BRGM

BRGM

# CHESSY (Rhone)

## Copper - Zinc

# Executive Report

Updated Nov. 1987

DAM/PM
B. CABROL

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## EXECUTIVE SUMMARY

The discovery by BRGM of the copper-zinc deposit at Chessy, close to Lyons in France, where the "Ancients had mined since about five centuries some surficial ore shoots, is the result of the application of modern exploration techniques to find hidden orebodies. The exploration was carried out with the proper means of BRGM, which holds the mineral rights over the area that contains the known orebody and its probable extensions. Since 1983, 2.5 km of drifts and 8.1 km of cored drilling have shown the existence of a massive-sulphide orebody of several million tons.

It occurs as a continuous body from 100 to 400 m depth and over a length of 700 to 900 m, consisting of layer that is 5 to 20 m thick, dipping at 35° toward SE. To the 4.4 million tonnes ore proven to date, must be added a considerable potential down-dip: this extension can be drilled off in the future as and when required, to ensure an extension of the presently-known mine life of 12 years, with an annual production of 330,000 tonnes ore.

To the east, the orebody is truncated by a regional, normal fault and all indications point at an extension at depth of the orebody in the down-thrown block; this possibility, however, should be investigated by the future mining company.

Taking into account the dilution factor inherent in exploitation, the run-of mine ore will grade 2.24 % Cu and 9.45 % Zn, which are values that are comparable to those of other orebodies of this type that are mined worldwide. To these two elements should be added barite (26 %) and pyrite (39 %) as by-products, which are of real value in the European industrial context.

The large size of the orebody will permit a high degree of mechanization of the underground mining, with the use of autonomous diesel-powered equipment. The mining method, as optimized with the help of the "Ecole des Mines" of Paris, could be as follows: sublevel stoping is followed by back-filling with waste bonded by cement; in areas of great thickness of the orebody, rooms perpendicular to the mineralized layer will be separated by pillars of the same dimension, which will then be mined in turn. In this manner, all variations in orebody thickness and dip can be accomodated for, as well as any problems in hanging-wall and foot-wall stability. This method ensures better recovery and less dilution than any of the other methods that were considered.

After a first crushing at the bottom of the mine, ore and waste will be extracted by conveyor belt, which reduces capital investment costs and allows great flexibility for further extension at depth, without having any impact on the environment. Truck haulage, using high-capacity articulated trucks, is being studied again. This solution could be adopted during the first years for mining the shallow North Block at least. Several tests have verified that the waste of the mine-layout works will form backfill of the required composition and resistance; the back-fill will be stowed by slinger-belt trucks, according to a procedure that has been well-tested in several recent European mines. In the case of Chessy, operational costs for backfilling are minimized because of the high density of the ore (4.2 t/m$^3$), which implies a reduced volume to be back-filled for each ton of extracted ore.

Ore beneficiation was tested in the laboratory at Orleans, but also several times on industrial scale in a continuous pilot plant. Even though complex, the processing is particularly effective; with excellent recovery rates that vary from 92 to 94 %, for Cu and Zn. Marketable concentrates of copper, zinc, barite and pyrite are obtained, which are of remarkable quality. The mill should be automated to a maximum, steered by a central computer.

The possible sites for mill-tailings dumps, have been inventoried and their capacities have been evaluated. Two sites close to the mill appear to be particularly attractive, but no final selection has been made. The retaining dam that should have long-term stability, will be built of run-of-mine waste and the potential tailings-pond volume is greater than that required for the projected mine life.

Industrial water supply, to the tune of 80 $m^3$/hr of new water, will in part be covered by pumping from the mine itself, but will come mostly from boreholes into the Triassic sandstone aquifer, which has already been tested by drilling and is close to the mine site. This would avoid any pumping of surface water, which could entail environmental problems.

In designing the mill flow-sheet, great importance was attached to choosing little-polluting reagents, which, aided by the automatic distribution of the reagents, will easily satisfy the standards to be respected before discharging into surface water.

Concerning concentrate transport, various solutions were studied. The investor will have to make the final choice between rail transport from a private siding, transport entirely by truck, mixed transport by truck and train, or truck and barge from the river port at Villefranche, or any combination of these solutions. The economic evaluation was based on the first choice that was studied in more detail, but, as said, the choice remains open and the operator will be able to adapt to transport-rate fluctuations.

The industrial-site planning was done in close collaboration with the various administrations concerned and taking into account all zoning regulations and environmental constraints. All necessary administrative steps have already been taken, if not entirely then at least in part, to accelerate final projet development.

The geographical location is particularly favourable as the site benefits of complete infrastructure facilities, either on-site on in the immediate neighbourhood; these include railway, electrical power, lodging facilities and the insertion of future workers in a social environment without problems. Experience has already shown that both the local population and the elected officials welcome this project, which is line with an ancient mining tradition. All environmental problems have always been approached from a viewpoint of mutual understanding.

Starting up the mine will create 130 to 150 jobs; the industrial development of the region will allow to make major use of sub-contractors.

Detailed engineering design and execution of the project will ask a little more than two years and a capital investment of about 242 million FF in constant 1986 francs. These amounts represent maxima that can be revised downwards, in particular for the mine-layout development work as its costs were based on non-negotiated figures. Chessy will, without doubt, be the most important French base-metal mine. Its polymetallic character, its size, and the grades of its ore make it into a mining affair that will be far more resistant to metal-price flucuations than many an other project. It will, moreover, be the first copper producer of significance in France.

The regional "metal" potential is promising as well, as other massive-sulphide bodies have been mined in the past. Regional exploration is presently being carried out, using methods that were developed and tested on the Chessy model.

The following study attemps to present the full range of possible technical solutions by discussing pros and cons of each, but without making a choise. The solutions that were adopted (mid-1986 prices) for the economic study are in general those that presented the best cost evaluations; this study therefore does not present an economic optimum and is open to further improvement in this field by the investor.

THE CHESSY DEPOSIT IN FIGURES

- Exploration : 2 500 m of drifts - 102 cored boreholes (8.1 km)

- In-situ reserves : 4.4 Mt (North block 1.9 Mt 43 boreholes, 3 030 t/m mineralized
  (South block 2.5 mt 5 boreholes, 19 060 t/m mineralized
  . Grade : Cu = 2.34 - Zn = 9.51 - Pyrite = 41.70 - Barite = 26.32

- Production rate considered :

  . Mine : 5 d/week - 2 to 3 shifts - 220 d/year - 330,000 t/year
  . Mill : 7 d/week - 3 shifts - 321 d/year

- Run of mine ore : 3.7 Mt

| | | | |
|---|---|---|---|
| (North block | 1.5 Mt | ) mine life | ) 12 years |
| (South block | 2.2 Mt | ) | ) |
| Potential reserves | 2.20 Mt | mine life | ) 6 years |
| TOTAL | 5.7 Mt | | |

  . Grades after dilution and selection : Cu = 2.24 % - Zn = 9.45 % - Pyrite = 39.24 % - Barite = 26.21 %

  . Specific gravity 4.2 t/m3

- Mining method : Sub-level stoping with cemented back-fill

  . Extraction : primary crushing underground + conveyor belt haulage

- Flotation of 4 main marketable concentrates

| Concentrate | Recovery | Grade | 1 000 Dry metric tonnes/year |
|---|---|---|---|
| Cu | 92 % | 28.3 % Cu | 18 - 24 |
| Zn | 94 % | 56.0 % Zn | 34 - 60 |
| Pyrite | 83 % | 52.5 % S | 69 - 122 |
| Barite | 77 % | 98 % $BaSO_4$ | 56 - 75 |
| Pb - Ag | for the record | p.m. | p.m. |
| | | | 230 |

- Personnel : 139 (Staff 9 - Technician, foremen 38 - Labourers 92)

- Transport : truck - barge - rail (which was retained for estimate) 1 200 t/day

- Selling of concentrate :

. Marketing

| | |
|---|---|
| cooper | : normal marketing formula - no impurities - good concentrate |
| zinc | : normal marketing formula - no impurities - very good concentrate |
| piryte | : very good concentrate |
| barite | : chemical quality (possibly white barite) |
| lead silver | : for the record |

| | |
|---|---|
| . Transport | : 137 to 161 F/t depending on concentrate and destination |
| . Loss | : 1 % |
| . Sales cost | : 0.8 % to 1.5 % |
| . Metal prices | : for the record |

| Aug. - Sept. - Oct. 1986 | October 1987 |
|---|---|
| 1 US$ = 6,65 FF | 1 US$ = 6,02 FF |
| Cu = 9 104 F/tm = 0,62 $/lb | Cu = 12 120 F/tm = 0,91 $/lb |
| Zn = 6 331 F/tm = 0,43 $/lb | Zn = 5 180 F/tm = 0,39 $/lb |

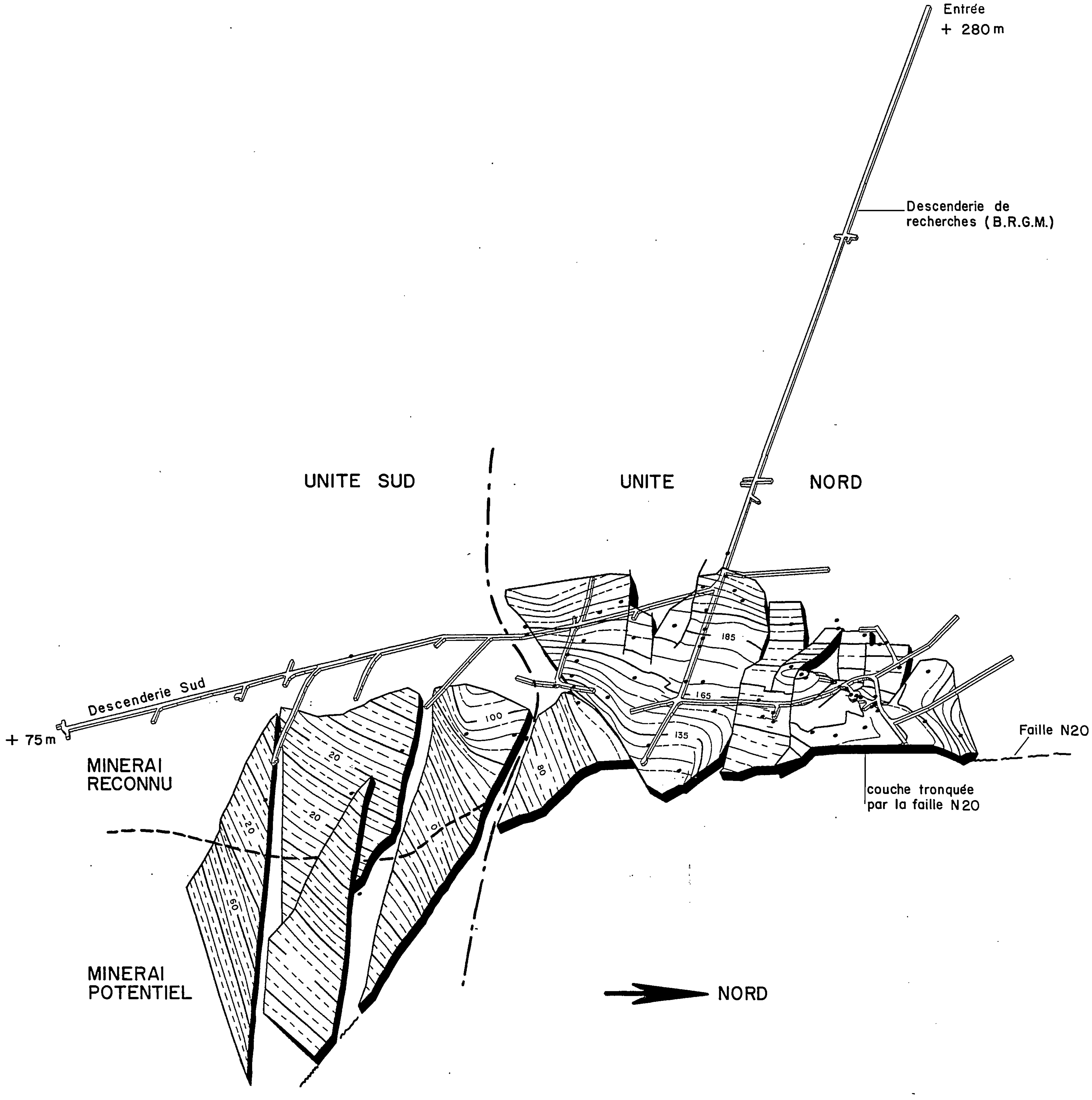
- Operating cost : F/1986/t ore

| | Personnel + sub-contrating | Consumables + energy | TOTAL |
|---|---|---|---|
| Mine | 49 | 52 | 101 |
| Mill | 27 | 49 | 76 |
| Technical services | 12 | 14 | 26 |
| Administrative and social services | 17 | 12 | 29 |
| TOTAL | 105 | 127 | 232 |

- Capital investment 1986 Million Francs

| | | |
|---|---|---|
| . Preliminary expenses : | | |
| - previous work by BRGM | | p.m. |
| - studies + engineering (1 year), followed by construction (2 years) | | 39.0 |
| - development work | | 10.0 |
| . Mine (including development) | 83.0 ) | |
| - mill | 60.0 ) | 191.0 |
| - technical services | 38.2 ) | |
| - administratives and social services | 9.2 | |
| . Acquisition of land | | 1.6 |
| | TOTAL | 241.6 MF |

. Working capital requirements (year 1) : minimal because of credit from suppliers




# CHESSY
## Bloc diagramme

**LEGENDE**

- Courbe de niveau du toit de la couche minéralisée, espacement = 10 m
- Courbe intermédiaire, esp. = 5 m
- Descenderies ou passages
- Intersection des sondages et du toit la couche minéralisée

# SUMMARY

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# *CHESSY*

# *BASIC DATA*

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# I - GENERAL INFORMATION

## 1 - LOCATION, HISTORICAL AND ADMINISTRATIVE BACKGROUND

Chessy is a small community of about 1 300 habitants that lies 25 km NW of Lyons in the Azergues valley at an altitude of 220 m.

