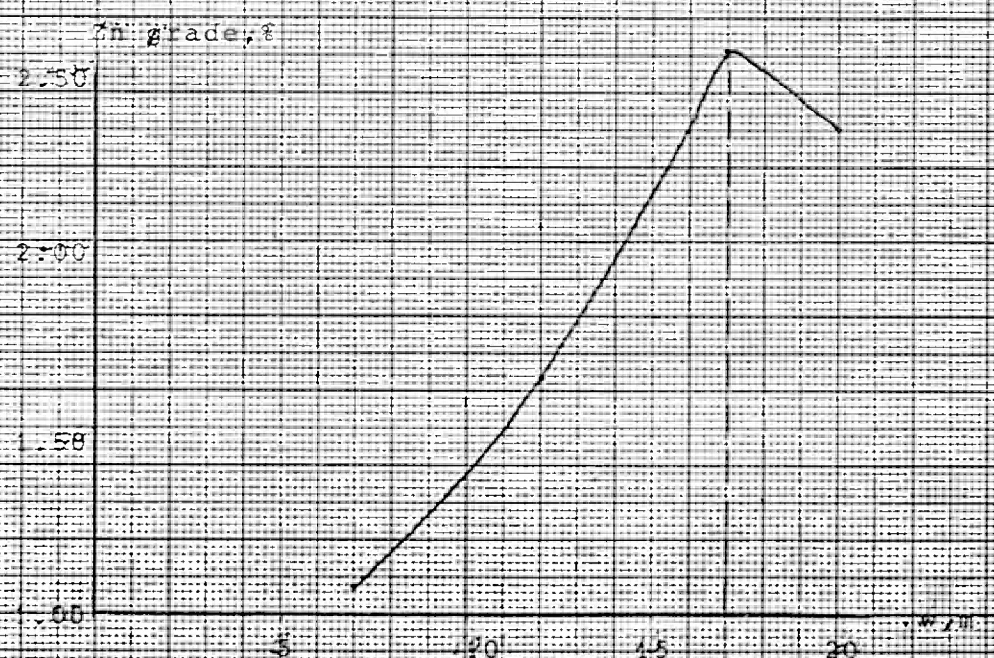

Zn grade,%
2.50
2.00
1.50
1.00
5
10
15
20
w, m

Fig.8

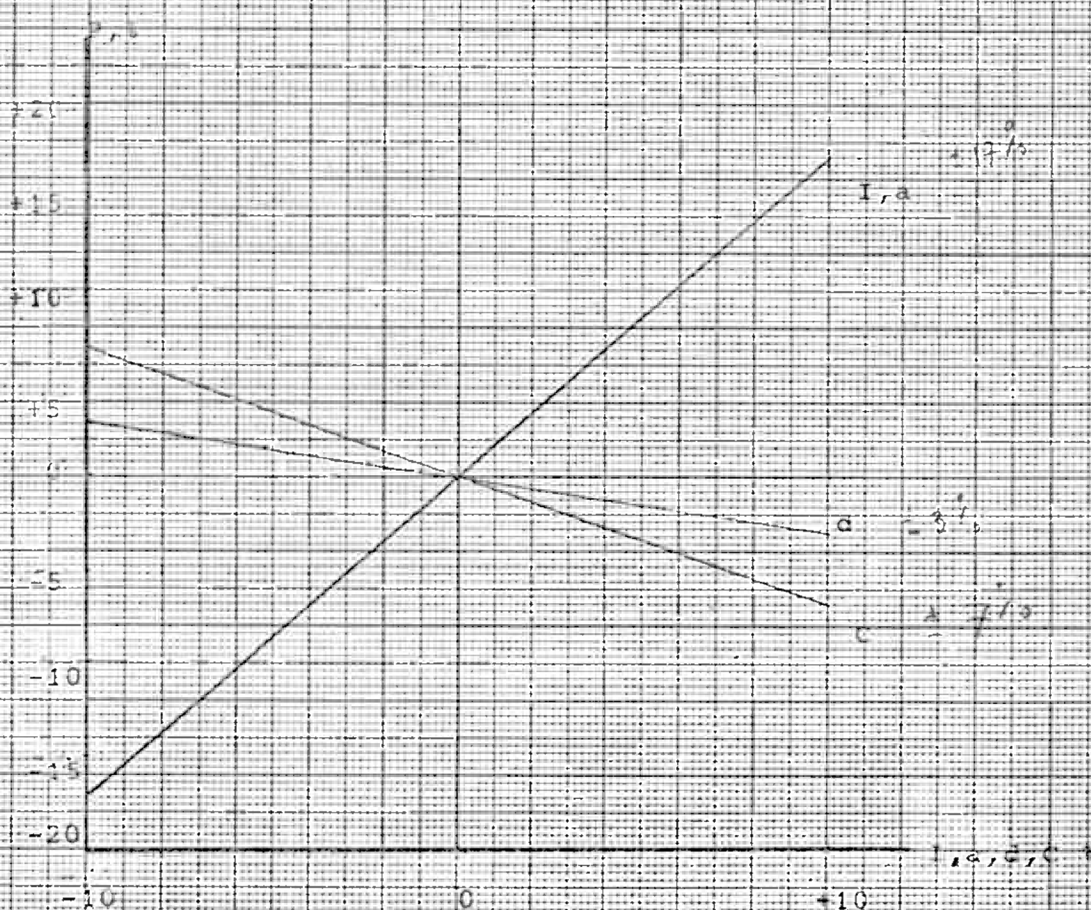


Fig. 9

3.4 Optimum model of rock dilution

The complete formulation of optimum dilution model for specific mine is a complicated problem.It is connected with special experiments to get interrelations between parameters.In order to illustrate how the model can be built up,the model with simplicity may be presented as an example.The simple model of rock dilution can be described by following steps:

(1)To establish the relationship between mining cost,C_m and its corresponding dilution,d:

$$C_m=F(d) \qquad C=C_c+C_m+C_0 \tag{7}$$

The planning mining cost of employed mining methods is shown in table 16.

Table 16 Mining cost(1988)

Mining methods	Cost, Kr/t	Expected dilution,%
Development drifts		
R&P system		7-11
LH sublevel stoping		14-22
Total	51	10-15

(2)To find the relationship between the concentrate recovery,r and rock dilution,d:

$$r=F'(d) \tag{8}$$

The relationship between r and a is shown in fig.10.According to fig.10 the function of $F'(d)$ is expressed by:

$$r=k(a_0(1-d))^{\frac{1}{2}}$$

(3)Finally,substituting formulas (7)and (8) into formula(2) we obtain optimum model of rock dilution as follows:

$$P=\frac{I_1}{g_1}k_1(a_1(1-d))^{3/2}+\frac{I_2}{g_2}k_2(a_2(1-d))^{3/2}-(C_c+C_m+C_0) \tag{9}$$

Where:

k_1,k_2-Experimental coeff.

P,I,g,a,d,C-ditto

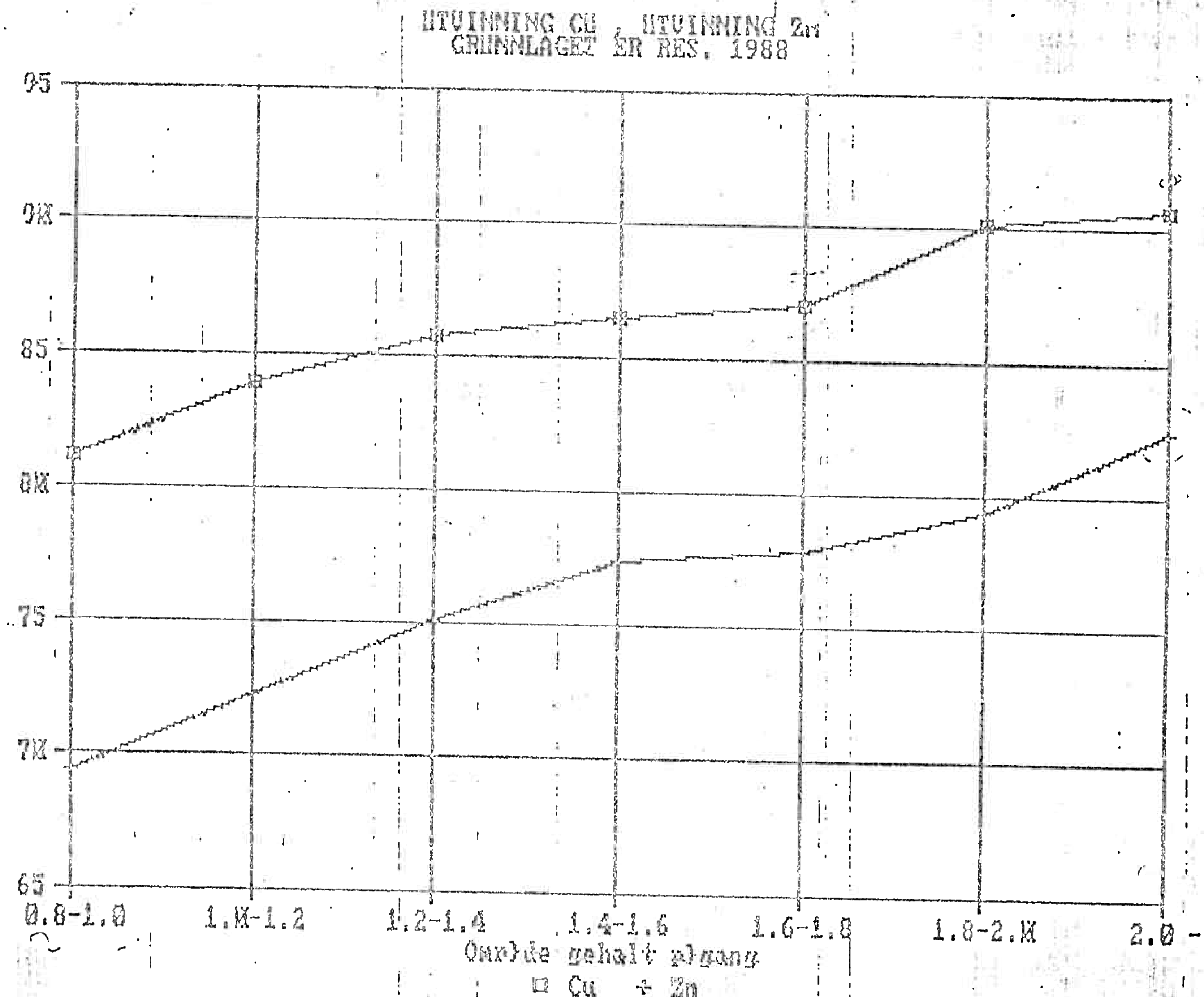

UTVINNING CU , UTVINNING Zn
GRUNNLAGET ER RES. 1988
95
90
85
80
75
70
65
UTVINNING
0.8-1.0
1.0-1.2
1.2-1.4
1.4-1.6
1.6-1.8
1.8-2.0
2.0 -
Område gehalt pågang
□ Cu + Zn

Fig.10

The relationship between P,I,C and d for giving example $k_1=7.5$, $k_2=6.5$ is shown in table 17.

Table 17 Relationship between P,I,C and d

$I_1=3094$ Kr/t, $I_2=1901$ Kr/t, $g_1=0.23$, $g_2=0.45$

$a_1=1.5\%$, $a_2=1.2\%$, $C_c=36$ Kr/t, $C_0=53$ Kr/t, $C_m=51$ (d=15%)

d	I	P, Kr/t		
%	Kr/t	for increament in C_m by		
		25%	30%	35%
5	205	36	30	23
10	189	36	34	31
15	173	33	33	33
20	159	19	19	19

Table 17 shows that the optimum dilution level corresponding the maximum profit varies in dependence with increament in mining cost, which is needed for reduction of dilution:

When 25% increament in C_m, optimum dilution level is 5-10%;

When 30% increament in C_m, optimum dilution level is 10-15%;

When 35% increament in C_m, optimum dilution level is 15% or so.

4. Rock dilution control

As stated above the rock dilution control is of more and more importance in connection with production situation at present and in future. Especially it is of great significance in respect of mining situation conditioned by complixety of orebody and pillar extraction with possible higher dilution. The production results from 1972-1988, ore reserves data for 1986 and pillar condition are shown in App.15. It can be seen e.g. that:

-The Cu grade mined during 1972-1986 exeeds the average grade of whole orebody by 0.5%, i.e. more rich Cu ore was taken for the past years.

-The Cu grade for 1981-1988 has decreased 0.17% in comparison with 1972-1980 period.

-The Cu grade of remaining reserves is below the average grade by 0.20%, and Zn grade-above average by 0.14%.

-Mining conditions at Grong Gruber are complicated owing to more irregular and thiner part of orebody envolved and in connection with difficulty of pillar extraction.

Based on above consideration and analysis in paragraph 2 and 3, the realistic appriches of rock dilution control related to stoping design and application of mining methods are listed below:

4.1. Stoping design

As has been discussed that the complicated conditions of Joma orebody influence in many ways on dilution level at Grong Gruber. Summarizing the numerical analysis of dilution caused by various conditions of orebody the possible dilution level under the influences of irregularity and unevenness of orebody are presented in table 18.

Table 18 Possible dilution level

Influencing factor.	Hanging wall	Footwall	Waste layer	Thickness of orebody	Ore grad distribution
Rock dilution,%	0-8	0-5	10-20	5-50	5-15

Table 18 shows that the rock dilution caused by complexity of Joma orebody varies widely in dependence with numbers of simultaneously influencing factors existing in mining block.The high dilution level occurs where exists combination of influencing factors.

The mixture of ore with waste rock in the irregular boundary and unevenness of grade distribution in planning stage are main factors influencing dilution level at Grong Gruber.

Prelimeraly study shows that flexible stoping design with using suggested optimum stoping boundary will lead to decrease rock dilution about 5%.The choice of suitable stoping boundary in planning stage must always be adapted to the actual mining conditions.

Suggested optimum model of rock dilution can be used for evaluating mining alternatives and selecting stoping parameters in planning stage.The optimum solution of rock dilution control would fulfill the demand of improving ore grade mined.Increasing ore grade mined is of decisive importance for improving economy at Grong Gruber.Reduction of 1% of rock dilution leads to improving income by 1-1.5%.

4.2. Stabilization of ore grade

The most important operating parameters influencing ore quality are:

-Rock dilution and in this connection ore grade mined;

-Fluctuation of ore grade and its stabilization.

The dilution and optimum ore grade mined have been described above.The stabilization of ore grade mined during production is discussed as follows:

In mine operations the fluctuation of ore grade obviously influences the dressing results.The production is to ensure uniformity in the grades or in the mineral properties of the ore. In order to illustrate how the ore grade can be controlled, the application of dynamic programming may be presented as an example.

Consider the mining sequence in LH sublevel stoping. There is an arrangement of 9 sections on each of 2 levels. Mining is going from 1-st level and downwards to 2-ed and from 1 to 9 sections and 3 sections are to be blasted in each round. See fig.11. Dynamic programming based on multi-stage decision is used for selecting the optimum sequence of sections to be blasted. The optimum sequence is chosen subject to the operational constraints and grade fluctuation requirments. The optimum criteria is to minimize the sum of deviations in grades. The results of calculations of deviations are shown in table 19.

Table 19 Calculation of grade deviation

		Alt. No 1					
Sequence		1	2	3	4	5	6
Combination of sections		1,2,3	4,5,6	10,11,12	13,14,15	7,8,9	16,17,18
Deviation,%							
	Cu	-0.34	-0.04	-0.16	0.15	-0.22	0.61
	Zn	0.20	-0.10	0.30	-0.36	0.13	-0.18
		Alt. No 2					
Sequence		1	2	3	4	5	6
Combination of sections		1,2,10	3,11,12	4,5,13	6,14,15	7,8,16	9,17,18
Deviation,%							
	Cu	-0.22	-0.15	-0.13	0.12	-0.11	0.28
	Zn	0.08	0.02	0.07	-0.13	0.06	-0.11
		Alt. No 1			Alt. No 2		
Sum deviation,%							
	Cu	1.52			1.00		
	Zn	1.30			0.05		

From table 19 and fig.11 it can be seen that the stoping sequence of Alt. No 2 has less grade fluctuation than Alt. No 1. In a real operation the mining sequence with regard to the stabilization of grades is much complex. It needs a computer simulation.