As its name indicates, Chessy-les-Mines has a long mining tradition. One kilometre north of the town, old waste dumps indicate such activity that dates back to at least the 15th century, and it is exactly at this location, but deeper, that the new orebody was discovered.

### *1.1 - Historical overview*

The first documents are from the period of the Baronat brothers who were associated with Jacques Coeur (1444 to 1452).

The ore at that time was extracted form various mines: the "red mine" contained oxide ore (cuprite) in faults, the "black mine" had chalcocite of supergene-cementation origin, but the most important was the "yellow mine" that yielded pyrite, chalcopyrite, and minor sphalerite and barite from between the "green horns" (basic rocks) and the "pink horns" (rhyolite). The orebody of the "yellow mine", dipping at 40° to 70° to the east, must have provided more than 150,000 tons ore that were roasted at the site and then leached to extract the copper by cementation.

In 1810, the Saxon Tranglot Wöllner discovered the "blue mine", containing well-crystallized secondary azurite (Chessylite) and malachite in Mesozoic sandstones. This body was mined until 1831 and yielded about 4 000 tons copper metal.

In 1886, the "exploration shaft Q" was sunk to 156 m depth, in the hanging wall of the "yellow mine", which discovered that the ore pinched out against a major fault. Mining activity then restricted itself to extraction of the pillars left by earlier miners.

All told and including all mine "colours", mining was developed over the centuries over a length of 800 m and to 50 to 70 m depth (occasionally to 140 m depth), yielding about 10,000 to 14,000 tons copper metal.

After the Sain-Bel pyrite mine was closed, and after the Compagnie Industrielle et Minière (Subsidiary of Rhône-Poulenc) had pulled out of the region, BRGM resumed exploration work, based on its world-wide experience in the search and discovery of massive-sulphide deposits and its mastery of modern exploration tools and techniques.

1978-1980: Helicopter-borne electromagnetic "Dighem" survey, followed by ground checks. Detailed gravimetric surveys.

1980-1983: Seventeen cored boreholes on a grid of 100 × 150 m that encountered major technical difficulties (7 holes had to be abandoned), of which 9 intersected mineralization, 4 with intersections of several tens of meters.

- Promising ore-processing tests were carried out on the drill cores.

1983-1986: Construction of a decline of 11 $m^2$ cross-section and 664 m length; exploration by means of drifts and drilling from the ends of the north and south vein drifts. Total of 2 500 m of decline and drifts, and 4 400 m of underground drilling.

- Test mining.
- Ore-processing pilot test on bulk sample.
- Economic, legal, and administrative evaluation.
- Acquisition of necessary ground rights.

### *1.2 – Administrative and legal position*

BRGM has carried out exploration by itself and from its own means, with the aid of the "Plan Métaux" over the period 1979-1984.

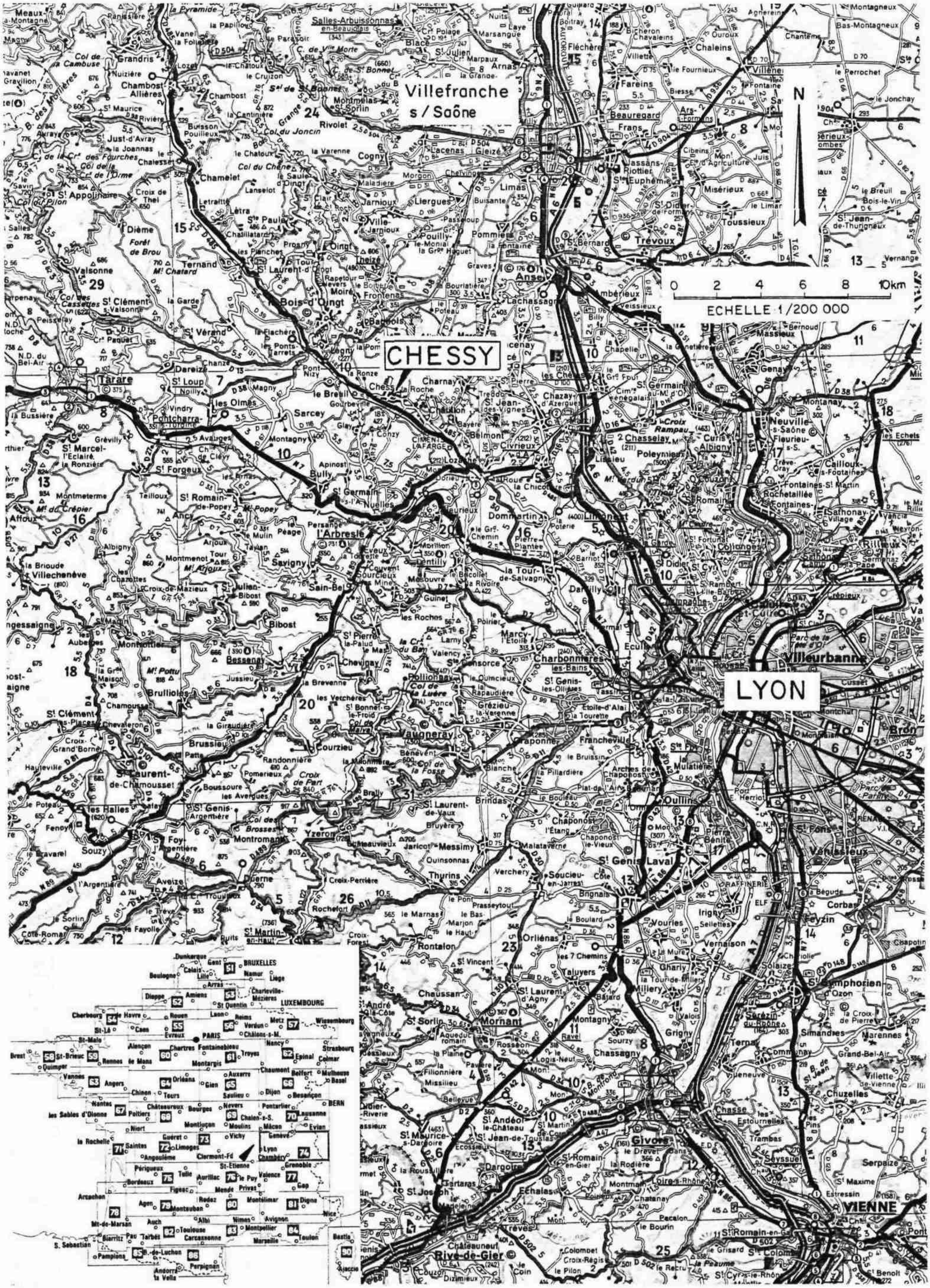
On 13 December 1985, BRGM requested a concession named "De la Ronze" with a surface area of 7.76 $km^2$, which is spread over six communities. This request was submitted to public enquiry in February-March 1986, and will be granted in 1988.

By agreement dated 30 December 1986, BRGM has acquired the right-of-first-refusal on all grounds held by the Compagnie Industrielle et Minière that were favourably located for the future mine and mill site. This domain covers almost 15 ha and will be acquired within six months following upon granting of the concession. At a later date, several parcels of grass- and bush-land should be bought in the La Goutte Granger and the La Ronze valleys, mainly to provide storage sites for the mine dumps and mill tailings.

From the start of exploration, and especially since the first steps were taken towards an industrial project, full information was provided to "Direction Regionale de l'Industrie et de la Recherche" (DRIR Rhône-Alpes) as well as to all other administrative bodies. This policy of dialogue has, until present, caused a smooth progress of the files, reducing to a minimum all discussions and thus the investigation periods.

### *1.3 – Topographical data*

Except for the IGN (French Ordnance Survey) and cadastral maps, maps at scale 1:1,000 were prepared for BRGM by the Albenque surveying firm from data gathered on special flights, with contours every meter and covering an area of 175 ha.



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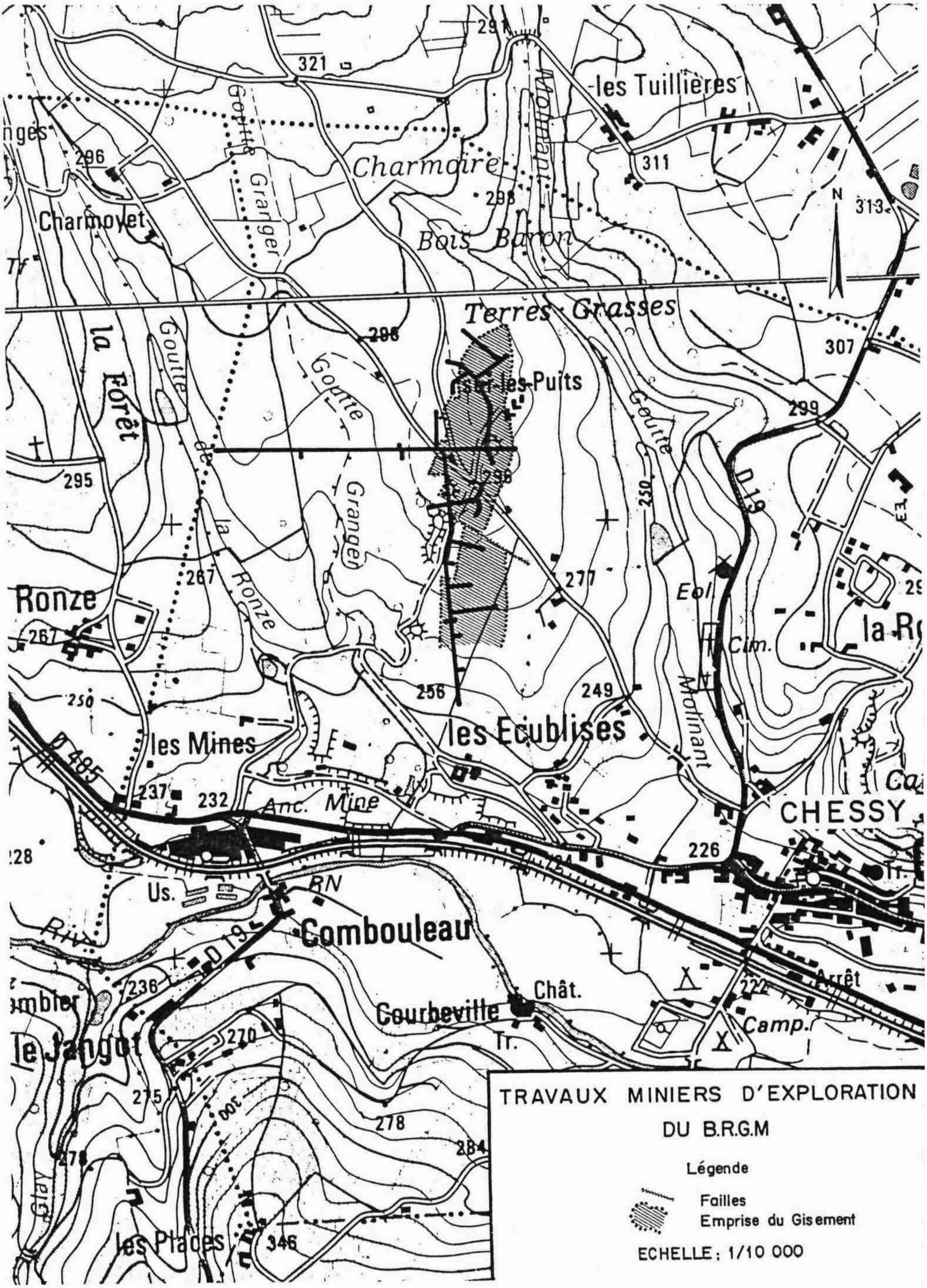
ECHELLE 1/200 000
CHESSY
LYON