Arrangement of LH stoping

Level: 429 section and grade, Cu-Zn %	1	2	3	4	5	6	7	8	9
	0.78 2.00	0.78 2.00	0.64 2.37	0.64 2.37	1.05 2.15	1.05 2.15	0.9 2.1	0.9 2.1	0.7 1.9
Level 416 section and grade, %	10	11	12	13	14	15	16	17	18
	0.98 2.00	0.98 2.00	1.14 1.45	1.14 1.45	1.26 1.61	1.26 1.61	1.75 1.73	1.75 1.73	1.54 1.75

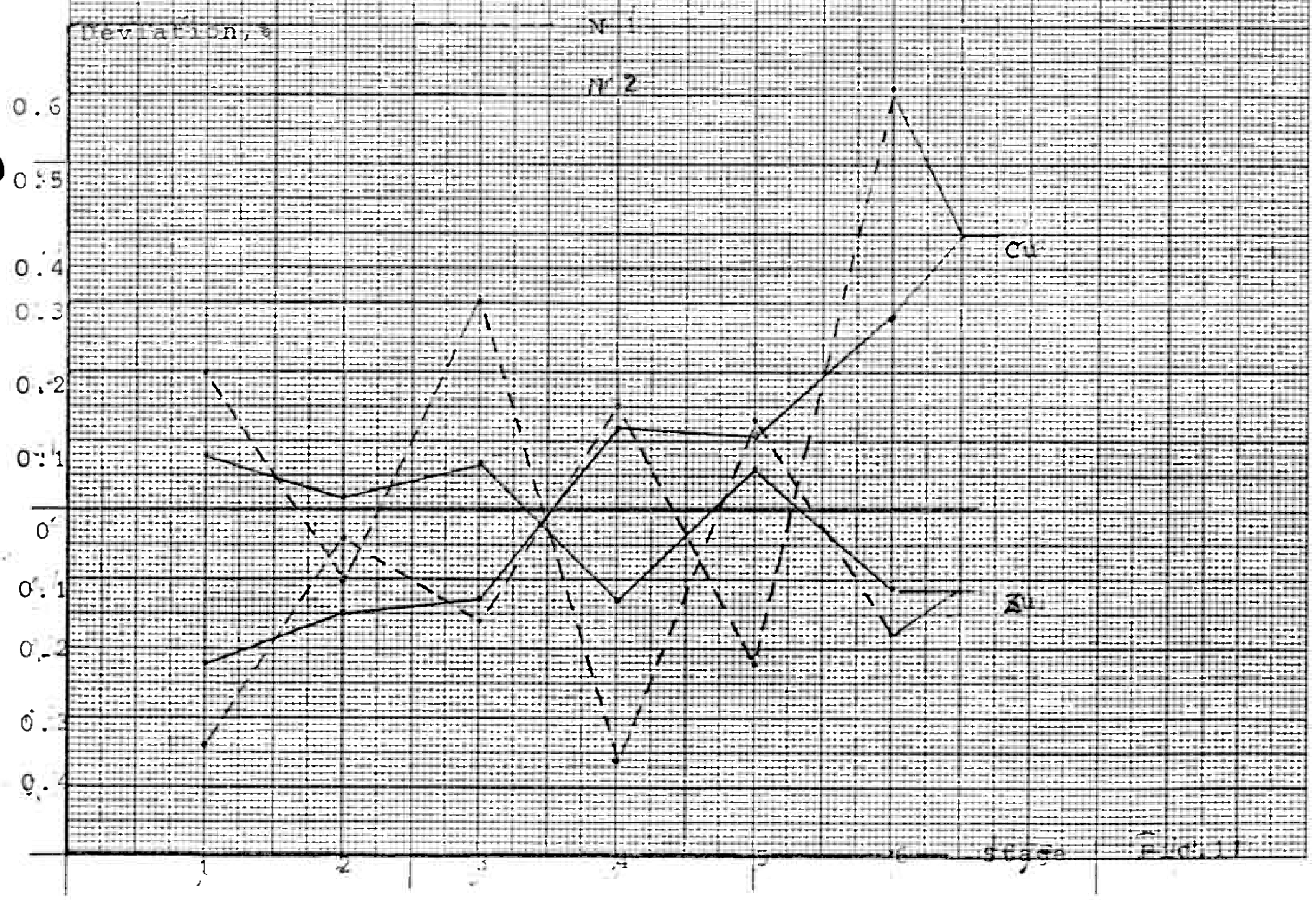


Fig. 11

4.3. Mining sequence and pillar extraction

The choice of optimum sequence at Grong Gruber provides normal ore production and ore quality control and also ensures the ground control during stoping and pillar extraction.

On selection of optimum mining sequence considerations should be taken as follows:

-Rock mechanics study of stopings and pillars by using computer simulation of Displacement Discontinuity Method.See App.17.

-Application of mathematical programming to determine production plan with consideration of ore grade control.

-Improving mining methods adapted to complicated conditions and selecting pillar extraction methods to suit the safety and the planed average grades.

During 1981-1982 years parts of pillars have been removed and big room with an area of 80*80 m and a height of 20 m without roof supporting has been formed due to excellent properties of rocks from a stability point of view and existence of a horizontal stresses in excess of the vertical stresses.It results in a stable ,selfsupporting roof.This is one of effectiv methods of pillar extraction with high recovery and minimum dilution in proper mining and rock mechanics conditions.

The investigation of pillar condition of area B1 and B2 profile (see App. 16) shows that the pillars with high ore grade and dimension of 10-20 m are suiatable for extraction.The recovery and dilution during pillar extraction are basically dependent upon the methods and conditions of area mined out.The area mined out was filled by waste rock only 30-40%.The situation of no full filling should considered in the time of pillar extraction.In the investigated area B1 and B2 the stopings in contact with area mined out are about 60%.It is necessary to take suitable means to control dilution and ore losses in the contact area.

4.4. Rock dilution for narrow vein orebody

Examine the area of B2-19-22 (360-420 level) and Y-31220-31400(520-580 level)shows that this part of orebody is a vein type with high ore grade,the veins generally lie at a shallow angle and average about 2-3 m in thickness.See App.18.In a general way formulas for defining dilution for narrow vein orebody are shown in App.19.The dilution is giving by mixing ore veins are being mined with waste rock to provide sufficient working space for mining equipment in the stopes.So the dilution level is determined by stoping dimension and equipment size.

Consider two versions of mining narrow vein orebody:

Version No 1-Application of large equipment and maintain present level of mining cost lead to incease dilution;

Version No 2-On the contrary,improving ore grade will lead to complication of mining boundary and will limit the application of large equipment,therefore mining cost will increase.

Compare two cases,differing in dilution and mining cost. thus

$$\frac{P_1}{P_2} = \frac{(I_1a_1+I_2a_2)(1-d_1)-C_1}{(I_1a_1+I_2a_2)(1-d_2)-C_2}$$

When $P_1 = P_2$

$$C_2-C_1=k(d_1-d_2) \tag{10}$$

Where $k=I_1a_1+I_2a_2$

$P_1, P_2, I_1, I_2, a_1a_2, C_1, C_2$-ditto

Version No 1 is giving index 1 and version No 2-index 2. According formula (10),we thus find that the cost difference between two versions is directly proportional to the difference of rock dilution,if the profits of two versions are equal.In other words,the profit can be increased by reducing rock dilution from d_1 to d_2 and at the same time increasing the mining cost from C_1 to C_2,i.e. the increment in costs is compensated by decreasing rock dilution.

Example. Consider a high grade narrow vein(B2-19 to B2-22, 360-420 levels) with thickness of orebody less than 3 m. The fiqures might be:

I_1=9.79 Kr/t;

I_2=4.23 Kr/t;

r_1=0.87; r_2=0.83;a_1=2.8%,a_2=1.47%;

C_1=140 Kr/t;

d_1=0.5(thickness of vein is 2.5 m, stoping width is 5 m for aviable equipment).

Substituting the data presented above,profit will be:

$P=(9.79*0.87*0.028+4.23*0.83*0.0147)(1-d)-C=290(1-d)-C.$

How will dilution decreased by increasing in costs in order to maintain the constant profit level.

When dilution decreases from 0.5 to o.26,the cost can be increased by 50% from 140 Kr/t to 210 Kr/t,with unchanged profit,see line(1) and (2) from point A to B in fig.12.

Profit calculation for two versions are shown in table 20.

Table 20 Profit calculation

Version	Cost,Kr/t				Expected dilution	P
	Mining	Mill	indirect	Sum	%	Kr/t
No 1	51	36	53	140	50	5
No 2	102(*)	36	53	191	25	26

*-asumed cost is twice of version No 1.

Table 20 shows that in case of double mining cost by using mining with less dilution the prifit increases to a great extent by lowering dilution.The stoping parameter for narrow vein orebody can be optimized by using the optimum model based on establishment of interdependencies between ore quality parameters.The method of stoping parameters optimization and equipment size selection is shown in App. 20.It should here be point out that the optimum stoping parameter can be obtained by maximizing profit for each specific cases.The influence of equipment size on dilution and mining cost can be examined using optimum model for narrow vein orebody with thickness of 0.5-5 m.

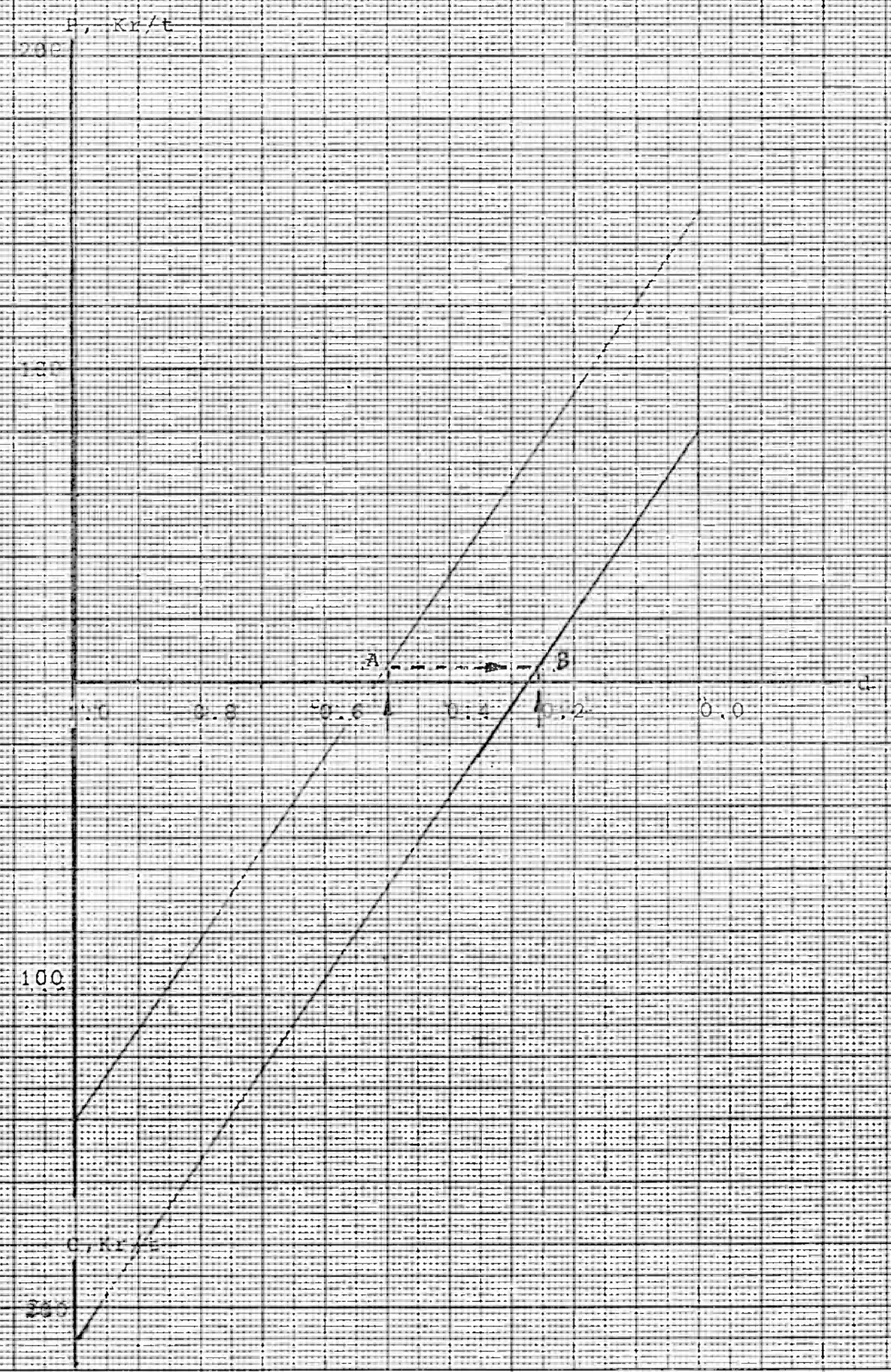


Fig.12

4.5.Prediction of dilution level and improvement in mining methods

Using numerical analysis method described in paragraph 2 dilution level of existing blocks can be predicted as shown in App.21 and table 21.It shows that the most blocks of X series have dilution level 10-15% or so and blocks of B2 series have average dilution of about 30%.It is observed that the tendency of increasing dilution is remarkeble.This situation is mainly caused by extraction of narrow veins and complicated orebody including existance of waste layer.

In order to control dilution for further production,the impro vements in mining methods can be suggested as follows(see fig. 13):

-Mining method with minimum dilution by means of separation of ore from waste or application of small equipment for narrow vein orebody with high grade;

-Selecting mining by dividing two-stage to be blasted in LH sublevel stoping for mining of various ore types and separating waste layer;

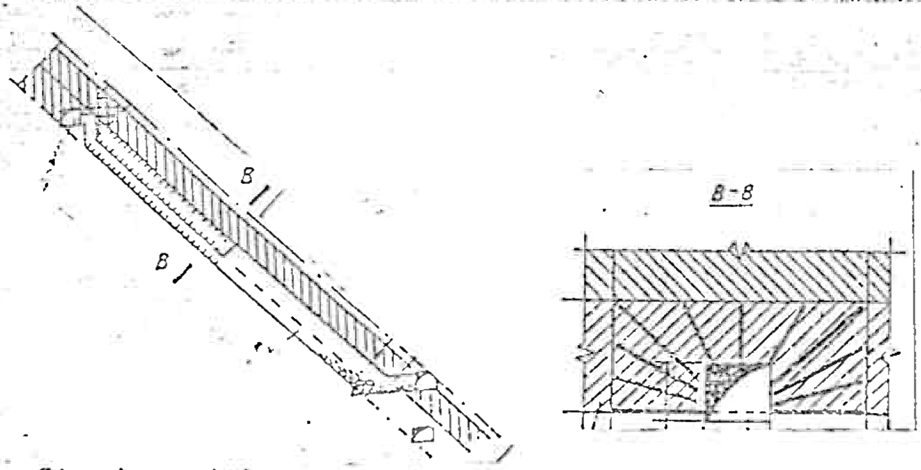
-Stoping with ore mucking by means of blasting forces for flat-dipping orebody;

-Pillar mining in contact with filling massive for complicated rock mechanics conditions;

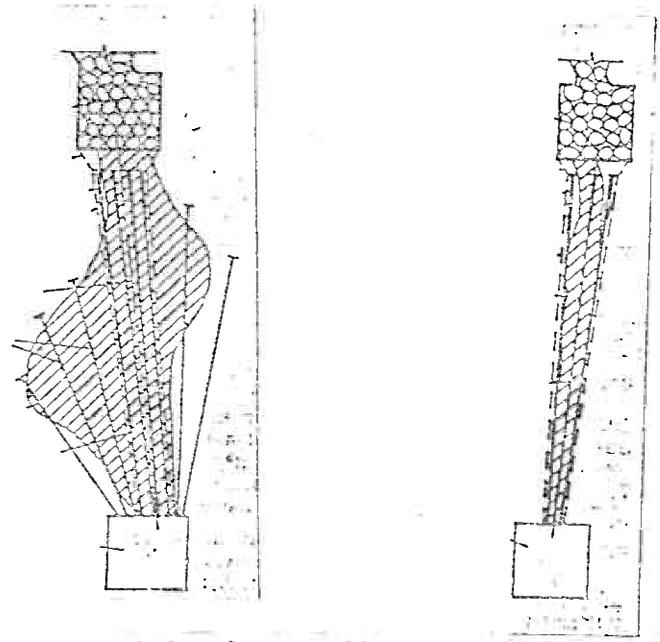
-Stoping with closer distance between holes for changeable forms of orebody.

Table 21 Estimatiom of rock dilution at Grong Gruber

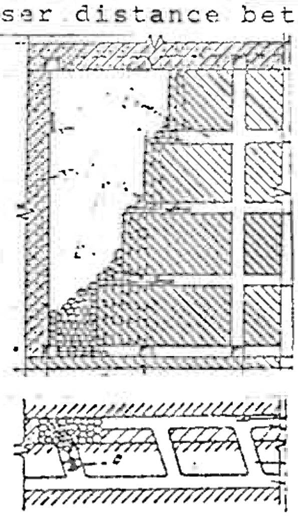
Block	Ore grade in situ,%		Rock dilution,%	Ore grade planned,%	
	Cu	Zn		Cu	Zn
2	0.68	1.91	9.4	0.62	1.73
3	0.29	2.80	10.0	0.26	2.52
4	0.44	3.02	9.3	0.40	2.74
5	0.56	1.61	10.2	0.50	1.45
6	0.81	2.97	11.0	0.72	2.64
7	1.71	2.03	13.3	1.48	1.76
8	0.82	0.72	13.0	0.71	0.62
9	1.07	2.23	10.7	0.96	2.55
10	0.59	2.85	8.0	0.54	1.80
11	0.78	1.96	16.0	0.66	1.65
12	1.67	1.50	5.0	1.59	1.43
13	2.01	1.77	14.9	1.71	1.51
14	0.93	0.26	30.0	0.56	0.16
15	0.91	1.40	9.3	0.83	1.27
16	3.76	0.59	50.0	1.88	0.30
17	1.08	1.22	10.8	0.96	1.10
X Series			14.0		
21	1.42	1.38	10.6	1.27	1.23
22	1.11	2.70	22.0	0.87	2.11
23	2.16	0.15	19.0	1.75	0.12
24	1.39	1.83	12.0	1.22	1.61
25	1.83	0.66	45.0	1.01	0.36
26	1.95	2.36	42.0	1.13	1.37
27	1.74	1.12	41.0	1.03	0.66
B2 SEries			31.0		
81	1.79	0.80	17.0	1.49	0.66
82	1.24	1.98	16.6	1.03	1.65
B1 Series			17.0		
51	3.05	1.56	22.0	2.38	1.22
52	1.32	1.28	18.5	1.08	1.04
53	1.36	0.67	47.0	0.85	0.36
54	2.40	1.52	30.0	1.68	1.06
55	0.79	2.55	17.0	0.66	2.12
56	1.70	0.33	23.0	1.31	0.25
Y Series			23.0		



Stoping with ore mucking by means of blasting forces



Stoping with closer distance between holes



Pillar mining in contact with filling massive

Fig.13 Scheme of variants of mining method

Conclusion

The rock dilution in Grong Gruber for long time period from starting up to now averages about 10-15% and has tendency of increase in it due to complexity of mining conditions in the coming years. The major nature factors influencing the rock dilution in the mine are irregularity and unevenness of the orebody, which influence on dilution level to considerable extent(5-20% and more). Economical analysis shows that decreasing rock dilution and increasing ore grade mined are of decisive importance for improving the economy at Grong Gruber. Reduction of dilution of 1% leads to improve the income by 1-1.5%, that is about 2 Kr/t under existing circumstances of the mine. Numerical analysis also shows that it could be possible to decrease the dilution by 5% in planning stage using optimum model of dilution for complicated conditions of Joma orebody.

The suggested optimum stoping design, modelling with grade control, dilution control for narrow vein orebody, improvement in mining methods and application of stoping technology with limited dilution are realistic approches to control rock dilution and to get better economy during further production at Grong Gruber.

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2. Song, X.:"Rock dilution control at Viscaria Mine", Outokumpu OY, Mining Service,Internal report,Outokumpu,June 1989.
3. Nielsen,K.:"Computers in mining", Proc. The Norwegian Mineral Industry Society.Fall Meeting,Stavanger,Sept.,1984.
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APPENDICES

Kopi E. RIAN

App.1-1

Grong Gruber A/S

JOMAFOREKOMSTEN

historisk oversikt:

	merkn.	mill. tonn råmalm	in situ gehalter % Cu	in situ gehalter % Zn	produserbare gehalter % Cu	produserbare gehalter % Zn
Sum forekomst 1972	1)	16,424	1,27	1,70		
Utbrutt malm:						
produsert i perioden sept. 72 - mai 87	2)	5,7	(1,79)	(1,42)	1,56	1,24
Tilleggsmalm:						
påvist i perioden sept. 72 - mai 87	3)	4,5	1,40	1,58		
Sum forekomst mai 1987	4)	15,224	1,13	1,79	(0,98)	(1,56)

App.1-2

MALMBEREGNING 1986

IN SITU TONNASJE

Område	Tonn	%Cu – %Zn – %S
B1	1060438	1.30 – 1.38 – 36.0
B2	2217259	1.65 – 1.14 – 32.4
X	11076338	0.93 – 2.01 – 40.0
Y	621643	1.85 – 1.50 – 29.5
Totalt	14975678	1.10 – 1.81 – 38.1

01.11.86 LBL

App. 1-3

GRONG GRUBER A/S

UTDREVET MALM

År	Cu	Zn	Tonn
1972	1,43	0,96	76.000
1973	1,61	0,87	298.000
1974	1,56	0,99	288.000
1975	1,95	1,14	340.000
1976	2,06	0,97	341.000
1977	1,52	1,16	352.000
1978	1,54	1,15	360.000
1979	1,37	1,07	368.000
1980	1,62	1,00	396.000
1981	1,60	1,03	402.000
1982	1,57	1,14	425.000
1983	1,48	1,15	483.000
1984	1,38	1,42	396.000
1985	1,30	1,51	443.000
1986	1,53	1,47	489 000
sum	1,56	1,16	5 457 000

MALMBEREGNING 1985

IN SITU TONNASJE

Sted	Tonn	Cu–Zn	Tonn Cu	Tonn Zn
B1	1060438	1.30 – 1.38	13786	14634
B2	2338741	1.68 – 1.14	39291	26662
X	11726354	0.96 – 2.00	112573	234527
Y	535229	1.91 – 1.21	10223	6476
Totalt	15660762	1.12 – 1.80	175873	282299

29.05.85 LBL

GRONG GRUBER A/S

UTDREVET MALM

År	Cu	Zn	Tonn
1972	1,43	0,96	76.000
1973	1,61	0,87	298.000
1974	1,56	0,99	288.000
1975	1,95	1,14	340.000
1976	2,06	0,97	341.000
1977	1,52	1,16	352.000
1978	1,54	1,15	360.000
1979	1,37	1,07	368.000
1980	1,62	1,00	396.000
1981	1,60	1,03	402.000
1982	1,57	1,14	425.000
1983	1,48	1,15	483.000
1984	1,38	1,42	396.000
Sum	1,59	1,10	4.525.000

App.1-6

(1)Ore reserves in planning for 1982-1985

Year	Ore reserves(plan) Tonnage,kt	Cu grade %	Ore mined Tonnage kt	Cu grade %	rock dilution,
1982	400	1.67	425	1.58	5.4
1983	480	1.56	483	1.48	5.1
1984	480	1.45	396	1.38	4.8
1985	484	1.34	443	1.30	3.0
1982-1985	1844	1.50	1747	1.44	4.0

(2)Ore reserves of major blocks for 1972-1984

Major production blocks	Ore reserves Tonnage,kt	Cu grade,%
10,11,12,21,22,23,24,25,51,52, 53,54,55,56,61,81,82.	5670	1.83

(3)Ore mined for 1972-1984

Year	Ore mined Tonnage,kt	Cu grade,%
1972-1984	4525	1.59

App.1-7

	Ore reserves in situ		Ore to be mined		d,%
	Tonnage,kt	Cu grade,%	Tonnage,kt	Cu grade,%	
X series					
A	1011	1.61	1011	1.34	17
B	1775	1.50	1775	1.23	18
C	604	1.26	98	1.84	-46
D	132	3.20	92	2.72	15
Sum	3522	1.55	2976	1.33	14
Y series					
B	174	2.02	174	1.60	21
C	157	2.38	99	2.13	11
D	61	2.20	38	1.85	16
Sum	392	2.19	311	1.80	18

App. 2.

... ORE QUALITY AND ITS DISTRIBUTION

1. Difination

In practice,rock dilution is a mining parameter useful for

evaluating the ore quality and efficiency of a giving mine situation.According to different mining situations rock dilution is defined as follows:

(1) Rock dilution is defined as weight percentage

$$d_1=q/Q \qquad d_2=q/Q' \tag{1}$$

This is a direct method used in cases where it is possible to directly measure the weight.

(2) Rock dilution is defined as grade percentage.

$$d_1=a_0-a/a_0 \qquad d_2=a_0-a/a \tag{2}$$

It is a inderect method for estimating rock dilution emploed in most mining situations.

(3) Rock dilution is defined as valume or thickness percentage.

$$d_1=m-m_0/m \qquad d_2=m-m_0/m_0 \tag{3}$$

It is also a direct method for estimating rock dilution used in in situation of narrow vein mining.

Where:

d_1,d_2- Rock dilutiom,%,

q- Tonnage of waste rock,t,

Q- Tonnage of ore mined, i.e. ore+rock,t,

a_0- Ore grade of the mineable ore reserves,%,

a- Grade of ore mined,%,

m- Stoping thickness,m,

m_0- Thickness of narrow vein,m.

Q'-Real ore present in Q,t.

It can be shown that formulas (1) and (2) are equivalent.

$$d=a_0-a/a_0=a_0-Q'a_0/Q/a_0=1-Q'/Q=q/Q$$

?. ORE QUALITY CONTROL PARAMETERS

The most widely used ore quality control parameters are metal recovery, ore recovery, rock dilution and ore quality Index useful for evaluating the economy and efficiency of mine operations. These parameters are defined as follows:

$$M = Q.a/Q_0.a_0 \quad \ldots (4)$$

$$R = Q/Q_0 \quad \ldots (5)$$

$$d = a_0 - a / a_0 \quad \text{or} \quad d = a_0 - a / a_0 - a' \quad \ldots (6)$$

$$I = a/a_0 \quad \ldots (7)$$

Where:
M-Metal recovery,%,
R-Ore recovery,%,
d-Rock dilution,%,
I-Ore quality Index,%,
Q-Tonnage of ore mined,t,
Q_0-Tonnage of ore reserves in situ,t,
a-Average grade of ore mined,%,
a_0-Average grade of ore reserves in situ,%,
a'-Average grade of waste rock,%.

Ore quality index, I, equals 1-d, indicating the ratio between metal recovery and ore recovery, I=M/R, can be considered as a parameter reflecting the difference between in situ grade and mined grade.

As discussed above, the rock dilution is expressed by $d=a_0-a/a_0$, or $d=a_0-a/a$, thus we get:

$$I=1-d \quad (8)$$
$$I=1/1+d$$

In the situation when the grade of rock, a' is not zero, the rock dilution can be calculated from following formulas:

$$d=a_0-a/a_0-a'$$
$$d=a_0-a/a-a'. \quad (9)$$

In a situation when the grade of waste rock has to be taken into account, the ratio between ore quality index and rock dilution is expressed by formula:

$$I = 1-(1-a'/a_0).d. \quad (10)$$

App.3-1

Place	Ore grade($\bar{a}$),%		$(a-\bar{a})^2$	
	Cu	Zn	Cu	Zn
BSK 429,BSK 416 x=95120	1.16	1.87	6.32	13.06

Cu: $s^2=6.32/11=0.57$, $s=0.76$, $v=0.76/1.16=65\%$;

Zn: $s^2=13.06/11=1.19$, $s=1.09$, $v=1.09/1.87=58\%$.

BSK 429 BSK 416 x=95080-95160	1.15	1.86	0.85	0.48

Cu: $s^2=0.85/9=0.09$, $s=0.307$, $v=0.307/1.15=27\%$;

Zn: $s^2=0.48/9=0.05$, $s=0.23$, $v=0.23/1.86=12\%$.