les Tuillières
Charmaire
Charmoyet
Bois Baron
Terres Grasses
Goutte Granger
la Forêt
Goutte de la Ronze
Goutte Granger
Goutte
Molmant
Ronze
les Mines
les Ecublises
Anc. Mine
CHESSY
la Ro
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Us.
RN
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D 19
Combouleau
Courbeville
Chât.
Tr.
Arrêt
Camp.
le Jangot
les Places
Glay
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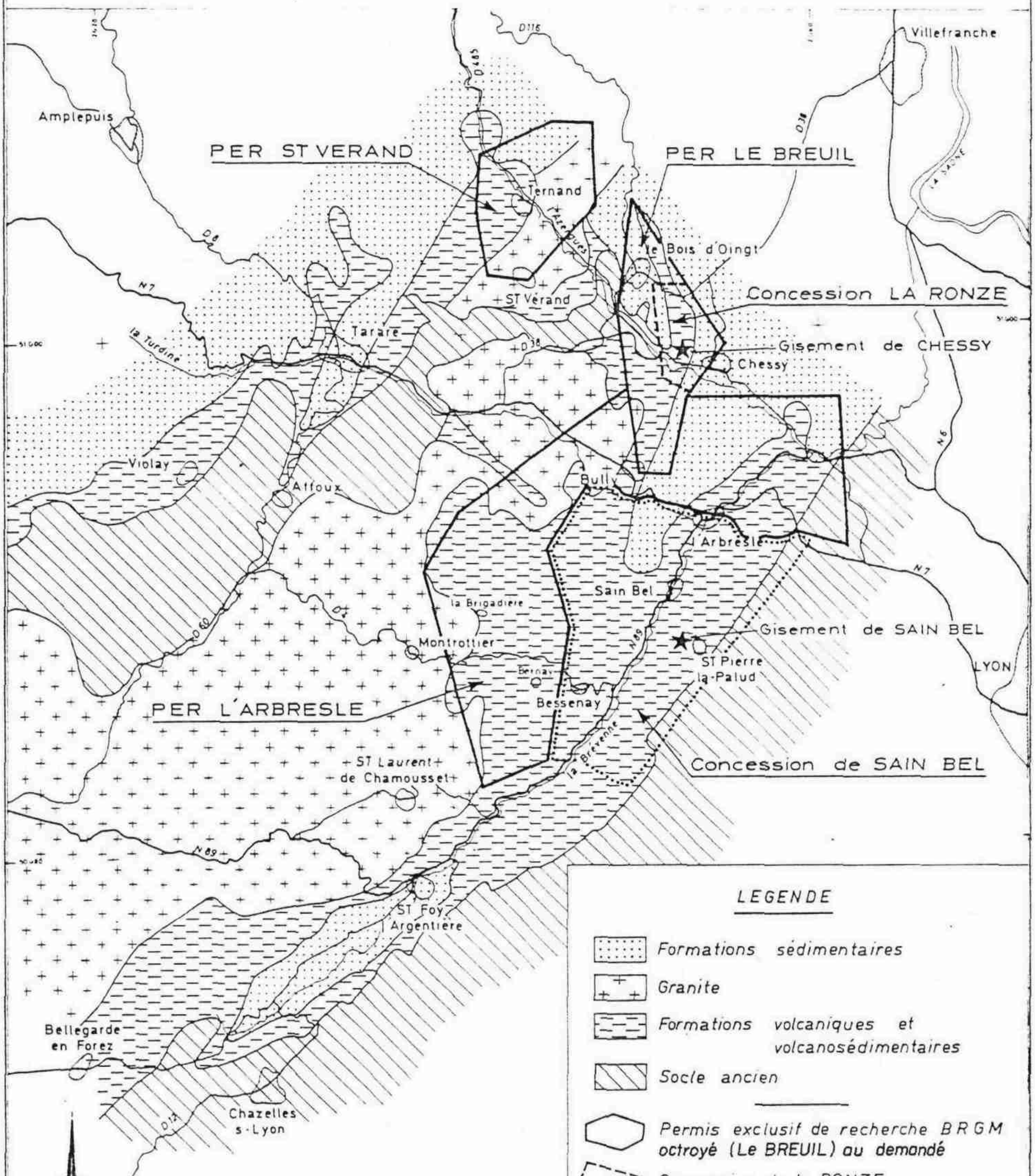
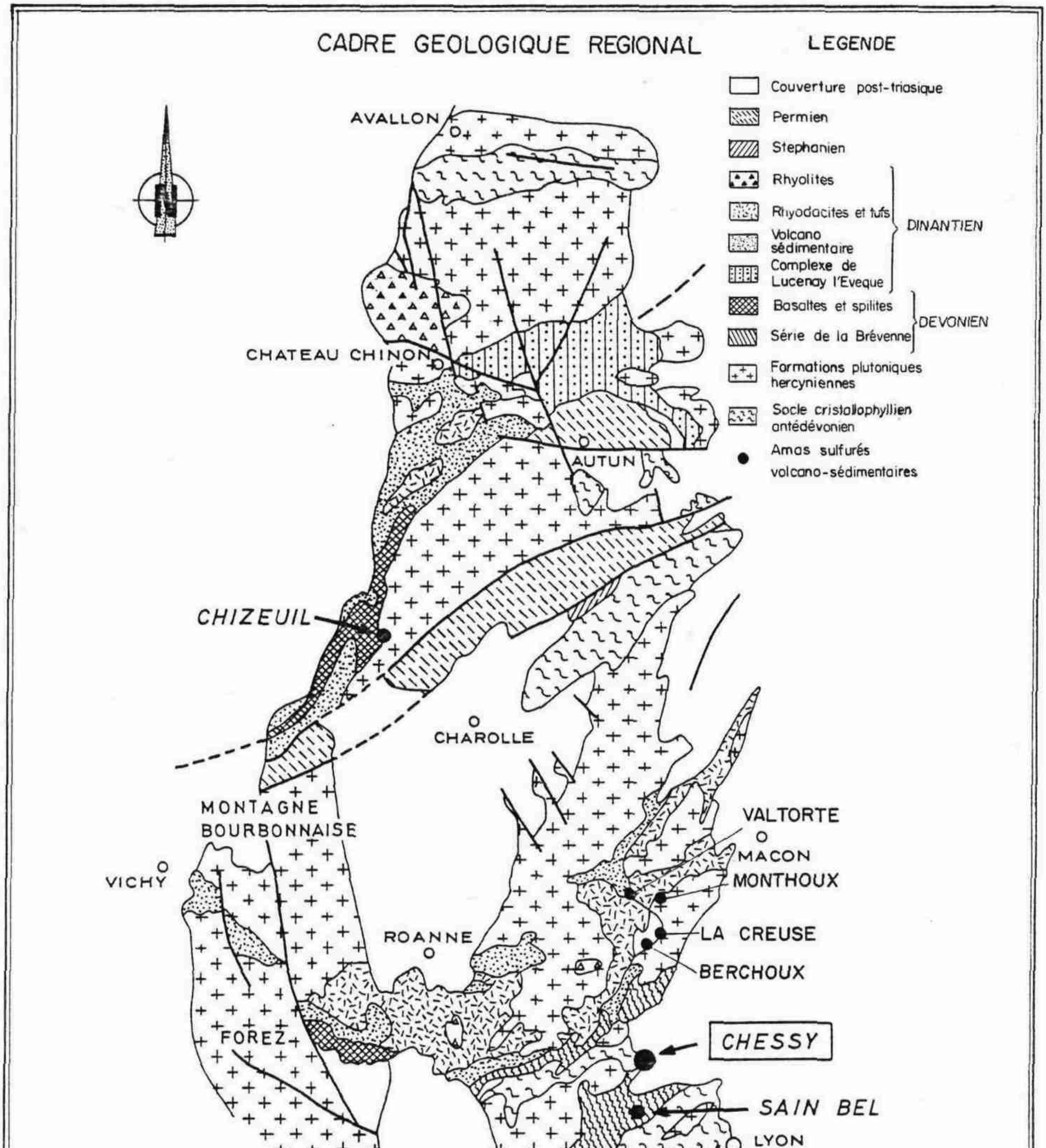
300
TRAVAUX MINIERS D'EXPLORATION
DU B.R.G.M
Légende
Failles
Emprise du Gisement
ECHELLE : 1/10 000

BRGM

# CARTE DES ZONES D'ACTIVITE DANS LA REGION DE CHESSY (Rhône)




CADRE GEOLOGIQUE REGIONAL
LEGENDE
Couverture post-triasique
Permien
Stephanien
Rhyolites
Rhyodacites et tufs
Volcano sédimentaire
Complexe de Lucenay l'Eveque
DINANTIEN
Basaltes et spilites
Série de la Brévenne
DEVONIEN
Formations plutoniques hercyniennes
Socle cristallophyllien antédévonien
Amas sulfurés volcano-sédimentaires
AVALLON
CHATEAU CHINON
AUTUN
CHIZEUIL
CHAROLLE
MONTAGNE BOURBONNAISE
VICHY
VALTORTE
MACON
MONTHOUX
LA CREUSE
BERCHOUX
ROANNE
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SAIN BEL
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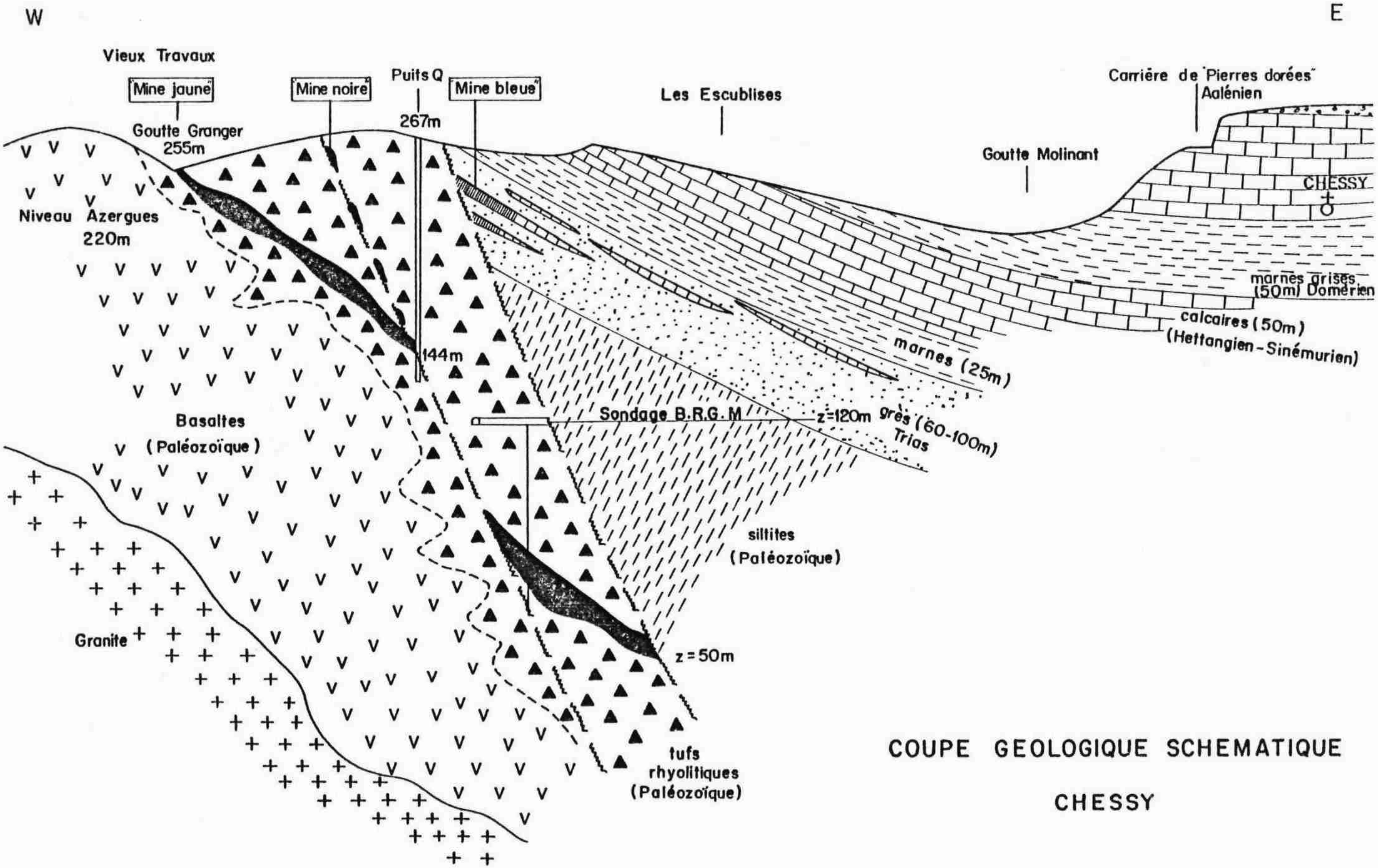
(D'aprés J.C. CHIRON 1972)




COUPE GEOLOGIQUE SCHEMATIQUE

CHESSY

## II - GEOLOGICAL SETTING

The Chessy deposit lies within Middle Paleozoic (Devonian) volcanic and sedimentary rocks, known as the "Série de la Brévenne". More particularly it is related to light-coloured volcanic rocks (rhyolites) and falls within the category of "massive-sulphide bodies", which all over the world (Canada, Japan, Australia, Iberian peninsula, etc.) cover a major part of the world-production of copper, lead and zinc.

### 1 - LOCAL STRATIGRAPHY

The pyrite, sphalerite, chalcopyrite and barite mineralization, which is from 0 to 20 m thick, is found at the contact between acid-volcanic units. From foot- to hanging-wall one can see:

- The lower acid unit, which can contain a breccia with disseminated sulphides (low Cu-Zn grades) related to the proximity of old stocks from which the stockworks were formed.
- A horizon of massive sulphides with pyrite, sphalerite, chalcopyrite, with accessory galena and a barite gangue, 0-20 m, and exceptionally up to 35 m, thick.
- An upper acid unit consisting of acid-lava flows.
- Basic, massive basalt, dark green and commonly bedded; these are rocks of good strength, abundant to the west of the mineralized horizon and which can intrude and completely replace all earlier-mentioned rocks, including the mineralized horizon.
- Granite of late-Hercynian age, crops out in the Ronze valley along a N-S axis with numerous off-shoots. It causes slight contact metamorphism, which imparts a hornfels aspect to the basalt and caused the sulphides to recrystallize in large grains.
- Lampophyre dykes commonly cut ore and host rocks; they strike about E-W, are subvertical and of variable thickness (0.10 to 4 m). Such dykes have scarred older faults, especially those that caused off-sets.
- Siltstone (black shales, and pelite), enriched in carbonates, are only known from the orebody area in the panel to the east of the N 20 fault. This very thick succession corresponds to a sedimentation cycle of very fine material.

### 2 - MAIN STRUCTURAL SETTING

An E-W section shows the existence of two well-defined compartments or panels that are separated by the N 20 fault.

- The west panel is composed of Paleozoic rock units and contains the known part of the orebody.