BLOCK		in situ tonn og gehalter TONN	Cu%	Zn%	S%
1	LH	0	0.00	0.00	0.0
1	STROSS	0	0.00	0.00	0.0
1	ORT	0	0.00	0.00	0.0
1	UDEF.	256153	0.63	1.46	0.0
1	SUM	256153	0.63	1.46	0.0
2	LH	0	0.00	0.00	0.0
2	STROSS	0	0.00	0.00	0.0
2	ORT	0	0.00	0.00	0.0
2	UDEF.	761344	0.85	1.32	0.0
2	SUM	761344	0.85	1.32	0.0
3	LH	0	0.00	0.00	0.0
3	STROSS	0	0.00	0.00	0.0
3	ORT	0	0.00	0.00	0.0
3	UDEF.	286543	0.28	3.35	0.0
3	SUM	286543	0.28	3.35	0.0
4	LH	0	0.00	0.00	0.0
4	STROSS	0	0.00	0.00	0.0
4	ORT	0	0.00	0.00	0.0
4	UDEF.	2054797	0.53	2.02	0.0
4	SUM	2054797	0.53	2.02	0.0
5	LH	0	0.00	0.00	0.0
5	STROSS	0	0.00	0.00	0.0
5	ORT	0	0.00	0.00	0.0
5	UDEF.	2080006	0.64	2.34	0.0
5	SUM	2080006	0.64	2.34	0.0
6	LH	0	0.00	0.00	0.0
6	STROSS	0	0.00	0.00	0.0
6	ORT	0	0.00	0.00	0.0
6	UDEF.	622349	0.55	3.08	0.0
6	SUM	622349	0.55	3.08	0.0
7	LH	0	0.00	0.00	0.0
7	STROSS	0	0.00	0.00	0.0
7	ORT	0	0.00	0.00	0.0
7	UDEF.	644275	1.61	2.07	0.0
7	SUM	644275	1.61	2.07	0.0
8	LH	0	0.00	0.00	0.0
8	STROSS	0	0.00	0.00	0.0
8	ORT	0	0.00	0.00	0.0
8	UDEF.	253098	3.24	0.79	0.0
8	SUM	253098	3.24	0.79	0.0
9	LH	0	0.00	0.00	0.0
9	STROSS	0	0.00	0.00	0.0
9	ORT	0	0.00	0.00	0.0
9	UDEF.	598737	1.59	1.99	0.0
9	SUM	598737	1.59	1.99	0.0
10	LH	0	0.00	0.00	0.0
10	STROSS	0	0.00	0.00	0.0
10	ORT	0	0.00	0.00	0.0
10	UDEF.	741085	0.83	2.37	0.0
10	SUM	741085	0.83	2.37	0.0
11	LH	0	0.00	0.00	0.0
11	STROSS	0	0.00	0.00	0.0
11	ORT	0	0.00	0.00	0.0
11	UDEF.	388999	0.73	1.63	0.0
11	SUM	388999	0.73	1.63	0.0
12	LH	0	0.00	0.00	0.0
12	STROSS	0	0.00	0.00	0.0
12	ORT	0	0.00	0.00	0.0
12	UDEF.	354186	1.10	2.04	0.0
12	SUM	354186	1.10	2.04	0.0
13	LH	0	0.00	0.00	0.0
13	STROSS	0	0.00	0.00	0.0
13	ORT	0	0.00	0.00	0.0
13	UDEF.	148009	2.15	1.64	0.0
13	SUM	148009	2.15	1.64	0.0
14	LH	0	0.00	0.00	0.0
14	STROSS	0	0.00	0.00	0.0
14	ORT	0	0.00	0.00	0.0
14	UDEF.	96560	1.67	0.49	0.0
14	SUM	96560	1.67	0.49	0.0
15	LH	0	0.00	0.00	0.0
15	STROSS	0	0.00	0.00	0.0
15	ORT	0	0.00	0.00	0.0
15	UDEF.	401505	0.72	0.98	0.0
15	SUM	401505	0.72	0.98	0.0
16	LH	0	0.00	0.00	0.0
16	STROSS	0	0.00	0.00	0.0
16	ORT	0	0.00	0.00	0.0
16	UDEF.	139183	1.67	1.89	0.0
16	SUM	139183	1.67	1.89	0.0
17	LH	0	0.00	0.00	0.0
17	STROSS	0	0.00	0.00	0.0
17	ORT	0	0.00	0.00	0.0
17	UDEF.	375936	1.21	1.02	0.0
17	SUM	375936	1.21	1.02	0.0
21	LH	0	0.00	0.00	0.0
21	STROSS	0	0.00	0.00	0.0
21	ORT	0	0.00	0.00	0.0
21	UDEF.	558058	1.36	1.19	0.0
21	SUM	558058	1.36	1.19	0.0
22	LH	0	0.00	0.00	0.0
22	STROSS	0	0.00	0.00	0.0
22	ORT	0	0.00	0.00	0.0
22	UDEF.	141087	1.24	2.09	0.0
22	SUM	141087	1.24	2.09	0.0
23	LH	0	0.00	0.00	0.0
23	STROSS	0	0.00	0.00	0.0
23	ORT	0	0.00	0.00	0.0
23	UDEF.	264032	1.69	1.28	0.0
23	SUM	264032	1.69	1.28	0.0
24	LH	0	0.00	0.00	0.0
24	STROSS	0	0.00	0.00	0.0
24	ORT	0	0.00	0.00	0.0
24	UDEF.	433924	1.47	1.41	0.0
24	SUM	433924	1.47	1.41	0.0
25	LH	0	0.00	0.00	0.0
25	STROSS	0	0.00	0.00	0.0
25	ORT	0	0.00	0.00	0.0
25	UDEF.	388435	2.22	0.75	0.0
25	SUM	388435	2.22	0.75	0.0
26	LH	0	0.00	0.00	0.0
26	STROSS	0	0.00	0.00	0.0
26	ORT	0	0.00	0.00	0.0
26	UDEF.	303858	2.45	1.59	0.0
26	SUM	303858	2.45	1.59	0.0
27	LH	0	0.00	0.00	0.0
27	STROSS	0	0.00	0.00	0.0
27	ORT	0	0.00	0.00	0.0
27	UDEF.	388980	1.97	1.05	0.0
27	SUM	388980	1.97	1.05	0.0
51	LH	0	0.00	0.00	0.0
51	STROSS	0	0.00	0.00	0.0
51	ORT	0	0.00	0.00	0.0
51	UDEF.	191922	1.87	1.88	0.0
51	SUM	191922	1.87	1.88	0.0
52	LH	0	0.00	0.00	0.0
52	STROSS	0	0.00	0.00	0.0
52	ORT	0	0.00	0.00	0.0
52	UDEF.	171257	2.01	1.36	0.0
52	SUM	171257	2.01	1.36	0.0
53	LH	0	0.00	0.00	0.0
53	STROSS	0	0.00	0.00	0.0
53	ORT	0	0.00	0.00	0.0
53	UDEF.	55599	1.84	1.38	0.0
53	SUM	55599	1.84	1.38	0.0
54	LH	0	0.00	0.00	0.0
54	STROSS	0	0.00	0.00	0.0
54	ORT	0	0.00	0.00	0.0
54	UDEF.	87445	1.66	1.16	0.0
54	SUM	87445	1.66	1.16	0.0
55	LH	0	0.00	0.00	0.0
55	STROSS	0	0.00	0.00	0.0
55	ORT	0	0.00	0.00	0.0
55	UDEF.	83648	1.11	1.97	0.0
55	SUM	83648	1.11	1.97	0.0
56	LH	0	0.00	0.00	0.0
56	STROSS	0	0.00	0.00	0.0
56	ORT	0	0.00	0.00	0.0
56	UDEF.	48468	2.49	0.39	0.0
56	SUM	48468	2.49	0.39	0.0
61	LH	0	0.00	0.00	0.0
61	STROSS	0	0.00	0.00	0.0
61	ORT	0	0.00	0.00	0.0
61	UDEF.	610000	2.10	0.80	0.0
61	SUM	610000	2.10	0.80	0.0
81	LH	0	0.00	0.00	0.0
81	STROSS	0	0.00	0.00	0.0
81	ORT	0	0.00	0.00	0.0
81	UDEF.	651788	1.28	1.68	0.0
81	SUM	651788	1.28	1.68	0.0
82	LH	0	0.00	0.00	0.0
82	STROSS	0	0.00	0.00	0.0
82	ORT	0	0.00	0.00	0.0
82	UDEF.	495999	1.45	0.98	0.0
82	SUM	495999	1.45	0.98	0.0
SUM		15077265	1.17	1.76	0.0

App.3-1

Coeff. variaton:

Cu- $s^2=18.26/32=0.57$

$s=0.76$

$v=0.76/1.17=65\%$

Zn- $s^2=15.76/32=0.49$

$s=0.70$

$v=0.70/1.76=40\%$

App.3-2

BORHULL NR.1028   FIL NR. 80

X= 95139.4   Y= 31594.8   Z= 434.92   RETN.=+35/300

	LENGDE		M	%Cu	%Zn	Sp.v.	V.M.	Cu.u.	Zn.u.
1	0.00 -	2.00	2.00	1.27	2.51	4.65	9.30	11.81	23.34
2	2.00 -	4.00	2.00	1.96	0.73	4.60	9.20	18.03	6.72
3	4.00 -	6.00	2.00	1.84	0.33	4.52	9.04	16.63	2.98
4	6.00 -	8.00	2.00	1.04	1.15	4.28	8.56	8.90	9.84
5	8.00 -	10.00	2.00	0.21	0.27	4.53	9.06	1.90	2.45
6	10.00 -	12.00	2.00	0.31	2.96	4.45	8.90	2.76	26.34
7	12.00 -	14.20	2.20	0.76	4.15	4.47	9.83	7.47	40.81
8	14.20 -	16.00	1.80	0.37	0.41	3.25	5.85	2.16	2.40
9	16.00 -	18.00	2.00	3.15	0.61	4.23	8.46	26.65	5.16
10	18.00 -	20.00	2.00	2.25	0.55	4.43	8.86	19.94	4.87
11	20.00 -	22.00	2.00	1.59	0.92	4.31	8.62	13.71	7.93
12	22.00 -	24.00	2.00	1.56	0.64	4.23	8.46	13.20	5.41
13	24.00 -	26.00	2.00	1.38	0.76	4.06	8.12	11.21	6.17
14	26.00 -	27.00	1.00	5.00	0.89	4.40	4.40	22.00	3.92
15	27.00 -	28.52	1.52	5.05	0.24	4.33	6.58	33.24	1.58
SUM			28.52				123.25	209.61	149.93
GJ.SN.				1.70	1.22	4.32			
16	33.60 -	35.00	1.40	0.21	3.39	4.21	5.89	1.24	19.98
17	35.00 -	37.00	2.00	0.25	3.99	4.35	8.70	2.18	34.71
18	37.00 -	38.25	1.25	0.51	3.74	4.03	5.04	2.57	18.84
SUM			4.65				19.63	5.98	73.53
GJ.SN.				0.30	3.75	4.22			

Cu: $s^2=32.17/14=2.30, s=1.52, v=1.52/1.70=89\%$;

Zn: $s^2=21.43/14=1.53, s=1.23, v=1.23/1.22=100\%$.

BORHULL NR.1029 FIL NR. 81

X= 95139.6 Y= 31594.8 Z= 433.86 RETN.=+18/300

	LENGDE		M	%Cu	%Zn	Sp.v.	V.M.	Cu.u.	Zn.u.
1	0.00 -	2.00	2.00	0.46	4.47	4.60	9.20	4.23	41.12
2	2.00 -	4.00	2.00	1.98	1.50	4.58	9.16	18.14	13.74
3	4.00 -	6.00	2.00	1.66	0.35	4.58	9.16	15.21	3.21
4	6.00 -	8.00	2.00	2.31	0.36	4.53	9.06	20.93	3.26
5	8.00 -	9.57	1.57	1.46	0.90	4.49	7.05	10.29	6.34
6	9.57 -	10.80	1.23	0.00	0.00	2.70	3.32	0.00	0.00
7	10.80 -	12.00	1.20	0.54	2.27	4.39	5.27	2.84	11.96
8	12.00 -	14.00	2.00	0.41	2.55	4.53	9.06	3.71	23.10
9	14.00 -	16.00	2.00	0.30	5.11	4.68	9.36	2.81	47.83
10	16.00 -	18.00	2.00	0.37	2.64	4.70	9.40	3.48	24.82
11	18.00 -	20.00	2.00	0.43	2.10	4.55	9.10	3.91	19.11
12	20.00 -	22.00	2.00	0.42	0.09	4.32	8.64	3.63	0.78
13	22.00 -	24.00	2.00	0.25	0.51	4.47	8.94	2.24	4.56
14	24.00 -	26.00	2.00	0.16	0.12	4.47	8.94	1.43	1.07
15	26.00 -	28.00	2.00	0.16	0.05	4.47	8.94	1.43	0.45
16	28.00 -	30.00	2.00	0.28	0.48	4.39	8.78	2.46	4.21
17	30.00 -	32.00	2.00	0.34	0.41	4.40	8.80	2.99	3.61
18	32.00 -	34.00	2.00	0.20	5.49	4.57	9.14	1.83	50.18
19	34.00 -	36.00	2.00	0.26	6.57	4.45	8.90	2.31	58.47
20	36.00 -	38.00	2.00	0.30	0.87	4.83	9.66	2.90	8.40
21	38.00 -	40.00	2.00	0.22	2.65	4.63	9.26	2.04	24.54
22	40.00 -	41.21	1.21	0.42	3.39	4.46	5.40	2.27	18.29
23	41.21 -	42.72	1.51	0.26	1.16	3.53	5.33	1.39	6.18
24	42.72 -	44.00	1.28	3.43	1.05	4.12	5.27	18.09	5.54
25	44.00 -	45.93	1.93	1.58	0.28	4.33	8.36	13.20	2.34
SUM			45.93				203.50	143.75	383.12
GJ.SN.				0.71	1.88	4.43			

Cu: s^2=17.28/24=0.72, s=0.85, v=119%;

Zn: s^2=3.69 s=1.92, v=1o2%.

App. 4.

DISTRIBUTION OF ORE QUALITY

The ore quality of deposits in situ in most cases is usually distributed uneven. The degree of unevenness of distribution can be evaluated by expression:

$$V = \frac{100 \cdot \sigma}{a}, \%$$

or $$V = 100 \sqrt{\frac{a_{max} + a_{min}}{a} - \frac{a_{max} \cdot a_{min}}{a^2} - 1}$$

Where:

V- Coefficient of variation;

σ-Standard deviation;

a, a_{max}, a_{min}- mean, max. and min. value respectively.

According to variation ratio the degree of unevenness of distribution is classified in following three ranks:

Ranking	Corresponding V values
1. Uniform distribution	$V < 20$ %;
2. Gradational distribution	$V = 20-100$ %;
3. Erratic distribution	$V > 100$ %.

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App. 5.

												Cu recov.	Zn recov.
MND	DATO	TONV	CUPG	ZNPG	CUKONSCU	CUKONSZN	ZNKONSCU	ZNKONSZN	AVGCU	AVGZN	CUUTV	ZNUTV
4	1	1772	0.95	2.66	24.52	3.08	0.20	51.80	0.15	0.35	84.50	84.05
4	2	1706	1.01	2.66	25.00	3.76	0.16	53.60	0.14	0.42	86.54	80.45
4	3	1725	1.06	2.50	23.48	3.68	0.20	53.20	0.14	0.43	87.10	78.32
4	4	1809	1.27	1.57	22.16	2.80	0.22	52.20	0.14	0.37	89.41	68.99
4	5	1838	1.50	1.54	21.28	2.40	0.22	51.40	0.16	0.20	89.92	78.26
4	6	1571	1.34	1.85	18.68	3.60	0.20	51.40	0.17	0.32	88.06	71.95
4	7	1804	1.05	2.34	21.20	3.04	0.21	50.40	0.20	0.48	81.69	75.78
4	8	1801	1.04	2.00	21.12	2.48	0.18	54.00	0.18	0.32	83.40	80.04
4	9	1818	1.00	1.83	22.12	2.76	0.16	54.20	0.16	0.36	84.61	75.81
4	10	1797	1.11	1.83	22.48	1.92	0.15	54.20	0.17	0.39	85.35	75.71
4	11	1784	1.26	1.65	23.00	1.68	0.18	54.60	0.19	0.34	85.65	76.06
4	12	1787	1.36	1.30	23.24	1.92	0.23	54.40	0.22	0.25	84.61	74.75
4	13	1836	1.16	1.76	22.96	2.96	0.18	53.60	0.17	0.27	85.96	78.42
4	14	1835	1.35	1.53	22.40	3.34	0.16	55.26	0.17	0.26	88.09	72.66
4	15	1814	1.40	1.29	22.51	2.31	0.20	56.40	0.21	0.29	85.81	69.52
4	16	1800	1.28	1.47	21.63	2.34	0.21	54.60	0.17	0.27	87.34	74.72
4	17	1797	1.27	1.62	20.70	3.10	0.17	54.60	0.15	0.26	88.80	74.76
4	18	1746	1.41	1.79	24.00	2.22	0.16	52.90	0.15	0.20	89.90	83.18
4	19	1820	1.86	1.45	22.44	2.70	0.16	53.40	0.19	0.33	90.58	65.37
4	20	1873	1.11	1.94	22.86	3.61	0.18	53.00	0.16	0.41	86.14	72.53
4	21	1243	0.86	2.15	21.92	3.68	0.19	53.00	0.17	0.34	80.78	79.77
4	22	1752	1.13	1.73	22.52	2.72	0.14	54.00	0.19	0.31	84.00	76.65
4	23	1676	1.31	1.87	22.56	2.00	0.21	53.20	0.21	0.27	84.76	81.42
4	24	1769	1.70	1.38	23.08	1.92	0.19	53.80	0.20	0.36	89.02	66.95
4	25	1629	1.24	1.83	21.12	1.68	0.27	53.60	0.25	0.27	80.75	82.01
4	26	1662	0.82	2.08	21.56	3.24	0.19	54.20	0.20	0.34	76.36	80.11
4	27	1647	0.93	1.85	21.68	3.52	0.17	53.80	0.16	0.29	83.38	78.50
4	28	1723	0.88	2.72	22.04	5.00	0.13	52.80	0.17	0.36	81.50	81.77
4	29	1717	0.86	2.66	23.12	4.28	0.15	54.20	0.18	0.30	79.84	84.75
4	30	1691	0.93	2.32	23.84	3.08	0.16	55.20	0.20	0.42	79.30	78.95

Cu: $s^2=2.91/29=0.10, s=0.32, v=0.32/1.18=27\%$;

Zn: $s^2=5.8/29=0.20, s=0.45, v=0.45/1.91=23\%$.