- The N 20 fault forms a major, regional feature that can be followed over several tens of kilometers, whose vertical down-throw to the east is more than 150 m, with an unknown horizontal displacement. This fault, which dips 80°E on the north side of the orebody and 60°E on the south side, cuts and interrupts the mineralization longitudinally.

- The east panel is down-thrown and shows sedimentary rocks of Triassic to Early Jurassic age that overlie the Paleozoic units. In view of the unknown direction and amplitude of the horizontal component of movement along the N 20 fault, it is not known where the amputated part of the orebody is located. This, however, could present a considerable economic potential.

## 3 - THE OREBODY

The known part of the orebody lies in the west panel and forms a continuous sedimentary unit over more than 800 m, along a N-S axis. Through the influence of tectonic events, it can be subdivided in a North Block and a South Block.

- The North Block is 450 m long, and between 100 and 160 m wide. Departing from an anticlinal structure with a general westerly dip on the extreme north side of the orebody, the unit becomes isoclinal, dipping at 35° to the east at the level of the main decline, before turning to a subvertical position further south.

On the northwest edge the orebody closes: enrichment in barite, decrease in sulphide content, and thinning of the unit all indicate the nearness of the edge of the sedimentary basin.

- The South Block, which is presently known over a length of 350 m and is up to 250 m (or more) wide, is down-thrown over 70 m when compared to the North Block, through the movement along a NW-SE fault. The block dips 30-40° toward the southeast.

## 4 - THE MINERALIZATION

The main mineralized horizon commonly shows vertical zonation, with at the top an ore with ribbons that are either rich in sphalerite, or rich in pyrite and chalcopyrite, and which shows sedimentary structures such as slumps and flowage folds. Below this lies a more-massive ore, little or not ribboned and very pyritic.

Laterally, when approaching the edges of the massive sulphides, pyrite and chalcopyrite disappear progressively, to be replaced by a well-ribboned ore, very barite-rich and with sphalerite beds. Even farther, only the barite is left in a "distal" facies.

The component elements of the ore are crudely recrystallized and, averaged out over the whole orebody, represent:

| | |
|---|---|
| 6.5 % chalcopyrite | (2.24 % Cu) |
| 0.3 % galena | (0.29 % Pb) |
| 15.0 % sphalerite | (9.60 % Zn) |
| 39.4 % pyrite | (total absence of pyrrhotite) |
| 26.1 % barite | (15.35 % Ba) |
| 12.7 % waste | (lamprophyre + silicified gangue + basic sills) |
| 100.0 % | |

Silver, present in modest quantities (21 g/t) is preferentially bound to galena (15 g/t Ag for 1 % Pb) as well as to tetrahedrite.

Sphalerite belongs to the Fe-poor pole, with 2.2 % Fe for 64.1 % Zn ; it contains cadmium but no germanium.

## III - EVALUATION OF IN-SITU RESERVES

### 1 - EXPLORATION BY DRILLING

As of November 1986, the BRGM Group (CFFM and the Department of Heavy Works) had carried out:

- 21 cored boreholes (S1 to S21) from surface, totalling 3 209 m;
- 82 cored boreholes (S101 to S182) underground, totalling 4 888 m;
- 84 percussion boreholes (S201 to S284) from surface to gather geological and structural information, totalling 2 913 m.

Borehole recovery in the ore was close to 100 %; in the hostrock if was variable.

The underground development work comprises 2 500 m, which was entirely carried out by the BRGM Dept of Heavy Works and consists essentially of:

- an access decline from surface to the 160 m level, 664 m long, sloping at 18 %, with a concrete floor and 11 $m^2$ in cross-section until the end;

- a drift to the north on the 164 m level, 9 $m^2$ section, following the hanging wall in part and in part the footwall, with cross-cuts for drill-holes. The north extension of the orebody was well defined in this manner, and it was also in the north part that the bulk samples were taken for pilot-processing tests and that two experimental rooms were excavated for geotechnical studies;

- a decline to the south with a cross-section of 11 $m^2$, a slope of 18 %, and 620 m long with cross-cuts every 40 to 70 m.

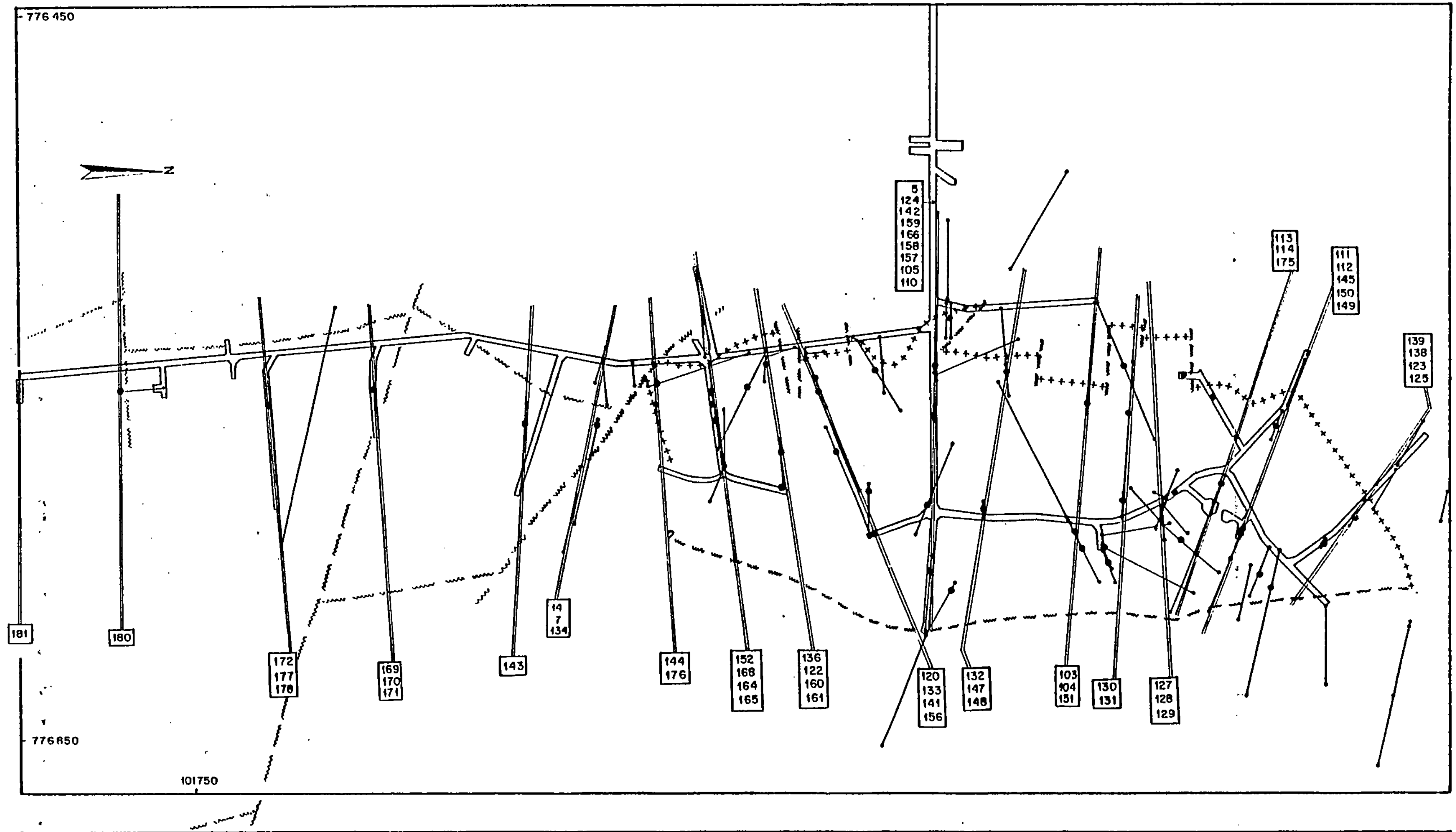
### 2 - EVALUATION OF IN-SITU RESERVES

The reserves were calculated from 43 representative borehole intersections for the North Block and from 5 for the South Block, which appears quite satisfactory for a massive-sulphide body of the volcano-sedimentary type and in view the low variability in grades as observed at Chessy.

The calculated specific gravities are 4.4 $t/m^3$ for the massive ore, and 4.2 $t/m^3$ when taking into account the barren intercalations such as dykes and sills.

No oxide mineralization was observed in the boreholes, but it must be remembered that the highest intersections were at 100 m below surface.

The reserves were estimated by means of the classical polygon method. A geostatistical evaluation has further confirmed and refined the reserve evaluation of the North Block, on which many data points were available.




++++++ Limite de la minéralisation

● Milieu intersection mineral

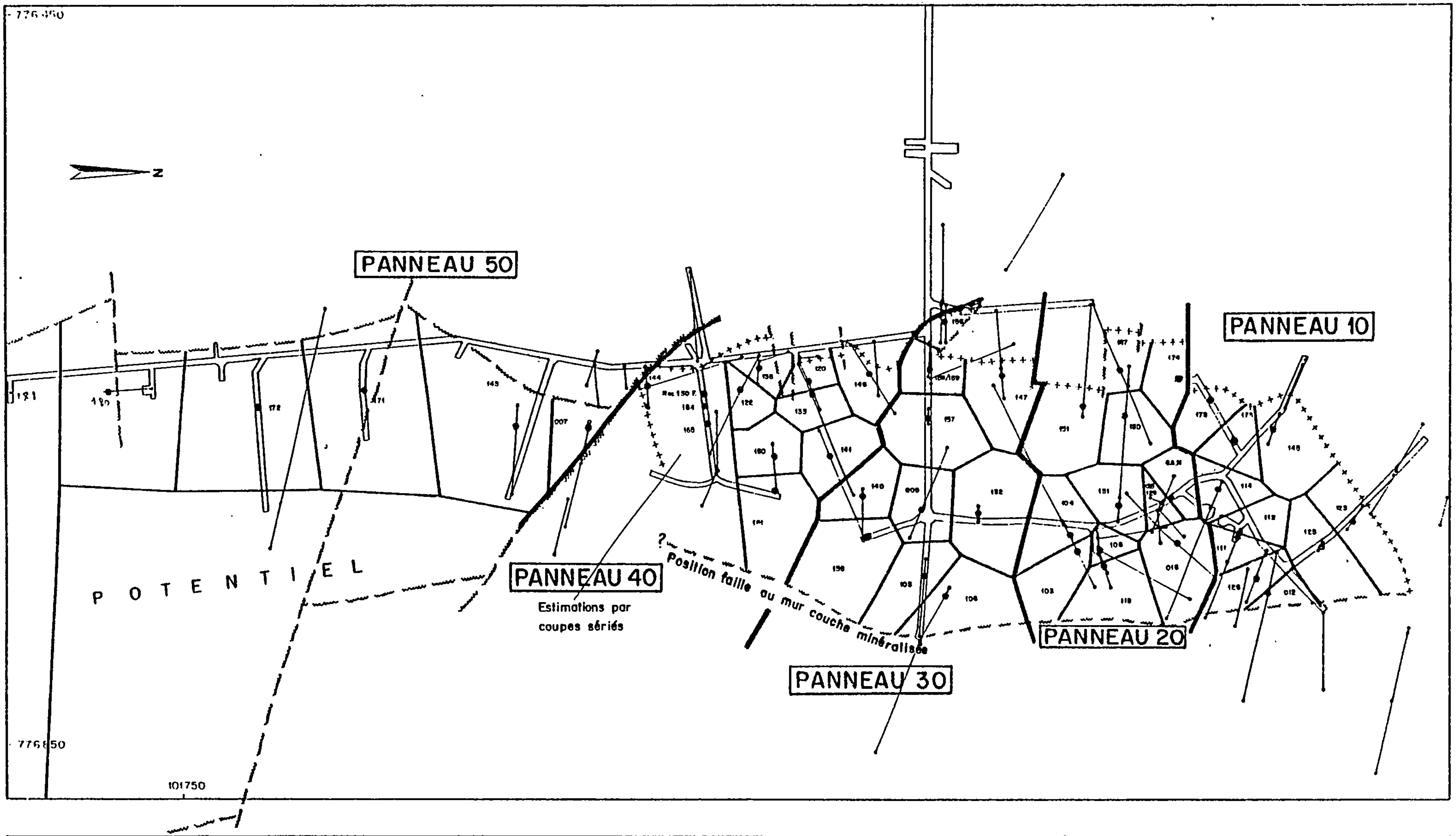
130 131 Numéro des sondages situés sur les coupes