App.6

Place	No	Area	Tonnage (t)	Ore grade,% Cu	Zn
429 BSK1 x-95140,y-31600					
without waste	5	11	900	0.46	6.82
	6+6'	122	10492	1.73	2.66
	7	188	166940	0.28	3.00
	Sum	321	28086	0.83	3.00
Including waste	5	11	900	0.46	6.82
	5'	12	672	0	0
	6	110	9460	1.73	2.66
	7	188	16694	0.28	3.0
	7'	12	672	0	0
	Sum	333	28398	0.76	2.86

App.7

2/x/LH				
x-95100,	a	76928	1.39	2.05
	b	75752	1.41	2.08
x-95120	a	73743	1.01	1.42
	b	72231	1.03	1.45
x-95140	a	61614	0.89	1.26
	b	60998	0.90	1.27
x-95160	a	62815	0.98	1.79
	b	60463	1.02	1.86

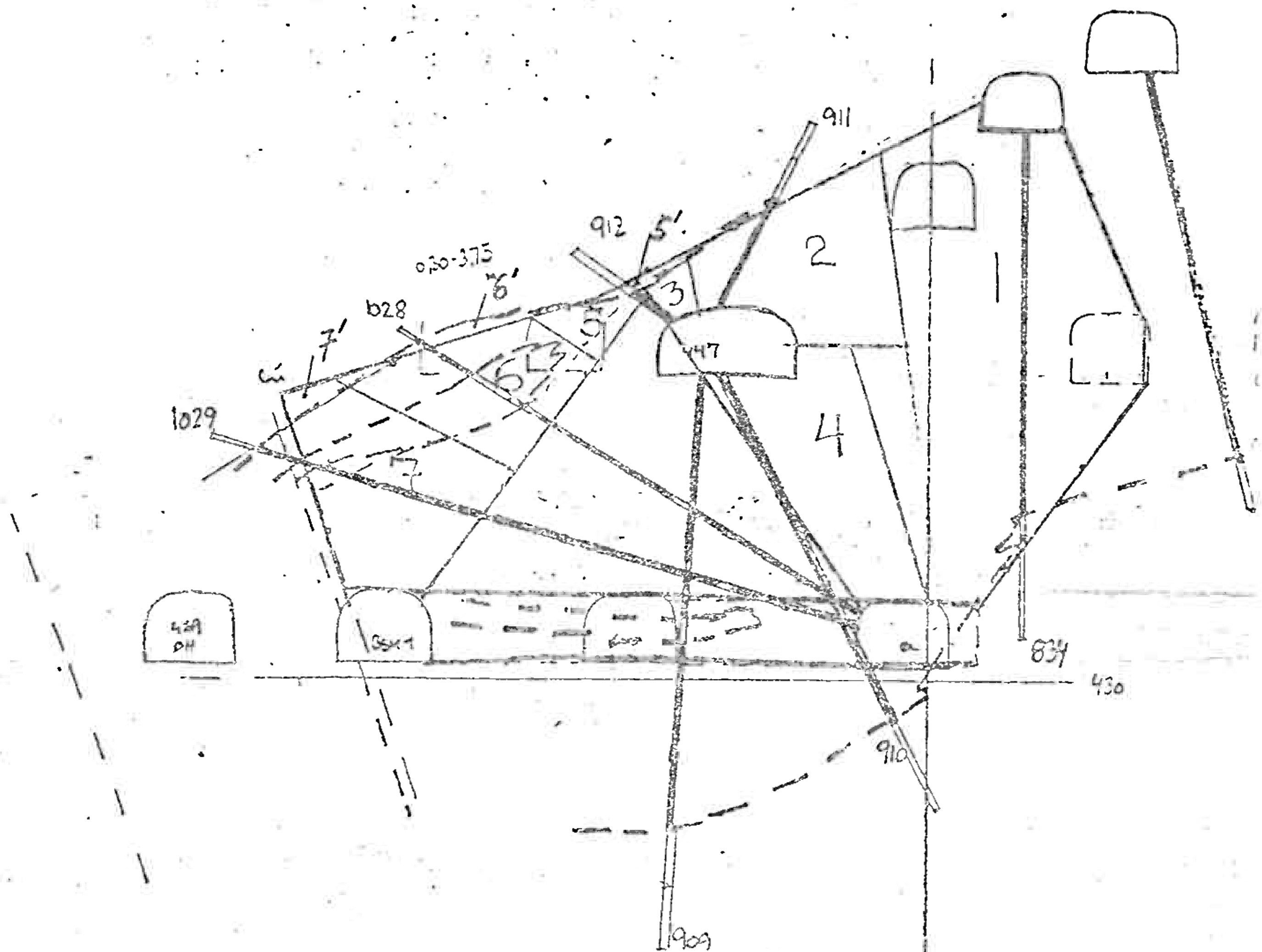
a-stoping boundary

b-boundary of orebody

App.8

x-95120	1-7	54341	1.25	1.80
	5'+6'	10304	0	0
	Sum	64645	1.05	1.52
x-95100	1-5	57112	1.80	1.75
	w	5600	0	0
	Sum	62712	1.64	1.59
x-95080	1-6	49213	1.52	1.74
	w	3920	0	0
	sum	53133	1.41	1.61

App. 6



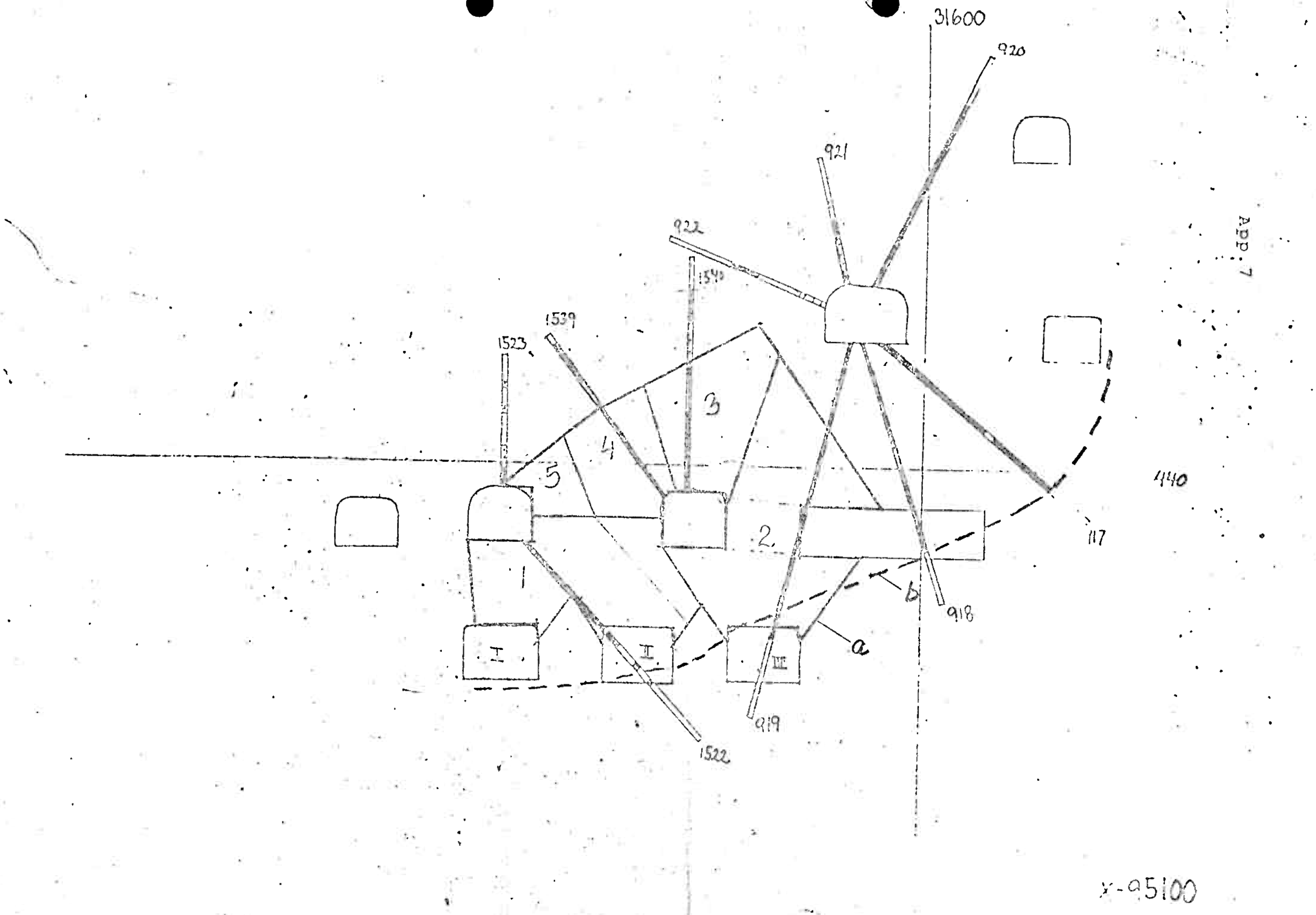
834
909 1,10-1,32-4,31
910 1,28-1,91-4,37
911 0,68-3,63-4,23
912 0,46-6,82-4,09
1028 1,70-1,22-4,32 /0,30-3,75-4,22
1029 0,71-1,88-4,43

31600

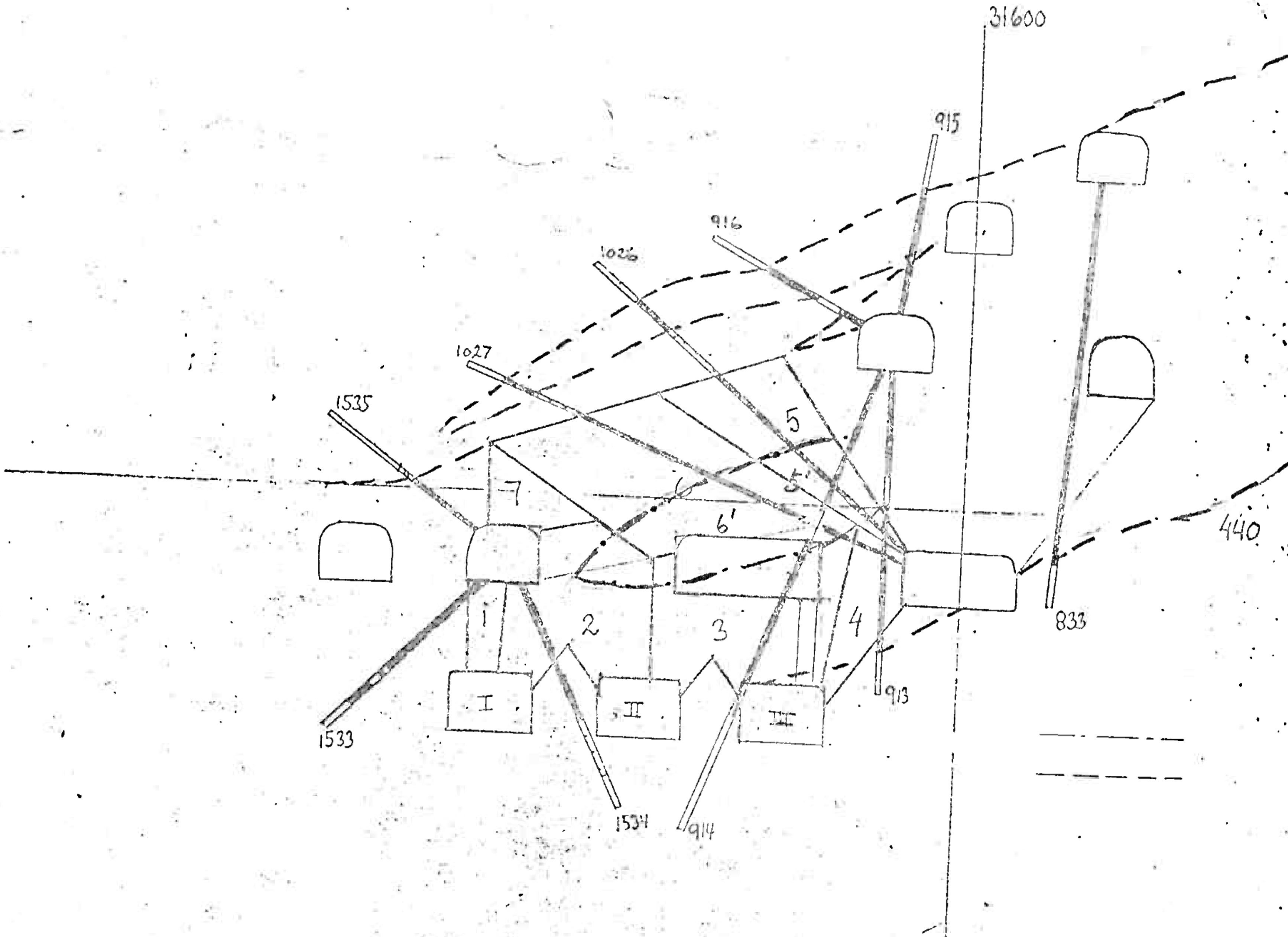
380

95140

APP. 7

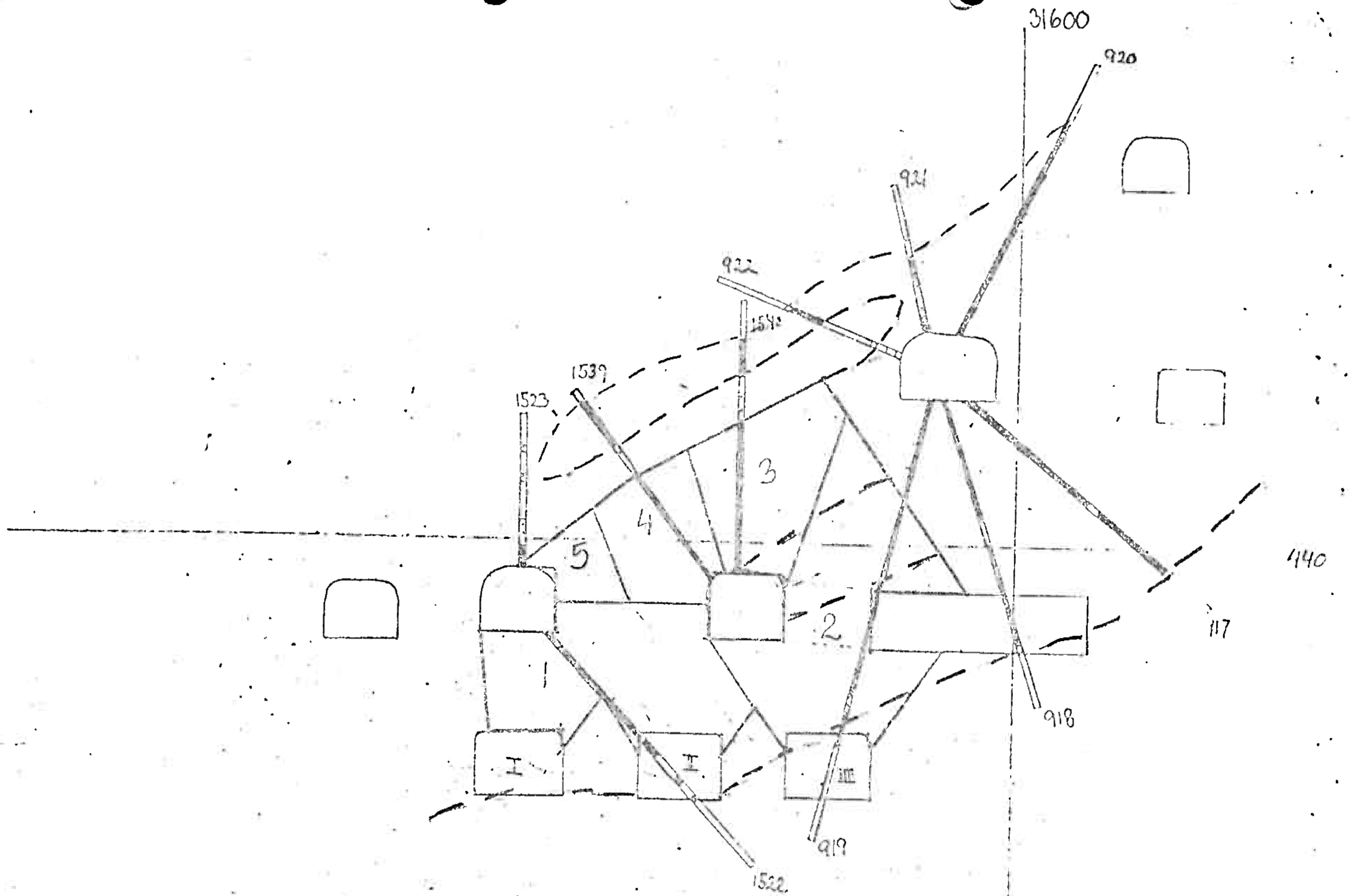

31600
920
921
922
1540
1539
1523
3
4
5
2
1
I
II
III
a
b
918
919
1522
117
440
x-95100