0 50m

CHESSY

POSITION DES COUPES

10-86




++++++ Limite de la minéralisation

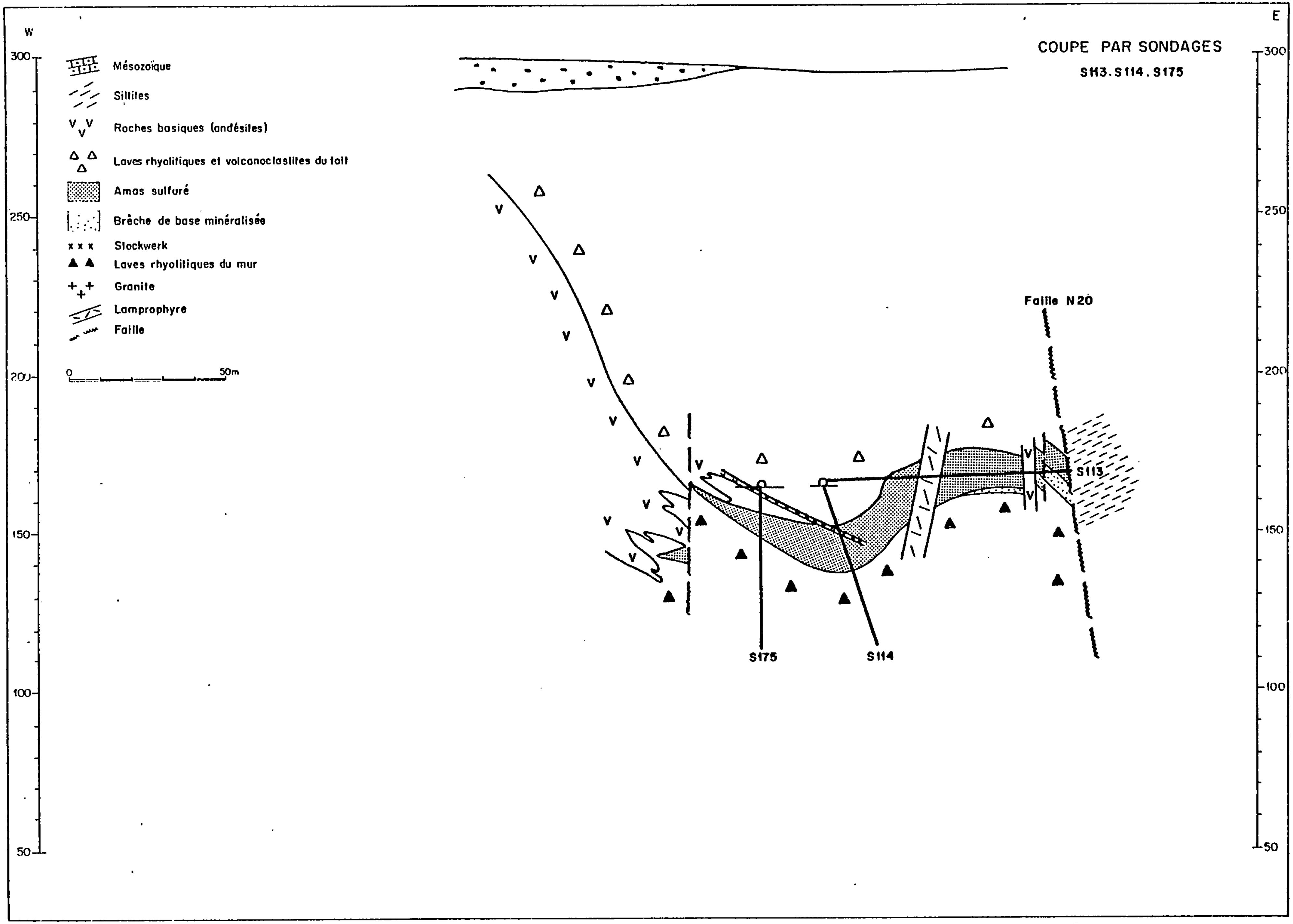
0 50m

• Milieu intersection mineral

CHESSY

INTERSECTIONS SIGNIFICATIVES
ET SURFACE D'INFLUENCE

6-86


COUPE PAR SONDAGES
S113.S114.S175
W
E
Mésozoïque
Siltites
Roches basiques (andésites)
Laves rhyolitiques et volcanoclastites du toit
Amas sulfuré
Brêche de base minéralisée
Stockwerk
Laves rhyolitiques du mur
Granite
Lamprophyre
Faille
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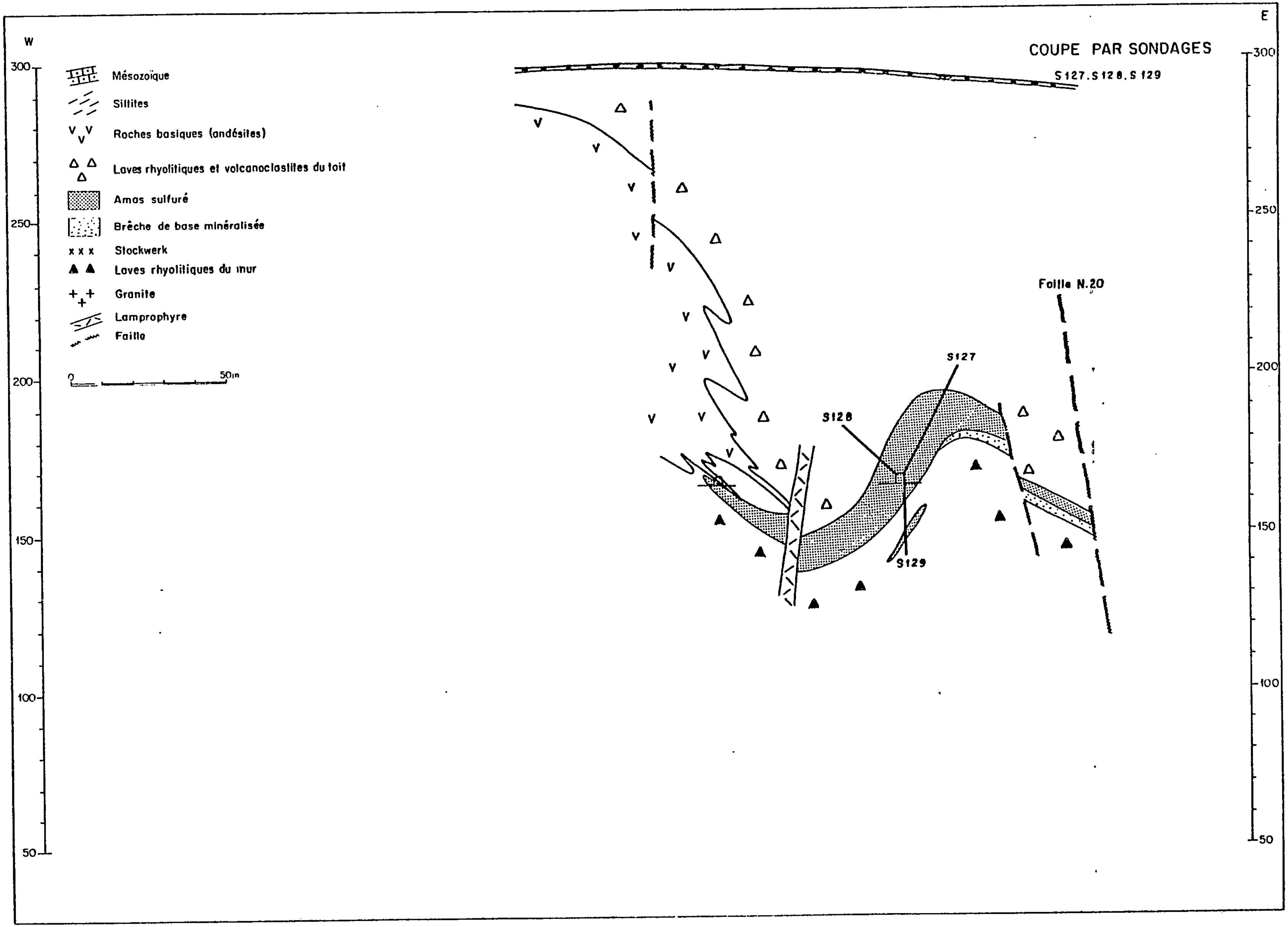
Faille N20
S113
S175
S114



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Siltites
Roches basiques (andésites)
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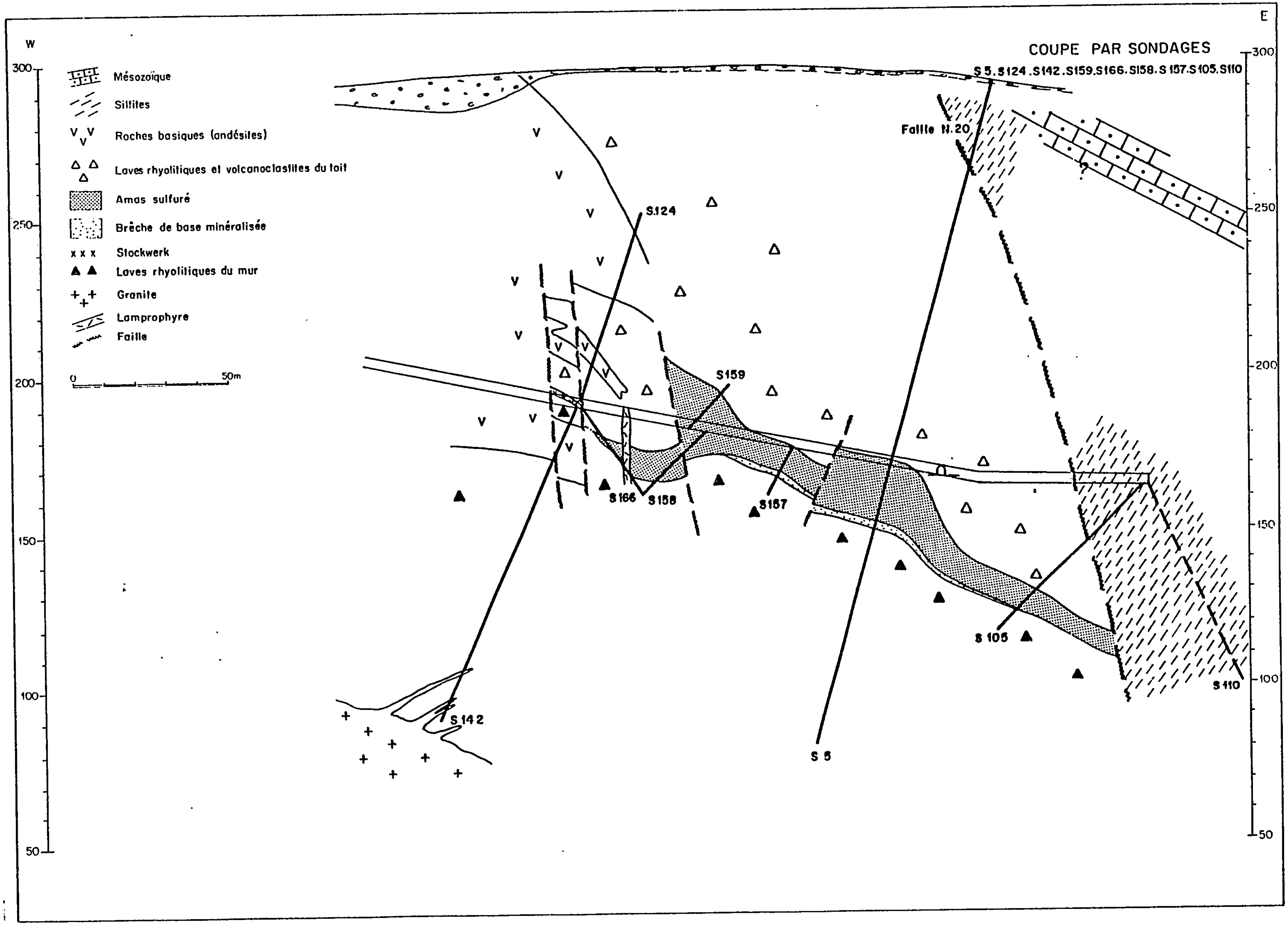
S127
S128
S129
300
250
200
150
100
50



W
E
COUPE PAR SONDAGES
S 5. S124 .S142 .S159.S166. S158. S 157.S105.S110
Mésozoïque
Siltites
Roches basiques (andésites)
Laves rhyolitiques et volcanoclastites du toit
Amas sulfuré
Brèche de base minéralisée
Stockwerk
Laves rhyolitiques du mur
Granite
Lamprophyre
Faille
0
50m
300
250
200
150
100
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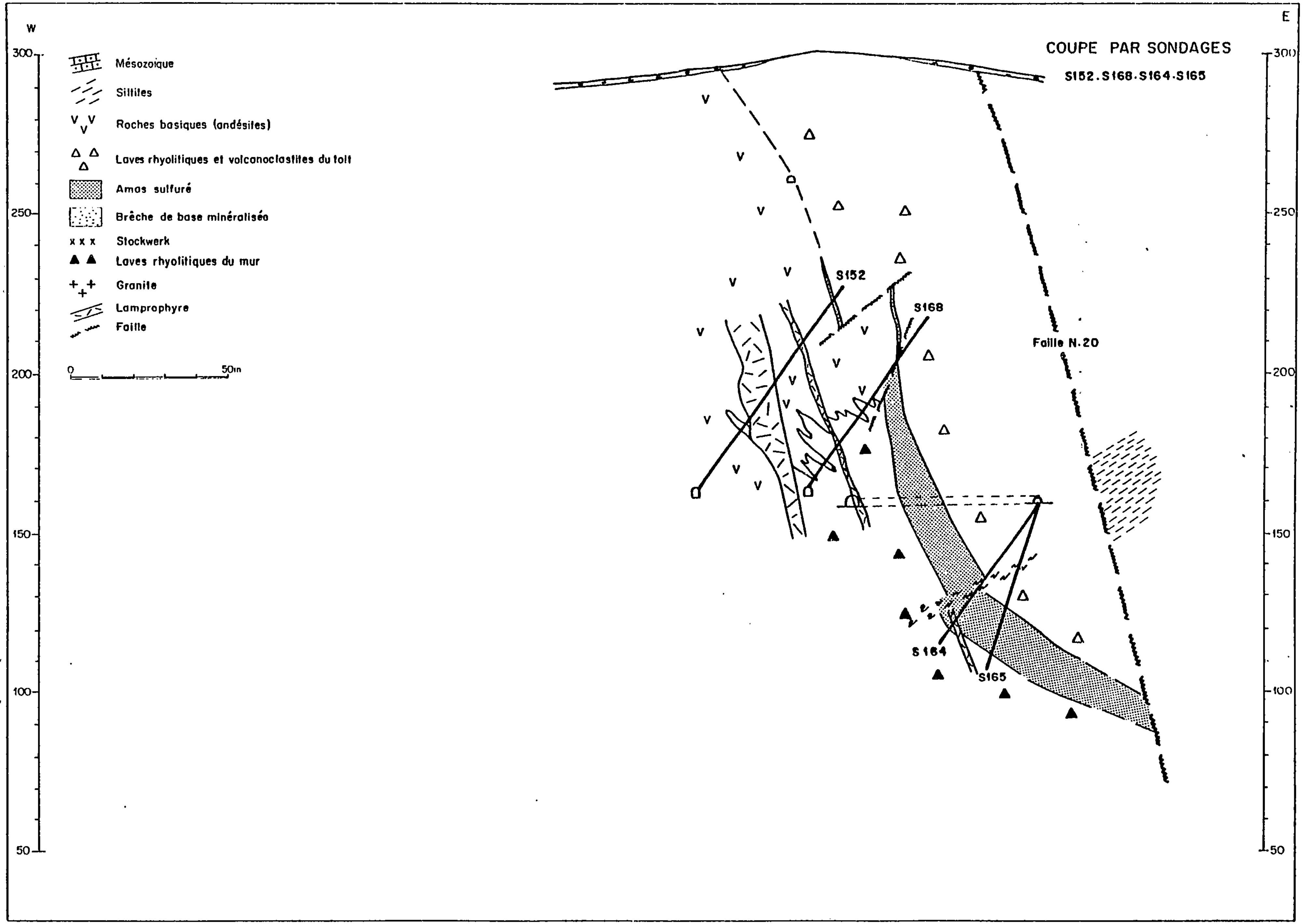
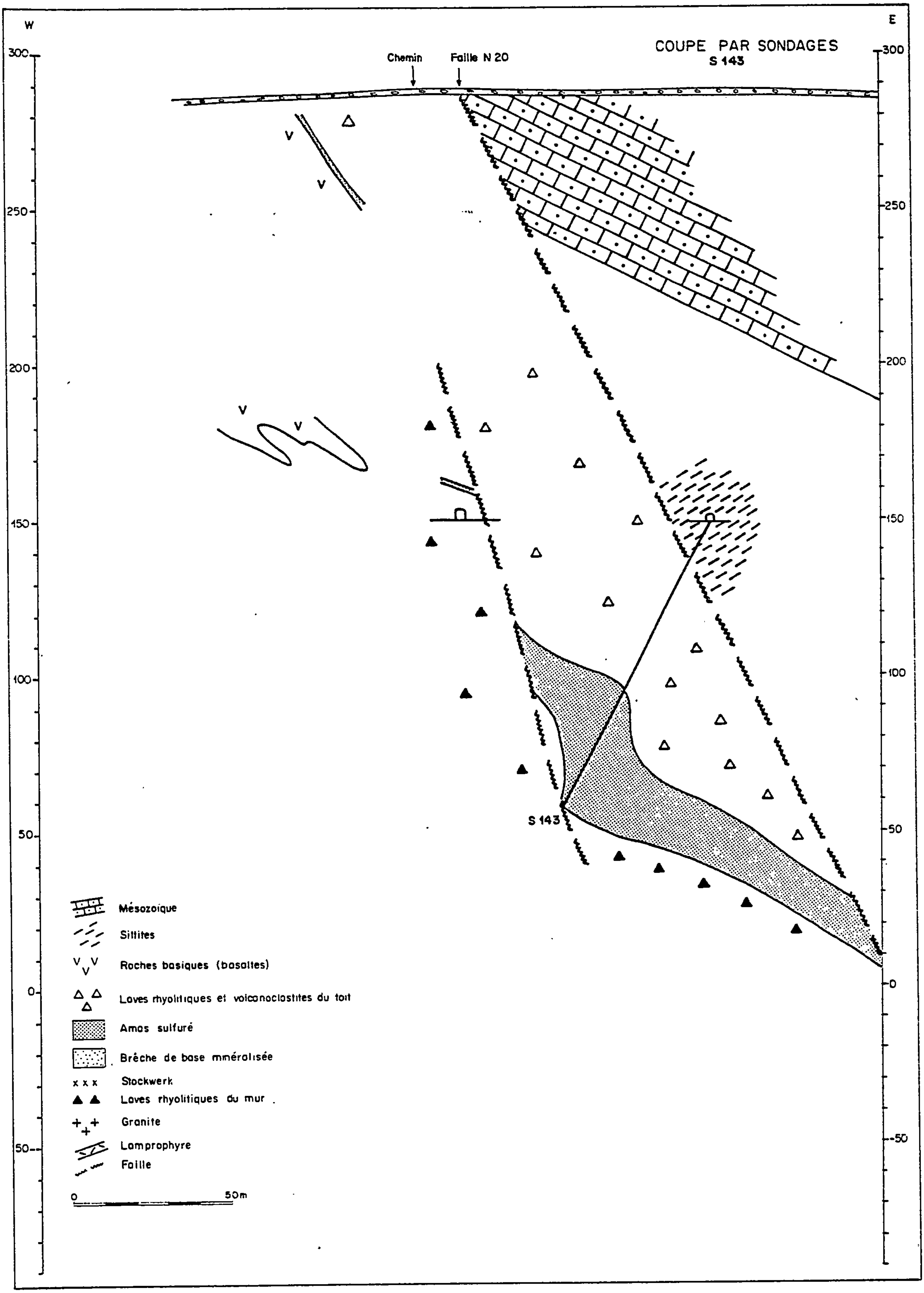
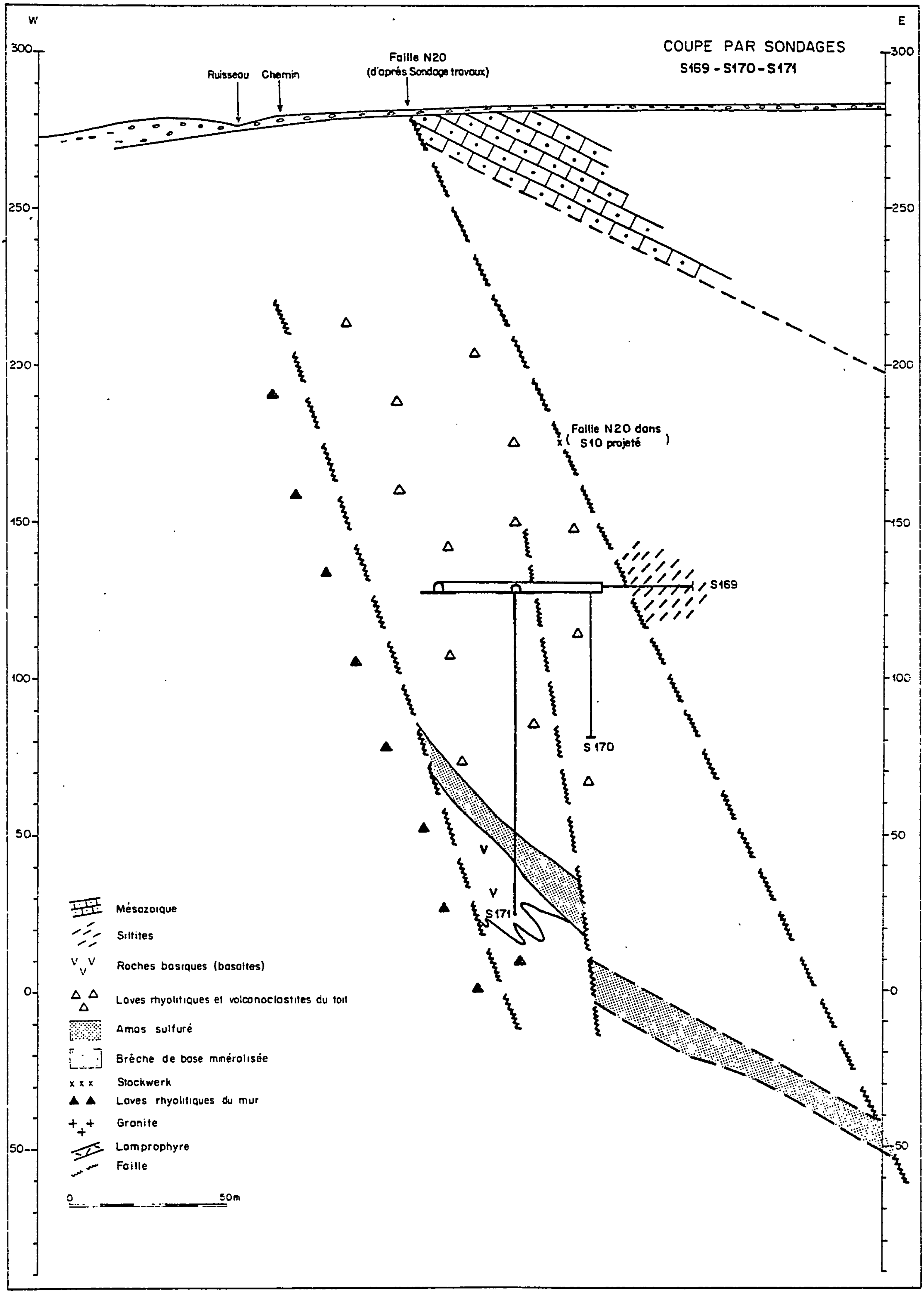
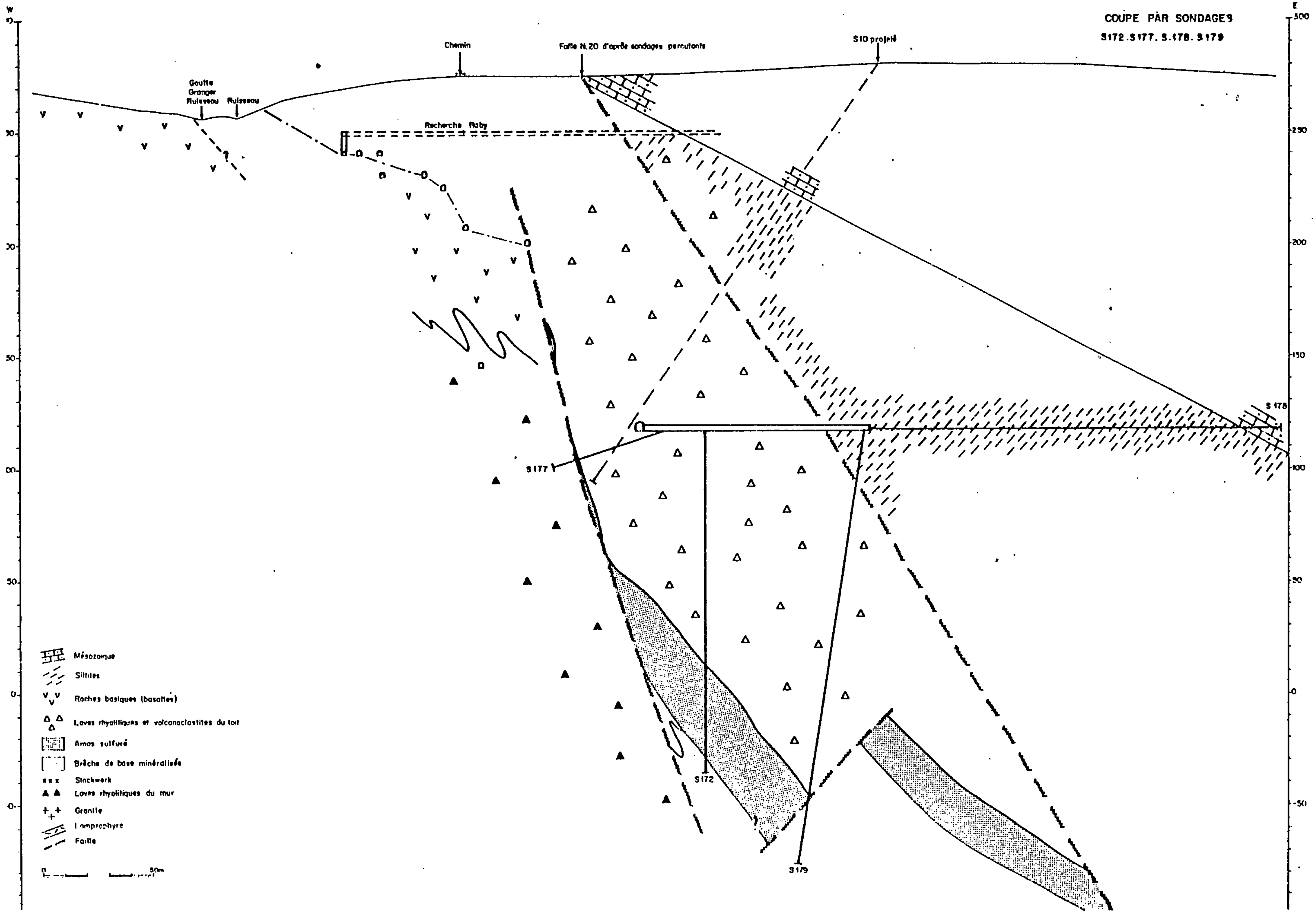
Faille N.20
S124
S159
S166
S158
S157
S142
S 5
S 105
S 110