App. 8

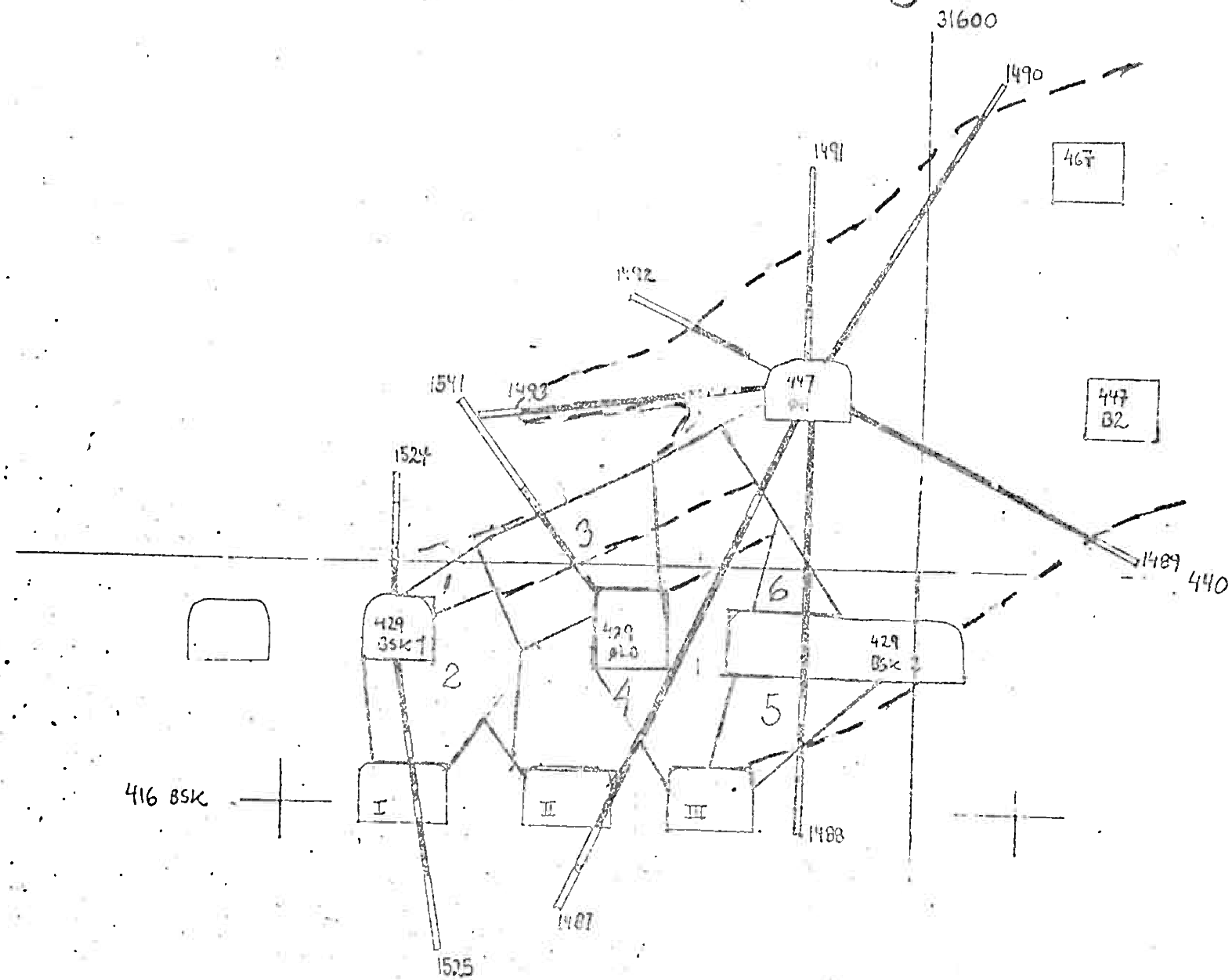


31600
915
916
1026
1027
1535
5
5'
6
6'
7
1
2
3
4
I
II
III
1533
1534
914
913
833
440

X-95120


31600
920
921
922
1541
1539
1523
3
4
5
2
1
440
117
918
919
1522
I
II
III

Y-95100



31600
1490
1491
467
1492
447
447
B2
1541
1493
1524
3
1489
440
6
429
BSK 1
2
429
PLO
429
BSK 2
4
5
416 BSK
I
II
III
1488
1487
1525

X-95080

App.9

Place		Tonnage,t	Ore grade;%	
			Cu	Zn
520,y-31760	2-4	12494	0.3	3.7
	w	1960	0	0
	Sum	14454	0.26	3.2

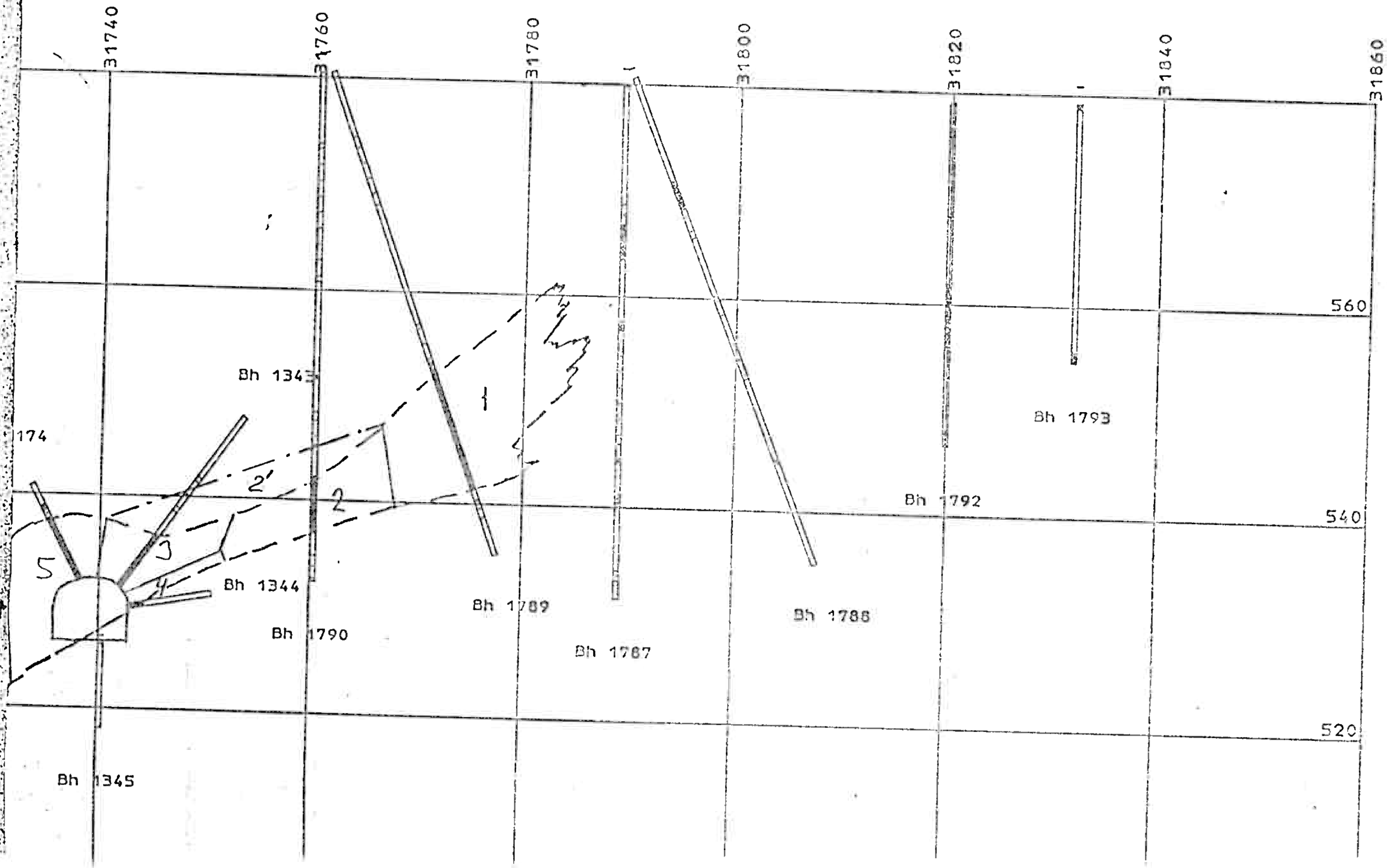
App.10

BSK429,x-95100			
Alt.No1. 7,8,9,10,11,12,13	76927	1.39	2.05
Alt.No2. 7,8,9,10,13	69003	1.54	2.17
Alt.No3. 7-13,14,15	92677	1.37	2.10

App.11

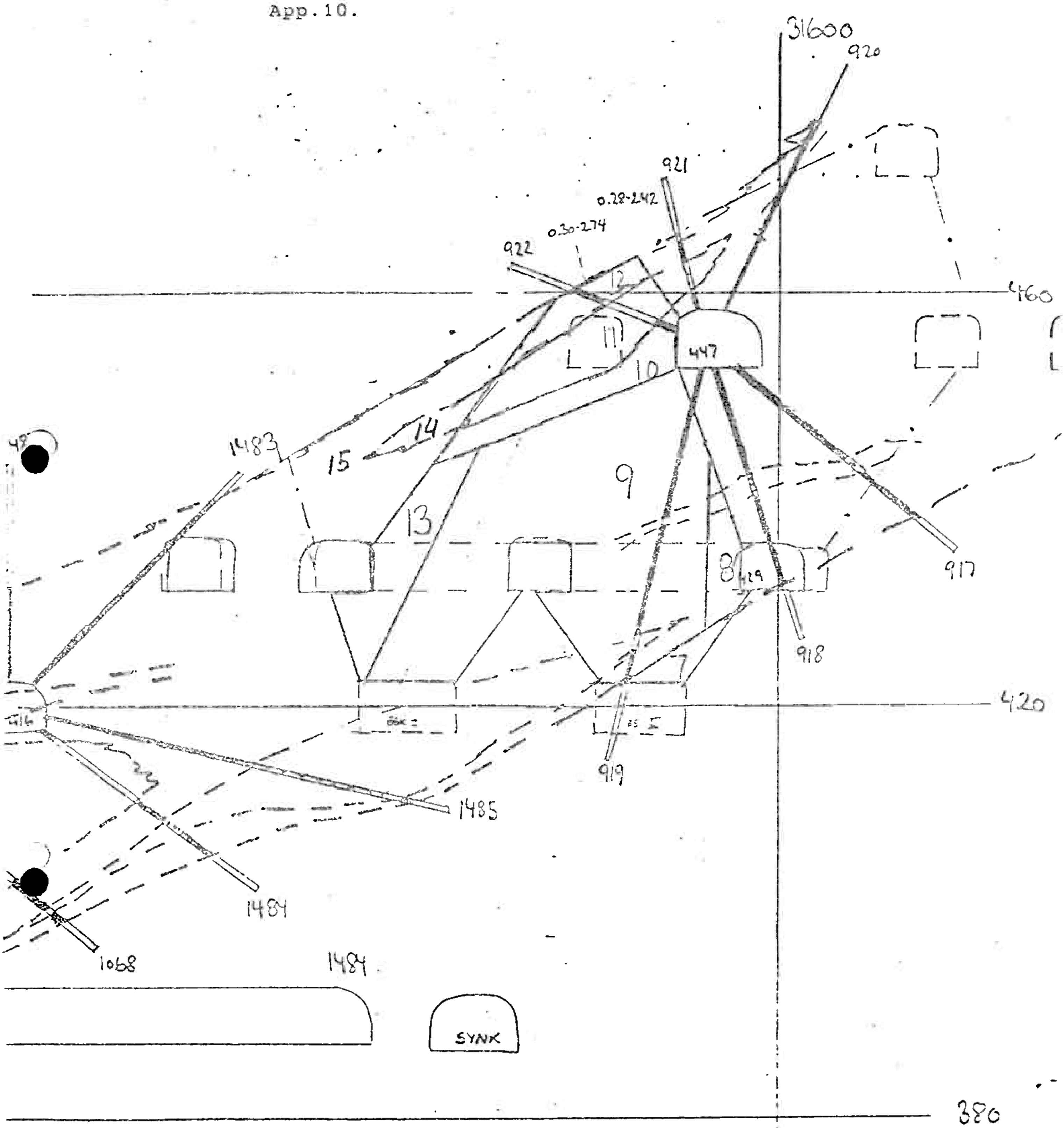
Alt.	Area			Ore grade mined%?	
	a'	a''	waste	Cu	Zn
Drill hole 1487 (a'=1.4-2.1%;a''=0.92-5.05)					
1	31.97	2.77	2.77	1.26	2.16
2	31.97	.0	.0	1.40	2.10
3	31.97	2.08	0.69	1.34	2.23
Drill hole 1488(a'=1.65-2.09%,a''=0.9-9.63%)					
1	25.35	1.29	1.29	1.54	2.34
2	25.35	0	0	1.65	2.09
3	25.35	1.04	0.25	1.61	2.37
Drill hole 1489(a'=0.76-1.73%,a''=0.97-3.40%)					
1	15	3.9	3.9	0.67	1.72
2	15	0	0	0.76	1.73
3	15	3.2	0.7	0.77	1.95

App. 9



31740
31760
31780
31800
31820
31840
31860
560
540
520
Bh 134
174
Bh 1344
Bh 1345
Bh 1790
Bh 1789
Bh 1787
Bh 1788
Bh 1792
Bh 1793
1
2
2'
3
4
5

App. 10.



917 0,79-2,49-4,27
918 0,84-2,42-4,27
919 1,68-2,30-4,36
920 0,50-2,88-4,08
921 0,28-2,42-3,62
922 1,10-0,24-3,24/0,30-2,74-3,85

X-95100

~~FORTROLIG~~ confidential

(12)

page 2/2

App.12. RG.

PLAN 1989 GRONG GRUBER A/S

MALMVERDI/PRODUKSJONSKOST

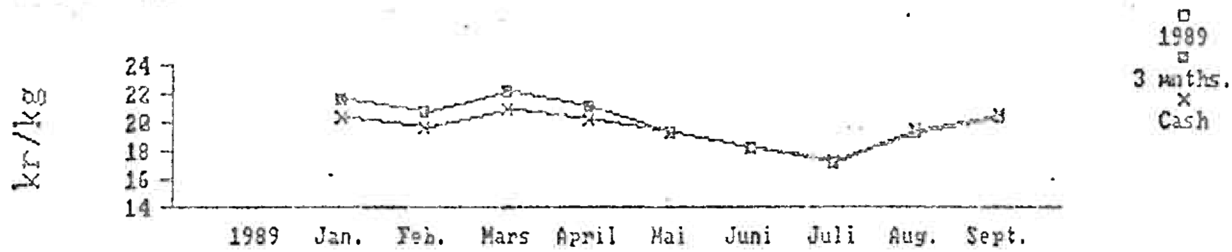
TALLENE KNYTTET OPP MOT REGNEARK
KONTRAKT BUDSJETT 89 OG SIR

		Kr/tonn	Kr/kg	%
CU	Metallinnhold i konsentrat		230	
	Bruttoverdi 1000 kg kons.	3094	13,45	100
	Fradrag smelteverk	855	3,72	27,65
	Cif-verdi Buy	2239	9,73	72,35
	Sir	252	1,10	8,16
	Verdi ab Joma Cu-innhold Value Content	1986	8,64	64,19
AG	Bruttoverdi	313	1,36	100
	Fradrag smelteverk	47	,20	15,00
	Cif-verdi	266	1,15	85,00
	Sir	0	0	0
	Verdi ab Joma Cu-innhold	266	1,15	85,00
CU + AG	Verdi ab Joma	2252	9,79	
ZN	Metallinnhold i konsentrat		450	
	Bruttoverdi 1000 kg kons.	3516	7,81	100
	Fradrag smelteverk	1410	3,13	40,12
	Cif-verdi	2105	4,68	59,88
	Sir	204	,45	5,80
	Verdi ab Joma Zn-innhold	1901	4,23	54,08

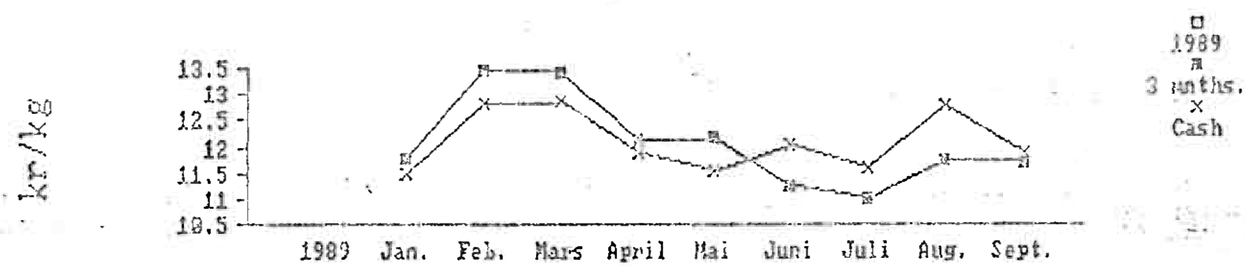
	RAMALM T	CU	ZN	AG gr.
PAGANG TONN	525000	1,40	1,73	
UTVINNING %		85,71	82,58	
~~MALM~~ TONN TONN METALL		6373	7500	
KONSENTRAT %		24	53	
KONSENTRAT mT		26555	14152	
AG gram pr. 1 tonn konsentrat				200
ANTALL KG AG				5311

MALMVERDI:

CU:	D33*D11/C31*1000 DVS. 6373*8,64/525000*1000	104,83
AG:	F37*D18/C31*1000 DVS. 5311*1.15/525000*1000	11,68
ZN:	E33*D28/C31*1000 DVS. 7500*4,23/525000*1000	60,36
MALMVERDI		176,87
TOTAL PRODUKSJONSKOST JOMA		138,69
FORTJENESTE PR. TONN RAMALM		38,18



Cu-pris pr.mnd. 1989



Zn-pr.mnd 1989

App.13.