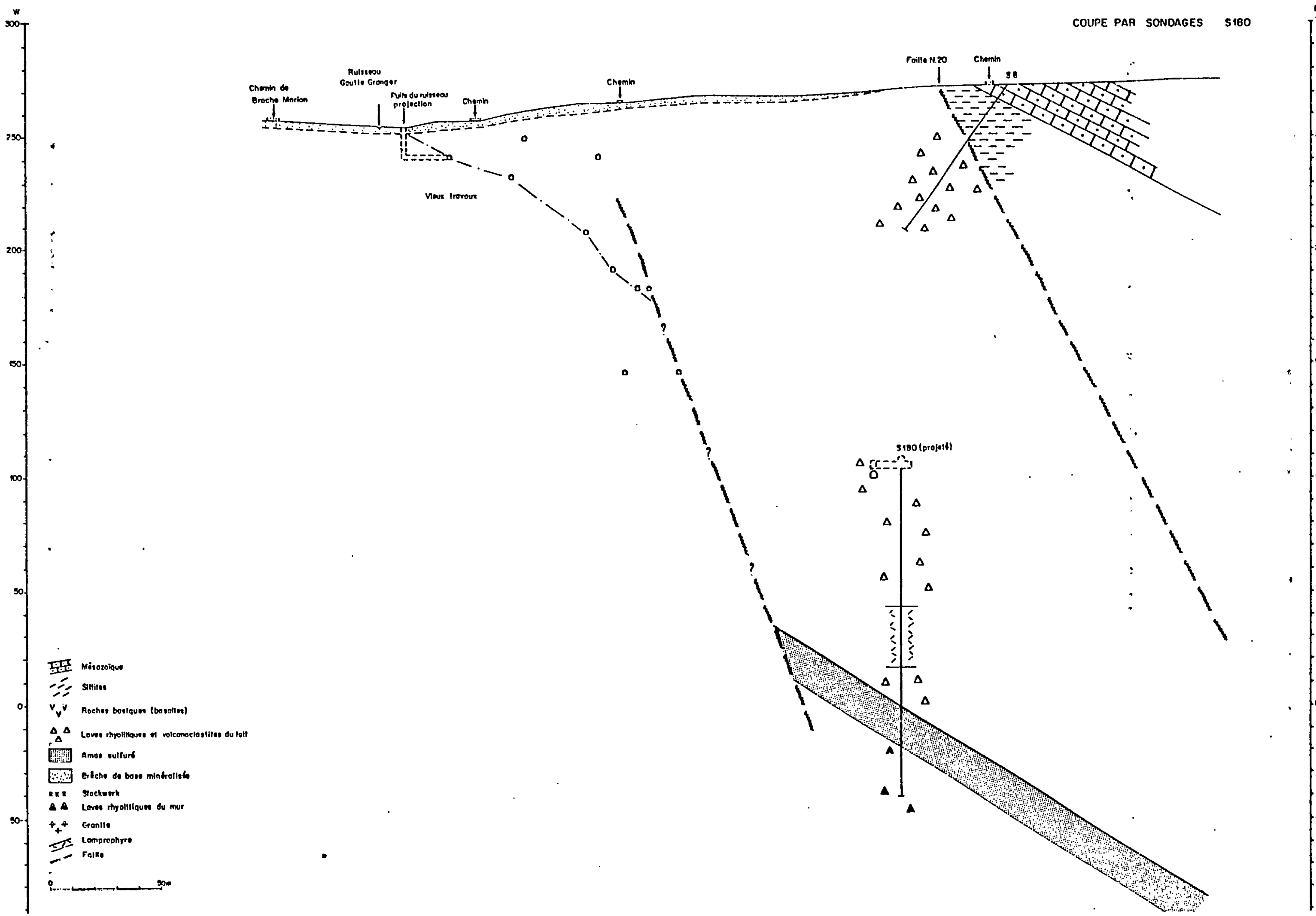

W
E
COUPE PAR SONDAGES
S152.S168.S164.S165
Mésozoique
Siltites
Roches basiques (andésites)
Laves rhyolitiques et volcanoclastites du toit
Amas sulfuré
Brèche de base minéralisée
Stockwerk
Laves rhyolitiques du mur
Granite
Lamprophyre
Faille
0
50m
S152
S168
S164
S165
Faille N.20
300
250
200
150
100
50


W
E
COUPE PAR SONDAGES
S 143
Chemin
Faille N 20
S 143
300
250
200
150
100
50
0
-50
Mésozoique
Siltites
Roches basiques (basaltes)
Laves rhyolitiques et volcanoclastites du toit
Amas sulfuré
Brêche de base minéralisée
Stockwerk
Laves rhyolitiques du mur
Granite
Lamprophyre
Faille
0
50m



W
E
COUPE PAR SONDAGES
S169 - S170 - S171
Faille N20
(d'après Sondage travaux)
Ruisseau
Chemin
Faille N20 dans
S10 projeté
S169
S170
S171
300
250
200
150
100
50
0
-50
Mésozoique
Siltites
Roches basiques (basaltes)
Laves rhyolitiques et volcanoclastites du toit
Amas sulfuré
Brèche de base minéralisée
Stockwerk
Laves rhyolitiques du mur
Granite
Lamprophyre
Faille
0
50m


COUPE PAR SONDAGES
S172.S177.S178.S179
W
E
Chemin
Faille N.20 d'après sondages percutants
S10 projeté
Goutte Granger Ruisseau
Ruisseau
Recherche Raby
S177
S172
S179
S178
300
250
200
150
100
50
0
-50
Mésozoique
Siltites
Roches basiques (basaltes)
Laves rhyolitiques et volcanoclastites du toit
Amas sulfuré
Brêche de base minéralisée
Stockwerk
Laves rhyolitiques du mur
Granite
Lamprophyre
Faille
0
50m


COUPE PAR SONDAGES S180
W
E
300
250
200
150
100
50
0
-50
Chemin de Broche Marion
Ruisseau Goutte Granger
Puits du ruisseau projection
Chemin
Chemin
Faille N.20
Chemin
S8
Vieux travaux
S180 (projeté)
Mésozoïque
Siltites
Roches basiques (basaltes)
Laves rhyolitiques et volcanoclastites du toit
Amas sulfuré
Brèche de base minéralisée
Stockwerk
Laves rhyolitiques du mur
Granite
Lamprophyre
Faille
0
50m

As a rule, only the massive ore contains high Cu an Zn grades; the breccia ore contains at best 1 % Cu and 2 % Zn, but the walls are barren.

| | Tonnage t | Cu % | Pb % | Ag % | Zn % | Pyrite % | Barite % | Waste % |
|---|---|---|---|---|---|---|---|---|
| North Block | | | | | | | | |
| Panel 10 | 348 464 | 1.71 | 0.85 | 27 | 9.00 | 15.04 | 42.78 | 22.32 |
| Panel 20 | 499 977 | 2.77 | 0.21 | 18 | 8.95 | 31.68 | 29.60 | 16.61 |
| Panel 30 | 739 101 | 2.34 | 0.16 | 15 | 7.51 | 45.09 | 22.15 | 14.21 |
| Panel 40 | 331 008 | 1.85 | 0.10 | 18 | 4.91 | 51.10 | 14.93 | 20.93 |
| Sub-total | 1 918 550 | 2.25 | 0.288 | 18.48 | 7.70 | 37.17 | 26.59 | 17.47 |
| South Block | | | | | | | | |
| Panel 50 | 2 496 259 | 2.40 | 0.29 | 23 | 10.89 | 45.18 | 26.12 | 4.57 |
| TOTAL AND AVERAGES | 4 414 809 | 2.34 | 0.29 | 21 | 9.51 | 41.70 | 26.32 | 10.17 |

Anybody can judge for himself that the potential ore in the South Block is very substantial. Overall, the average tonnage of the orebody as already known is 67 t/m$^2$.

## IV - GEOTECHNICAL STUDIES TO DETERMINE THE MINING METHOD

### 1 - PRELIMINARY STUDIES

Preliminary studies comprised:

- a survey of the fracture density of the ground;
- physical and mechanical characterization of the ore and the wall rock;
- measurement of rock behaviour in the experimental rooms by means of extensometers on rods of 1 m to more than 11 m length, placed in the boreholes drilled in the roof of drifts; such drifts were then enlarged to the potential dimensions of future rooms, which allowed gradual measurements of the elastic behaviour of the rock during the enlargement to 8 m diameter;
- numerical modelling, the results of which were calibrated with the measurements of roof displacement of the experimental rooms.

#### *1.1 - Dimensioning of abandoned pillars*

A first approach to the limit-design of pillars in mining method that involves abandoned pillars, shows that for rooms with a width of 6 m and pillars with sides of 5 m width and a slenderness ratio of 2 (h = 10 m), a stability with a theoretical recovery factor close to 80 % is obtained. This mining method was not retained as it does not guarantee long-term stability.

#### *1.2 - Dimensioning of caved stopes*

The calculations led to the design of caved stopes of 60 m width with (theoretical) recovery of 100 % and separated by rib pillars 40 m wide. The lack of flexibility, inherent in this method when the mineralized layer becomes more inclined, and the low overall recovery rate of about 60 %, have led us to abandon this method.

#### *1.3 - Determination of cemented back-fill*

Study of this method rests mainly on knowledge of the mechanical properties of the cemented back-fill. The waste from the mill and in particular its size fraction will be insufficient in quantity, and for this reason a back-fill was tested that consists of barren rock from the mine-development works. Tests covered grain size, required cement quantities, and variable setting times: 54 test pieces of large size were thus tested.

The maximum value of resistance to simple compression over 28 days was obtained for the 0/50 mm grainsize range, with a dosage of 200 kg cement per cubic meter; this resistance is 9.1 MPa on average.

In view of the remaining pore space between the clasts, it can be seen that the mechanical quality of the back-fill could be improved by changing the grainsize range of the fine and coarse particles, while retaining a reasonable cement dosage (100 to 150 kg/m$^3$). Further tests should explore this possibility.

### *1.4 – Stability of the back-fill: the case of undercut-and-fill mining*

As a first approach, the maximum width had to be determined over which the back-fill slab could be left bare during mining of the ore, as it is known that the span can be considerable, especially at intersections.

For a cemented back-fill of the 0/50 mm grainsize range, dosed with 200 kg cement per cubic meter, the slab should have a thickness of 2.5 m for a maximum-allowable span in the range of 8 m, if it is compared to a beam on two simple level posts. Greater spans call for thicker slabs and/or a cemented back-fill with better mechanical properties: for instance, for a 5-m-thick slab consisting of 0/50 mm back-fill dosed with 150 kg cement per cubic meter, the maximum-allowable span is 11.6 m.

### *1.5 – Stability of back-fill and pillars: case of sub-level stoping with transverse room and pillars*

The room width was much more determined by roof constraints than by the ore, which has much better mechanical characteristics. Under the condition that the roof must be left unsupported over very limited areas and should be rapidly reinforced by bolting, grouting, or shot-creting before stowing the cemented back-fill, it seems reasonable to plan for rooms of 8 m width.

If one assumes a roof thickness of 200 m and an undefined surface of the layer:

- with open rooms (without back-fill) the stress loading on ore pillars of the same size will be 11 MPa (as compared to a maximum-allowable resistance to simple compression of the ore of 90 MPa);

- in back-filled rooms in the final plase, the stress loading on the cemented back-fill will be 5.4 MPa.

The range of 5.4 to 11 MPa thus defined, an average value of 8 MPa was retained for further calculations that confirmed the final characteristics of the back-fill, which in any case should be revised downwards. Another possibility lies in reducing the pillar width, if the operator considers this desirable.

As concerns recovery of the pillars while retreating upwards (pillars sandwiched between cemented backfill), it was possible to verify that, in view of the rough interface ore/back-fill, horizontal stresses, and the shear strength of the back-fill, no sliding of the sub-caved ore has to be feared.

# V - ORE-PROCESSING STUDIES

## 1 - MINERALOGICAL STUDIES

From the mineralogical viewpoint, except for microscopic examination of numerous samples that showed the monotony and simplicity of the mineralogical associations, studies on the liberation size were carried out. For a given grinding mesh (e.g. $d_{80}$ = 80 micrometers), the envelope curve of optimal couples of grade and zinc recovery that could be obtained in flotation concentrates, shows the extremely favourable aptitude for good separation by physical means. This is equally true for chalcopyrite, which, together with sphalerite, forms the intrinsic value of the ore.

Furthermore, several analyses were made with a microprobe on the various minerals present in the orebody; the results confirm the classical type of the mineralization; the sulphides in particular do not contain any impurity of note, such as arsenides or antimonides, that could affect their commercial value.

## 2 - ORE-PROCESSING STUDIES

### *2.1 - Bench tests*

From the viewpoint of ore-processing, 270 different laboratory tests were carried on different ore types from the Chessy orebody. These tests successively had the following objectives:

- obtaining a good-quality Cu and Zn concentrate with high recovery;
- elimination of pyrite to obtain a barite concentrate (chemical-grade);
- research into the impact of the ore variability, which is only felt in the need for different grinding conditions; the flotation method remains the same.

The methods used were quite standard, with in particular the introduction of an aeration phase of the pulp in order to depress the pyrite.

Desliming of the pulp before floating of the pyrite is foreseen in the adopted flow-sheet; its main aim is to eliminate the pyrite fines, in order to produce as pure as possible a barite concentrate.

Furthermore, a certain nunber of additional objectives were aimed at:

- elimination of the lead that is occasionally present in the copper concentrate;
- the effect of water recycling;
- the effect of aging of the ore;
- bleaching of the barite concentrate.

*2.2 - Semi-industrial pilot-plant tests*

The test work in the pilot plant (through-put of 500 kg/hr) was carried out to base the feasibility study on continuous ore-processing tests that integrated the recycling steps, which are difficult to simulate on the laboratory scale. These tests led to the obtention of four concentrates from two different ore types, one dominated by barite and the other corresponding to the pyrite pole; the concentrates obtained were for chalcopyrite, sphalerite, pyrite and barite.