. Optimization of ore grade mined in planning

As discussed in[1]that the boundary condition and unevenness of ore grade distribution influence on rock dilution.It should be possible to give a mathematical interpretation of relationship between mining boundary,rock/waste contact and ore grade distribution.Such an analysis would become a useful tool when estimating the optimum ore grade mined in planning stage.
From the geometry of rock/waste contact and ore grade distribution the ore grade in situ and ore grade mined for certain mining boundary([illegible]).The relationship can be derived, based on following expressions:

$$a(0)=\frac{m*h*a(1)+n*h*a(2)/2}{m*h+n*h/2}=\frac{m*a(1)+n*a(2)/2}{m+n/2} \qquad (4)$$

$$a=\frac{m*h*a(1)+\{l*h-h*l*l/2*n\}a(2)}{m*h+l*h}=\frac{m*a(1)+l*a(2)-l*l*a(2)/2*n}{m+l} \qquad (5)$$

and

$$a=\frac{m*a(1)+a(2)(w-m)-a(2)(w-m)^2/2\cdot n}{w} \qquad (6)$$

Where:

a(0)-Ore grade in situ,%;
a-Ore grade mined for giving mining boundary,%
w-Width of mining area,m.

The maximum ore grade mined with respect to mining boundary l may be obtained by equating the partial derivative,da/dl, of Eq.(5) to zero:

$$\frac{da}{dl}=\frac{(m*a(1)+l*a(2)-a(2)*l^2/2*n)-(m+l)(a(2)-a(2)*l/n)}{(m+l)^2}$$

$$\frac{da}{dl}=0$$

$$(m*a(1)+a(2)*l-a(2)*l^2/2*n)-(m+l)(a(2)-a(2)*l/n)=0$$

or

$$-a(2)*l^2/2*n-m*a(2)*l/n+m*(a(2)-a(1))=0$$

Solving for l.the optimum mining boundary l-opt.,is:

$$l\text{-opt.}=m*\left(\left(1+\frac{2*n}{m}\left(1-\frac{a(1)}{a(2)}\right)\right)^{1/2}-1\right) \qquad (7)$$

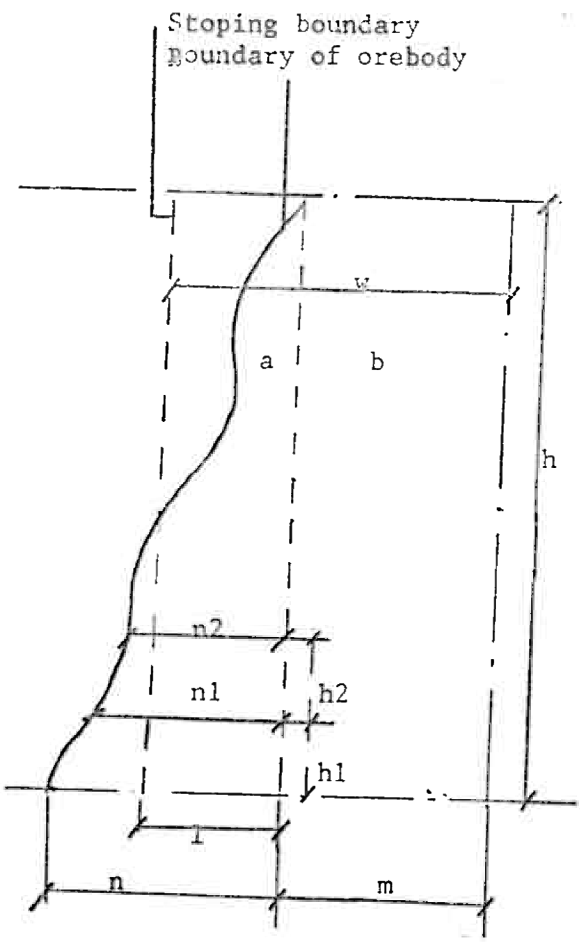
and the optimum width of mining boundary w-opt. will be:

$$w\text{-opt.} = m*\left(1+\frac{2*n}{m}\left(1-\frac{a(1)}{a(2)}\right)\right)^{1/2} \qquad (8)$$

Substituting Eq.(7) in Eq.(5), the maximum ore grade mined a-max., can be calculated from:

$$a\text{-max.} = a(2)*\left(1+\frac{1}{n/m}\right)-\left(a(2)-a(1)+\frac{a(2)}{2n/m}\right)*\frac{1}{\left(1+\frac{2n}{m}\left(1-\frac{a(1)}{a(2)}\right)\right)^{1/2}}$$

$$-\frac{a(2)}{2n/m}\left(1+\frac{2n}{m}\left(1-\frac{a(1)}{a(2)}\right)\right)^{1/2} \qquad (9)$$



$$\bar{n}=\sum_{1}^{n} ni * hi/ h$$

a-Area with high grade a(2),%
b-Area with low grade a(1),%

App.14

x-94920		
Stoping boundary	Tonnage,t	Zn grade,%
1	12612	1.08
2	16184	1.63
3	20660	2.51
4	24160	2.30

App.15

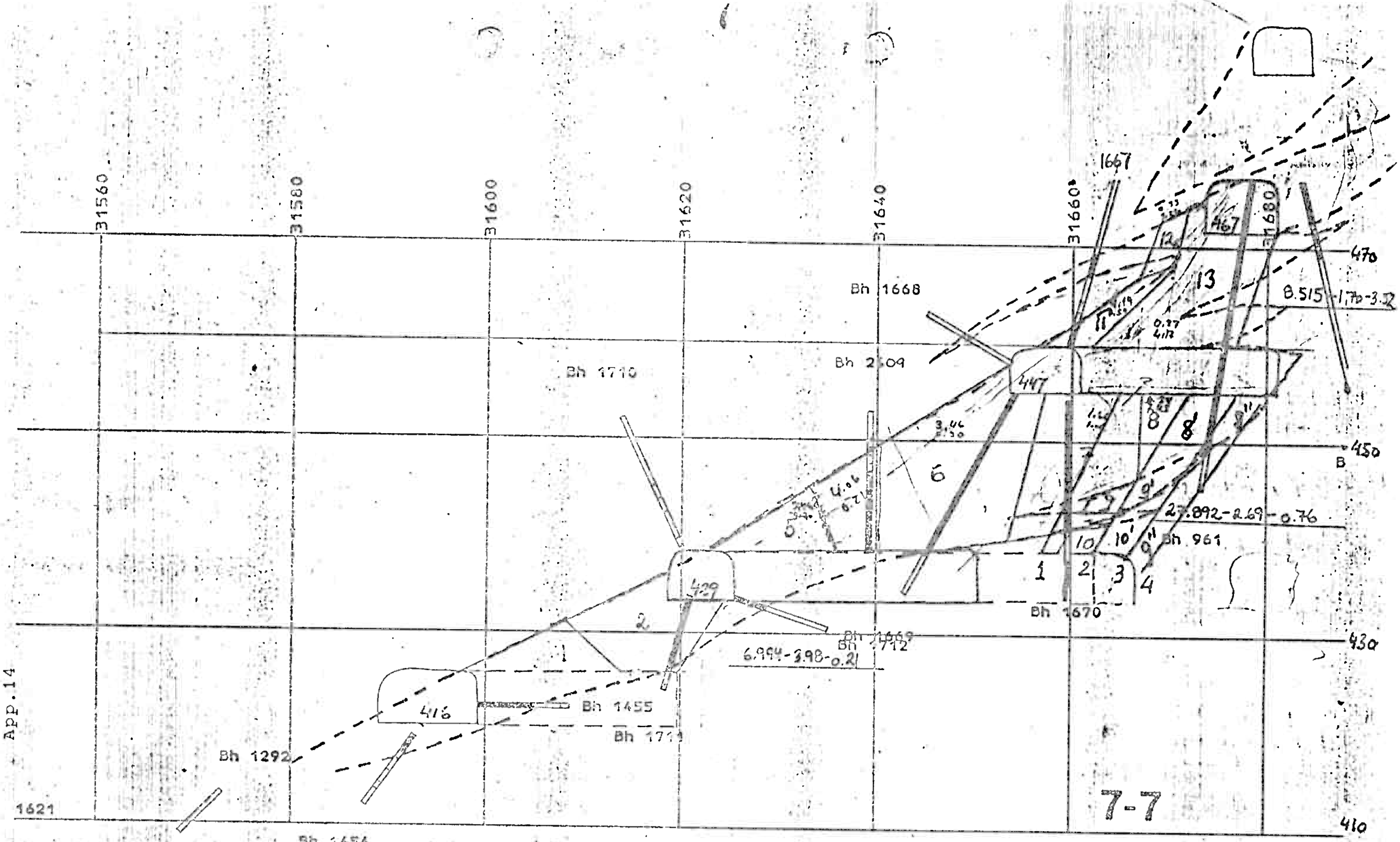
Year	Annual production,kt/y	Ore grade,%
1972-1980	308	1.64-1.04
1981-1988	459	1.47-1.40

Ore reserves(1972-1986)	Tonnage,Mt	Ore grade,%
Total	20.924	1.30-1.67
Reserves mined out	5.948	1.80-1.32
Remaining reserves	14.976	1.10-1.81

App.16

Place	Pillar		Area mined out				Number of block to be mined in condition	
	B	h	No filled		Filled			
			B	h	B	h	normal	complicated
B1-3	15	20	30	50			1	3
B1-4	20	20	25	55			1	3
B1-5	20	15	20	45			1	
B1-7	20	20	10	10	30	55	1	2
B1-8	8	8	30	20	30	35		3
B1-9	8	8	25	20	25	30		3
B2-2	20	10	50	10				2
B2-6	20	20	45	15	25	30	1	
B2-9	20	20	40	70	30	35		1
B2-11	10	15	70	15	30	35	1	1

App.14



VERTIKALPROFIL 1:5
X = 94920 16/

In examining the stability of room/stopes and pillars, There are three aspcts to be considered:

a) stress change that are caused by the excavation,

b/ deformation of the room´s roofs and pillars,

c/ parameters optimization of rooms and pillars.

Under certain geological conditions, stress distribution depends mainly on stopes geometry, stoping arrangement and sequence, and number of simultaneous working stopes.

stress distribution around, and stability of rooms and pillars are discussed using Displacement Discontinuity Method and its Computer programme (MINTAB).

MINTAB is a computer programme developed for analyzing pillar extraction scheme in a tabular orebody which is flat-lying or dipping at any angle up to 90^0 to the horizontal. The thickness t of the orebody is small compared to its lateral extent and the depth below the surface, Threfore, the orebody can be viewed as a simple plae crack of negligible thickness. The programme is based upon the Displacement Discontinuity Method (DDM). One of the basic assumptions in the present version of MINTAB is that pillars or intact portions of the unmined seam remain rigid. In other words they are in compressible as a result of load re-distribution resulting from mining. In reality, however, the pillars, remants or abutments are deformable and the deformation would be proportional,within the elastic limit,to the lead transferred onto them,Threfor,for mor realistic simulations in mining of tabular orebody,the compressibility of the seam should be taken into account.

Based on available information of the geology of the orebody and physico-mechanical properties of the host rocks,a mine area with dimensins of 120 m by 120 m has been taken for computer simulation. The mine area is diveded into 3 large squares in X-direction (strike) and 3 large squares in Y-direction (dip).Each large square is further divided into 5x5 small squares.The area consist of (3x5)x(3x5)=225 small squares,and the width of small square is 8 m.

A printer plot routine was designed and incorporaded to represent:stresses $\sigma_{zz}, \sigma_{xx}, \sigma_{yy}, \sigma_{xz}, \sigma_{zy}, \sigma_{xy}$;

displacements DX, DY, DZ;

released energy E and strength/stress ratio- SSR,which provides a general idea of stability for the mining conditions. A factor of safety less than 1.0 indicates an unstable condition where total stress is exceeded the mobilized strenth of pillars.

App.18

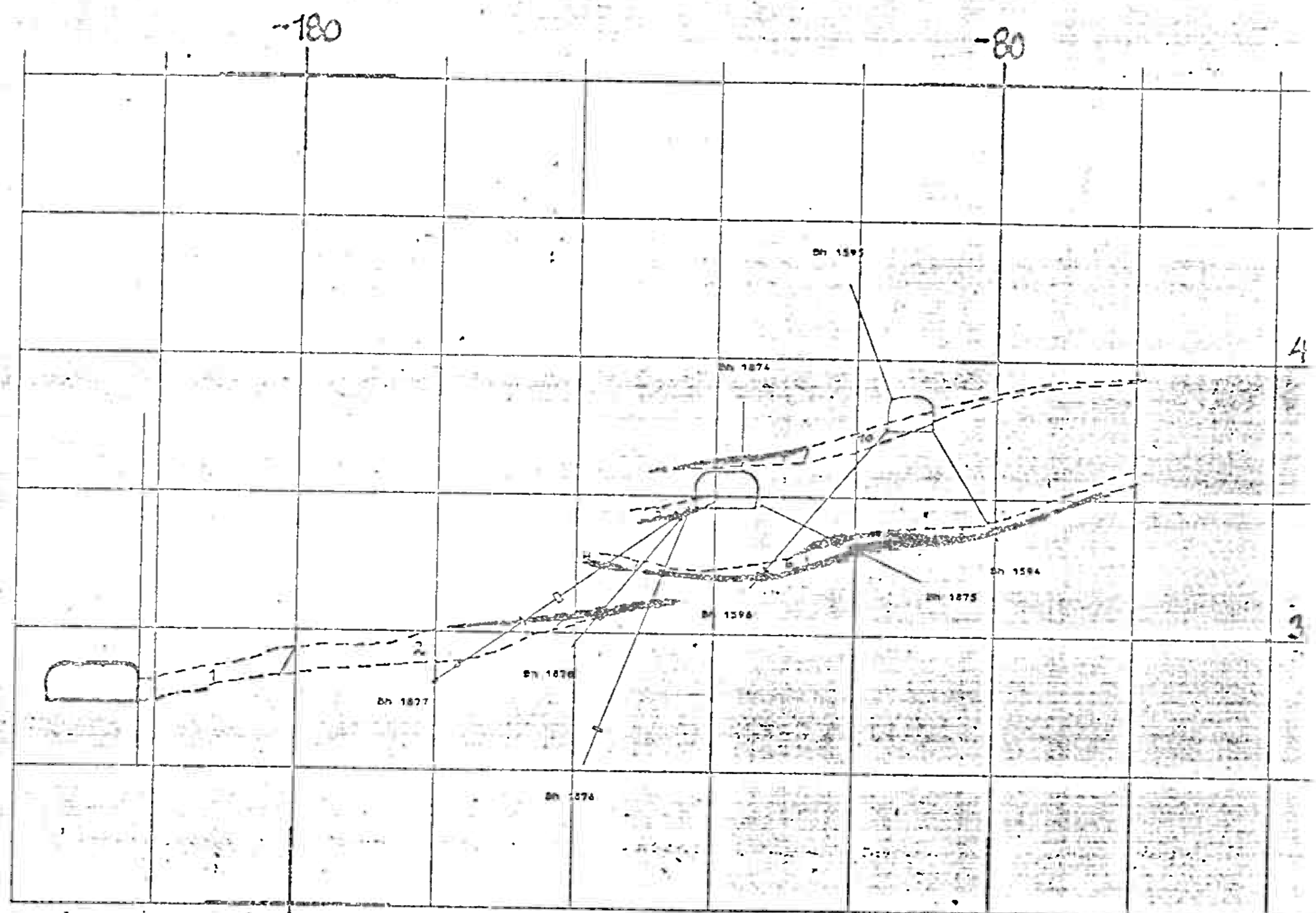
B2(360-420)

Profile	Length of orebody		Ore grade,%
	m>3	m<3	
B2-19	20	90	1.82-1.02
B2-20	0	200	1.87-1.14
B2-21	30	90	3.13-2.71
B2-22	0	140	4.38-1.01
y-31220	0	25	3.0-0.5
y-31240	0	55	5.84-0.52
y-31260	10	100	1-14-1.04
y-31280	30	150	1.35-1.33
y31300	90	80	1.11-1.77
y31320	50	90	1.53-1.35
y-31360	30	90	1.97-1.16
y-31380	50	100	2.05-1.57
y-31400	10	60	2.76-1.54

App.18

VERTIKALPROFIL B2-20

Zn = 1.0 – 2.0
Zn = 2.0 – 3.0
Zn > 3.0



Zn = 1.0 – 2.0
Zn = 2.0 – 3.0
Zn > 3.0

VERTIKALPROFIL
B2 = 20.

[illegible]. Ore quality optimization for narrow vein orebodies

1. Calculation formulas

In a general way formulas for defining rock dilution were introduced by Eustace Alaphia Wright (6).
Two basic methods are available for estimating rock dilution for narrow vein orebodies:
-Direct method

$$d_1=m'q/(m_0+m')q=m'/m_0+m' \qquad (14)$$

$$d_2=m'q/m_0q=m'/m_0 \qquad (15)$$

-Indirect method

$$d_1=a_0-a/a_0-a' \qquad (16)$$

$$d_2=a_0-a/a-a' \qquad (17)$$

Where:
m'-thickness of ripping waste rock,m,
m_0-thickness of ore veins,m,
q-productivity per meter thickness,t/m,
a_0,a' and a-Average grade of ore veins,of waste rock and of the mixture of ore and waste rock mined,respectively,% or g/t.
It is usually possible to directly measure the thicknesses m_0,m' and m during stoping of narrow veins orebodies.
It can be shown that formulas (14)and (16)are equivalent and that formulas (15) and (17)are also equivalent.
Formulas (14)and (15)can be converted using the transformation k,reflecting ratio of m to m_0.

$m/m_0=k$,where $k\geq 1$

Thus:

$$d_1=m'/m=1-1/k$$

$$d_2=m'/m_0=k-1$$

The corresponding values d_1 and d_2 are shown in table 4 and in fig. [illegible].

Table 4 Rock dilution of formulas (14) and (15)

k	$d_1=k-1/k$	$d_2=k-1$
1	0	0
1.2	0.17	0.20
1.4	0.29	0.40
1.6	0.38	0.60
1.8	0.44	0.80
2.0	0.50	1.00
2.5	0.60	1.50
3.5	0.71	2.50
5.0	0.80	4.00

3. Stoping thickness parameters optimization

In general stoping thickness exercises great influence on ore losses and rock dilution and mining cost for narrow veins orebody.Increasing stoping thickness will lead to increse rock dilution and to decrease ore loss. The connection between mining cost and the order of si: for rock dilution and ore losses will be discussed by using optimal model of ore quality in a specific case of narrow veins orebody.

. Adaptation of equipment size to thickness of orebody

The equipment size is one of major factors,relating to effect on rock dilution and mining cost,specially for narrow veins orebody.It is well known fact that as the thickness of orebody increases,the use of more equipment size is needed.The influence of equipment size on rock dilution and mining costs can be examined using optimum model of ore quality for mining of narrow veins orebody with thickness of 0.5-5.0 m.

Comparing two drill equipment,differing in mining cost and rock dilution,the total costs of both drill methods can be calculated by :

$C=C_c+C_m/1-d$

Where:

C-total cost,v/t,

C_c-Concentrate cost,v/t,C_c=7 v/t,

C_m-mining cost,v/t

and difference in total costs will be:

$C_1-C_2=C_c+C_{m1}/(1-d_1)-C_c+C_{m2}/(1-d_2)$

Small-diameter drill is giving index 1,and middle-diamete drill index 2.

Consider the curves $C_1=f_1(m_0)$and $C_2=f_2(m_0)$in fig.10, showing total costs of different drill methods for different thickness of orebody.The point of intersection of two curves will correspond the thickness of orebody of about 3 m,i.e. middle-diameter drill will be rational, when thickness of orebody $m \geq 3$m.

Estimatiom of rock dilution at Grong Gruber

Block	Ore grade in situ,%		Rock dilution,%	Ore grade planned,%	
	Cu	Zn		Cu	Zn
2	0.68	1.91	9.4	0.62	1.73
3	0.29	2.80	10.0	0.26	2.52
4	0.44	3.02	9.3	0.40	2.74
5	0.56	1.61	10.2	0.50	1.45
6	0.81	2.97	11.0	0.72	2.64
7	1.71	2.03	13.3	1.48	1.76
8	0.82	0.72	13.0	0.71	0.62
9	1.07	2.23	10.7	0.96	2.55
10	0.59	2.85	8.0	0.54	1.80
11	0.78	1.96	16.0	0.66	1.65
12	1.67	1.50	5.0	1.59	1.43
13	2.01	1.77	14.9	1.71	1.51
14	0.93	0.26	30.0	0.56	0.16
15	0.91	1.40	9.3	0.83	1.27
16	3.76	0.59	50.0	1.88	0.30
17	1.08	1.22	10.8	0.96	1.10
X Series			14.0		
21	1.42	1.38	10.6	1.27	1.23
22	1.11	2.70	22.0	0.87	2.11
23	2.16	0.15	19.0	1.75	0.12
24	1.39	1.83	12.0	1.22	1.61
25	1.83	0.66	45.0	1.01	0.36
26	1.95	2.36	42.0	1.13	1.37
27	1.74	1.12	41.0	1.03	0.66
B2 Series			31.0		
81	1.79	0.80	17.0	1.49	0.66
82	1.24	1.98	16.6	1.03	1.65
B1 Series			17.0		
51	3.05	1.56	22.0	2.38	1.22
52	1.32	1.28	18.5	1.08	1.04
53	1.36	0.67	47.0	0.85	0.36
54	2.40	1.52	30.0	1.68	1.06
55	0.79	2.55	17.0	0.66	2.12
56	1.70	0.33	23.0	1.31	0.25
Y Series			23.0		

App.21.

Block	Profile	No.	Area for different thickness(m) <3 m	5-20m	> 20 m	Dilution estimated by m	Irregularity H.W.	F.W.	W.L. %	Total %
1			2	3	4	5	6	7	8	9
2	x=94840	1		40		7				7
		2		68		7				7
		3		119		7				7
		4		178		7				7
		5		137		7		3		10
		6		138		7				7
		7		152		7		2		9
		8		56		7	5			12
		9	18			50				50
		Sum	906							9.4
3	x=94960	1	23			50				50
		2	48			50				50
		3		219		7				7
		4			169	5				5
		5			128	5				5
		7			60	5				5
		Sum	657							11
	x=95020	1		170		7		3		10
		2		83		7				7
		3		12		7				7
		4		5		7				7
		Sum	270							9
		Av.								10
4	x=95080	1		142		7		5		12
		2		39		7		5		12
		3		126		7		5		12
		4		256		7		5		12
		5		36		7		5		12
		6		150		7	8			15
		Sum	749							12.6
	x=95220	1			734	5		3		8
		2			470	5		3		8
		3			480	5		3		8
		4			99	5	3			8
		5			174	5		3		8
		Sum	1957							8
		Av.								9.3

1			2	3	4	5	6	7	8	9
5	x=95080	5		5		7		5		12
		6		15		7		5		12
		7		170		7		5		12
		8		106		7		5		12
		9		175		7		5		12
		10		110		7		5		12
		11		67		7		5		12
		12		76		7		5		12
		13		114		7		5		12
		14		180		7	5	5		17
		15		90		7	5	5		17
		16		167		7		5		12
		17			167	5				5
		18			67	5				5
		19			187	5				5
		20			25	5				5
		21			234	5				5
		Sum	1955							9.1
	x=95220	4			50	5		3		8
		5			113	5		3		8
		6			76	5		3		8
		7			216	5		3		8
		8			24	5		3		8
		9			242	5		3		8
		10			108	5		3		8
		11			134	5		3		8
		12			227	5		3		8
		13			128	5		3		8
		14			165	5		3		8
		15		190		7	5	5		17
		16		109		7		5		12
		17		246		7	5	5		17
		18		285		7		5		12
		19		270		7	5	5		17
		20		152		7		5		12
		Sum	2735							11.[illegible]
		Av.								10.2
6	x=94960	6			22	5				5
		7			18	5				5
		8			173	5				5
		9		77		7	5			12
		10		37		7	3			10
		11		50		7				7
		12		19		7				7
		13		30		7	3			10
		14		10		7	3			10
		15		65		7	6			13
		16		56		7	2			9
		17		105		7	2			9
		19		38		7				7
		Sum	700							8.3

1			2	3	4	5	6	7	8	9
	x=95020	3		41		7				7
		4		7		7				7
		5		91		7				7
		6		89		7				7
		7		155		7				7
		8		70		7				7
		9		9		7				7
		10		33		7		3		10
		11		43		7		3		10
		12		60		7		2		9
		13			309	5				5
		14			68	5				5
		15			287	5			15	20
		16			148	5			15	20
		Sum	1410							13
		Av.								11
7	x=94840	9	39			50				50
		10		47		7		5		12
		11		159		7		3		10
		12		76		7				7
		13		111		7				7
		14		66		7				7
		15		16		7		3		10
		16		88		7				7
		17		55		7				7
		18		74		7		5		12
		19		170		7	8			15
		20		40		7		5		12
		Sum	941							13.3
8	x=94840	21		23		7		3		10
		22		132		7		3		10
		23		47		7		5		12
		~~24~~		~~242~~		~~7~~	~~3~~	~~5~~		~~15~~
		Sum	444							13
9	x=94960	17		8		7			15	22
		18		17		7			15	22
		19		32		7			15	22
		20		28		7			15	22
		21		81		7				7
		22		27		7				7
		23		23		7			10	17
		24		39		7			10	17
		29		92		7			10	17
		30		33		7			10	17
		32		26		7			10	17
		33		35		7			10	17
		34		78		7			10	17
		Sum	517							16

	1		2	3	4	5	6	7	8	9
	x=95020	16			6	5				5
		17			13	5				5
		18			128	5				5
		19			143	5				5
		20			103	5				5
		21			36	5	5			10
		22			50	5				5
		26			50	5				5
		Sum	527							5.4
		Av.								10.7
10	x=95080	20			144	5	3			8
		21			323	5		3		8
		22		46		7	5			12
		23		75		7		5		12
		24		16		7	5			12
		25		145		7		5		12
		26			175	5				5
		27			200	5				5
		Sum	1124							8
11	x=95220	20		30		7		5		12
		21		394		7		5		12
		22		261		7		5		12
		23		261		7		5		12
		24			114	5			20	25
		25			62	5			20	25
		26			28	5			20	25
		27			23	5			20	25
		28			116	5			20	25
		29			79	5			20	25
		30			61	5			20	25
		Sum	1456							16
12	x=95080	26			18	5				5
		28			95	5				5
		29			64	5				5
		31			79	5				5
		33			129	5				5
		34			146	5				5
		35			346	5				5
		Sum	877							5
13	x=94960	35	23			50				50
		36		31		7			15	22
		37		33		7			15	22
		38		38		7			15	22
		39		12		7			15	22
		40		12		7			15	22
		Sum	149							34

1			2	3	4	5	6	7	8	9
	x=95020	26		67		7				7
		27		103		7				7
		29		31		7				7
		30		103		7				7
		31	17			50				50
		32		26		7				7
		33		37		7				7
		34		36		7				7
		35		32		7				7
		Sum	1077							8.6
		Av.								14.9
14	x=94840	24	96			30				30
		Sum	96							30
15	x=94840	25		88		7		5		12
		28		21		7		5		12
		29			30	5		3		8
		32			103	5		3		8
		33			65	5		3		8
		34			26	5		3		8
		Sum	332							9.3
16	x=94960	40	21			50				50
		41	14			50				50
		42	30			50				50
		Sum	65							50
17	x=94840	33			45	5		3		8
		34			75	5		3		8
		35		16		7		5		12
		36		23		7		5		12
		37		93		7		5		12
		38		38		7		5		12
		39		83		7		5		12
		Sum	371							10.8
21	B2=4	1			57	5	3			8
		2			110	5				5
		3			141	5		3		8
		4			56	5	3			8
		5			28	5	3			8
		6			61	5				5
		7			159	5				5
		8			79	5		3		8
		9			116	5		3		8
		10			63	5	3	3		11
		11			77	5	3			8
		12		62		7	5	5		17
		13		24		7	5	5		17
		14	24			50				50
		Sum	1056							10.6

1	2	3	4	5	6	7	8	9
22 B2=7 4		79		7	5			12
5		132		7	5			12
6		22		7	5			12
7		19		7		5		12
8		117		7		5		12
9	21			50				50
10	117			50				50
Sum	507							22
23 B2=14 19	10			50				50
20	17			50				50
21		43		7	5			12
22		78		7		5		12
Sum	148							19
24 B2=7 1		137		7		5		12
2		94		7		5		12
3		157		7		5		12
Sum	388							12
25 B2=14 1		45		7		5		12
2	40			50				50
3	22			50				50
4	8			50				50
5	47			50				50
6	17			50				50
7	12			50				50
8	12			50				50
9	30			50				50
10	29			50				50
11	43			50				50
12	6			50				50
13	7			50				50
14	20			50				50
15	9			50				50
16	4			50				50
Sum	347							45
26 B2=23 1		59		7	5		15	27
2		145		20	5	5		30
3		26		20	5	5		30
4		25		7	5			12
5		5		7	5			12
6	23			50				50
7	45			50				50
8		15		7	5			12
9		14		7		5		12
10	20			50				50
11	13			50				50
12	7			50				50
13	23			50				50
14	20			50				50
Sum	438							42

1			2	3	4	5	6	7	8	9
27	B2=28	1-8	366			50				50
		Sum	366							50
	B2=26	1	45			50				50
		2		23		7	5			12
		4	20			50				50
		5		7		20	5	5		30
		6		15		20	5	5		30
		7		50		20	5	5		30
		Sum	159							38
		Av.								41
81	B1=2	1		147		7	5	5		17
		2		123		7	5	5		17
		3		90		7	5	5		17
		Sum	360							17
82	B1=9	1		37		7		5		12
		2		53		7		5		12
		3		66		7		5		12
		4		31		7	5	5		17
		5		96		7	5	5		17
		6		90		7	5	5		17
		7		88		7	5	5		17
		8		73		7	5	5		17
		9	68			50				50
		Sum	602							16.6
51	y=31500	1	83			50				50
		2	12			50				50
		3	28			50				50
		4	21			50				50
		5		240		7		5		12
		6		142		7		5		12
		Sum	526							22
52	y=31420	1		57		7	5	5		17
		2		50		7	5	5		17
		3	20			50				50
		4		80		7	5	5		17
		5		120		7	5	5		17
		6		100		7	5	5		17
		Sum	427							18.5
53	y=31360	7	50			50				50
		8	74			50				50
		9		9		7	5			12
		Sum	133							47
54	y=31360	1		10		7	5	5		17
		2		10		7	5	5		17
		3		90		7	5	5		17
		4	30			50				50
		5	10			50				50
		6	30			50				50
		Sum	180							30

1			2	3	4	5	6	7	8	9
55	y=31300	5		92		7	5	5		17
		6		102		7	5	5		17
		7		80		7	5	5		17
		8		95		7	5	5		17
		Sum	369							17
56	y=31300	1	31			50				50
		2		75		7	5			12
		3	27			50				50
		4		68		7	5			12
		Sum	201							23

H.W.-Hanging wall
F.W.-Footwall
W.L.-Waste layer

Possible dilution level

Influencing factor	H.W.	F.W.	W.L.	Thickness of orebody		
				<3 m	5-20 m	>20 m
Rock dilution,%	0-8	0-5	10-20	50	7	5