The pilot-testing took about 400 hours, divided in three continuous periods of 120 hours and one period of 5 times 8 hours. Particularly detailed analysis of the pilot process was made possible by systematic and frequent sampling, using an almost instantaneous XRF instrument and calculating coherent material-balances by means of a specially adapted computer program.

The basic data for the present study were in large part provided by this pilot-testing work.

*2.3 - The results*

The main conclusion is that the ore is particularly suitable to obtain high-quality concentrates with good recovery, by means of a flexible and well-proven process. Very good results were obtained after a very short (2 days) period of tuning the pilot plant. Furthermore, a sudden change in ore type during the pilot testing did only slightly disturb the operations.

Reagent consumption dropped when compared with the bench testing and can be further optimized.

The balance of the stable periods during pilot testing corresponds to the following results:

| | Baritic Ore | Piritic Ore |
|---|---|---|
| Zinc | | |
| . Recovery | 94 % | 93 % |
| . Zn grade | 56 % | 56 % |
| Copper | | |
| . Recovery | 92 % | 93,5 % |
| . Cu grade | 29 % | 27 % |

The limiting factor in zinc recovery, is the zinc content of the copper concentrate, which corresponds to about 4 % of the zinc recovery; schematically, it seems that half the zinc is free and the other half is present as mineralogical mixtures.

Desliming, which has not been optimized as yet, causes a barite loss of 10-15 % and a pyrite loss in the range of 8 to 12 %. It can be reasonably expected that such losses can be reduced by at least 5 %, for both barite and pyrite.

the pyrite and barite concentrates grade, respectively, 98 to 99 % $FeS_2$ and $BaSO_4$ and the recovery factors are both higher than 90 % in the flotation stage that follows upon desliming.

# VI - WATER RESOURCES

## 1 - WATER REQUIREMENTS

For a polymetallic-flotation plant, valuable experience has been gathered by Canadian and Scandinavian companies. Such experience derives from long-term tests at true scale, on plants that are well "run-in"; recycling of more than 25 % of the water shows up problems in recovery factors and grades of the concentrates, after 1 to 2 months.

For this reason, the Chessy project was planned around recyling of 25 % of the water, but once the flotation section will be operational it can be attempted to increase the recycling rate. At the planned recycling rate, the plant will need 80 $m^3$/hr of new water for a planned through-put of 1,080 t/day.

## 2 - DEWATERING

While excavating the exploration drifts and other underground works, BRGM engineers noted that water inflow showed little variation, oscillating between 10 and 20 $m^3$/hr. Such water, which is rarely acid, could satisfy at least part of the water requirements of the mill.

## 3 - SPRINGS

The existing springs in a radius of several kilometers around the orebody, have been inventoried and their flow was measured. They issue mostly from sedimentary-rock units, such as the Triassic sandstone and the Early Jurassic (Liassic) limestone, and not from the basement rocks, whether they be granites or volcanic.

Flow rates of the most important springs vary between 10 an 30 $m^3$/hr; certain of these springs have been tapped and those that remain untapped lie at several kilometers from the mine site.

## 4 - THE AZERGUES RIVER

This river, which is a torrent, could theoretically supply the required water as the minimum daily flow rate is 290 $m^3$/hr. However, close to the future mine site lies a dyeing factory, the "Etablissements Mathelin", which has taken water from the Azergues for over a century at a constant rate of 125 to 145 $m^3$/hr. The effluents from this factory contain organic and mineral products that, not withstanding the treatment the water is subjected to, render this water useless for a flotation plant.

## 5 - THE TRIASSIC AQUIFER

Triassic sandstone, which hardly occurs in outcrop, rests directly on the Paleozoic basement; its thickness varies from 30 to 100 m. The sandstone is coarse-grained, commonly poorly consolidated, and shows in its upper part intercalations of limestone and marl lenses. The sandstone is repeatedly porous and permeable, and has shown good water yield from several exploration boreholes, in particular hole n° 178 with a flow of 25 $m^3$/hr.

This aquifer is by far the most promising, as one or a few drilled wells should provide all water requirements of mine and mill. According to the BRGM hydrogeologists, any connection with springs in the neighbourhood of the mine site is improbable.

# VII - CHARACTERISTICS OF THE CONCENTRATES

## 1 - PHYSICAL CHARACTERISTICS

### *1.1 - Moisture content*

Filtration tests have shown that with the most classical filtration equipment (a simple drum filter), moisture contents (related to dry weight) will be as follows:

- copper concentrate : 9-10 %
- zinc concentrate : 11 %
- pyrite concentrate : 8 %
- barite concentrate : 6 %

### *1.2 - Conditioning*

Storage, loading and transport are conceived as in bulk, but this does not exclude a later possibility of packaging in bags, on pallets, or in tanks, for the higher-value products, such as dried and micronized barite.

### *1.3 - Grain size*

In general, the grainsize of the concentrates is around 70 to 90 micrometers.

## 2 - CHEMICAL CHARACTERISTICS

### *2.1 - Copper concentrate*

As for grades, one can note:

- Cu (28-29 %)
- Pb (1.5 %)
- Zn (4 %)
- Ag (60-80 g/t)
- no penalizing impurities.

### *2.2 - Zinc concentrate*

Grades are:

- Zn (56-57 %): above the average (53 % Zn);
- Fe (3-4 %): low grade as compared to some marmatite concentrates that reach 8-12 % Fe;

- Cd (0.2-0.3 %): little or no value;
- no valorizing elements, such as Ag and Ge, nor any penalizing elements, such as Ba or Bi.

### *2.3 - Pyrite concentrate*

Grades are:

- S (51-52 %): very high grade as compared to other concentrates, which typically contain 42-49 % S;
- Cu (0.03 %); Zn (0.34 %), Pb (< 0.005 %): residual base metals with a very low threshold;
- As, Sb, Hg, Tl are not penalizing.

This is, in fact, a concentrate of exceptional quality.

### *2.4 - Barite concentrate*

Chemical characteristics are:

- $BaSO_4$ (97.1-98.3 %): very high grade;
- $SrSO_4$ (1.4-1.5 %): to be watched;
- $SiO_2 + Fe_2O_3 + Al_2O_3$ (0.4-0.6 %): very much below the standard;
- F and heavy-metal traces not penalizing.

Physical characteristics are:

- density: 4.41 $t/m^3$, which is much above that of a standard concentrate (4.2 $t/m^3$);
- whiteness: can be improved by appropriate processing for specific markets, such as the paint industry.

In general, this can be called a concentrate of very good quality.

### *2.5 - Lead concentrate*

The grades of this concentrate have not yet been optimized, but are in preliminary fashion:

- Pb (40-65 %),
- Cu (4-20 %),
- Ag (600-1 000 g/t).

# *CHESSY*

# *THE MINING PROJECT*

********

## I - EVALUATION OF MINEABLE RESERVES

The thin mineralized veins were diluted, in order to obtain a true thickness of 2.50 m, which is considered to be the minimum width that can be mined with the equipment as envisaged for Chessy.

From experience, and in view of the recommended mining method that calls for cemented back-fill, the following recovery rates were applied for each polygon:

| Dip of the ore in degrees | Recovery factor |
|---|---|
| 0-15 | 90 % |
| 16-44 | 80 % |
| 45-65 | 90 % |
| 66-90 | 95 % |

The proportion of waste mined varies according to dip and thickness of the mineralized layer, and was defined for each case by simple geometry, for a drift 5 m wide and 4.5 m high.

| Dip | Dilution % in volume for a true thickness of | | | | Dilution % in weight for a true thickness of | | | |
|---|---|---|---|---|---|---|---|---|
| in degrees | 2.5-4.5 m | 4.5-10 m | 10-15 m | 15-20 m and over | 2.5-4.5 m | 4.5-10 m | 10-15 m | 15-20 m and over |
| 0 - 15 | 11 | 6 | 3 | 2 | 8 | 4 | 2 | 2 |
| 16 - 44 | 39 | 19 | 11 | 8 | 25 | 12 | 7 | 5 |
| 45 - 65 | 17 | 8 | 5 | 3 | 11 | 5 | 3 | 2 |
| 66 - 90 | 11 | 6 | 3 | 2 | 8 | 4 | 2 | 2 |

After application of recovery and dilution factors, the extracted ore tonnage corresponds to 85 % of the in-situ reserves.

For each polygon, the pit-head revenue was calculated for one tonne ore at April 1986 prices. In this way, eight polygons could be eliminated as their pit-head value was less than total operating costs (mine and mill), which, at the time, were evaluated at 270 FF/ton. This operation led to a lowering of the mineable tonnage, but an increase of the average grade.

The mineable tonnage, whose quality and quantity will be sufficient to feed the flotation plant, is as follows.

| | Tonnage t | Cu % | Pb % | Ag g/t | Zn % | Pyrite % | Barite % | Waste % |
|---|---|---|---|---|---|---|---|---|
| North Block | | | | | | | | |
| Panel 10 | 284 291 | 1.59 | 0.89 | 27 | 8.98 | 14.24 | 43.32 | 22.70 |
| Panel 20 | 447 795 | 2.63 | 0.21 | 17 | 8.59 | 29.55 | 28.29 | 20.91 |
| Panel 30 | 614 400 | 2.28 | 0.15 | 14 | 7.44 | 44.08 | 21.98 | 15.55 |
| Panel 40 | 212 221 | 1.87 | 0.11 | 18 | 5.,13 | 44.52 | 16.17 | 25.75 |
| Sub-total | 1 560 707 | 2.20 | 0.30 | 18 | 7.74 | 34.53 | 26.88 | 19.78 |
| South Block | | | | | | | | |
| Panel 50 | 2 190 067 | 2.27 | 0.29 | 22 | 10.68 | 42.58 | 25.73 | 8.11 |
| TOTAL AND AVERAGES | 3 750 774 | 2.24 | 0.29 | 20 | 9.45 | 39.24 | 26.21 | 12.97 |

One should keep in mind that to this tonnage will be added the potential reserves, evaluated at present at 2 million tons.

## II - CHOICE OF RATE AND ORGANIZATION OF WORK

### 1 - DETERMINING THE LEGAL WORKING TIME

According to the mining regulations, the number of paid eight-hour shifts is distributed as follows (the example of the year 1986 is taken):

| | |
|---|---|
| - calendar days | 365 |
| - sundays and saturdays | - 104 |
| - free days other than weekends | - 10 |
| - days making up for lost time (fortieth hour) | - 6 |
| Number of days with paid shifts | 245 |
| Paid holidays 25 working days | - 25 |
| Number of shifts worked per year | 220 |

### 2 - PRODUCTION CAPACITY OF THE MINE

In view of the length of the orebody at a given level, as well as of the proposed mining method, a production rate was envisaged of 1,500 t/d or 330,000 t/year for 220 working days,

In fact, the impact of the drying/setting time of the cemented back-fill alternating with each mining phase, limits the number of productive faces.

Daily production will come from two eight-hour shifts, the third shift being used for specific operations such as back-filling.

### 3 - PRODUCTION CAPACITY OF THE MILL

In order to reduce the capital investment in the mill on the one hand and to avoid the production losses inherent in stopping and starting-up on the other hand, it is proposed to operate the mill on a continuous basis for 321 days per year.

The working time will be covered by five teams; if one assumes an availability rate of 95 %, the mill should be designed for a capacity of 1,080 t/d.

### 4 - WORKING RATE OF ANCILLARY SERVICE DEPARTMENTS

The working rate of the technical, administrative, and social services should be based on a five-day week, to the tune of 220 days per year.

## 5 - START-UP OF PRODUCTION

It was assumed that nominal production will be reached after one year; during this first year it was supposed that all production costs will have to be supported, but with 75 % of the corresponding nominal revenues.

## III - THE MINE

### 1 - CONCEPTION OF THE MINING METHOD

The mining method was established in collaboration with the "Ecole des Mines" at Paris; it takes into account the absolute constraints that were imposed after discussions with the Administration:

- avoid all risks that at the medium or long term a sudden and uncontrolled caving-in of the surface could occur;
- not to hinder the wine-growing activities of the region;
- respect the environement (landscape, noise, pollution, etc.).

On the conceptual plane, a "made-to-measure" mining method was sought for:

- which would be applicable in the thickest parts of the mineralized layer;
- which would ensure roof stability, by baring the roof as little as possible and over as short periods as possible;
- which would ensure an acceptable recovery factor with minimum dilution;
- which would allow slashing with high productivity and thus low cost, as soon as the difficulties of roof strength have been overcome and mining takes place within the ore, which has excellent strength.

To begin with, all possibly suitable methods were considered and several of these were studied in detail, which led to the selection of cut-and-fill sub-level stoping, with alternating mining and back-filling cycles.

The principle of the method consists in subdividing the orebody in units of about 50 m length, with access at each end and every eight meters of vertical level. One begins by drifting along the footwall in the ore and driving parallel cross-cuts from the drift to the hanging wall of the ore. After opening a slot-raise between these cross-cuts and those of the sublevel above, one continues with slashing by retreating from hanging wall to foot-wall, all the while taking care that the pillars will be of the same dimension as the room. After stowing the rooms, the pillars will be mined in their turn and back-filled, following the same mining method.

Obviously, where the thickness of the ore is around 5 meters, if will suffice to open just a longitudinal room, which will be mined as cut-and-fill or by means of sub-level stoping followed by a back-filling operation.

In this manner, 40 % of the ore tonnage will be produced by drifting along the footwall and driving the cross-cuts, and 60 % will be mined by slot-raising and slashing between sub-levels, and by mining the rib pillars.

## 2 - THE MINE LAY-OUT

The ore, stoped over 8 m width, will be hauled by diesel LHD to an ore chute with landing stations all 24 vertical meters. The ore and waste are drawn, crushed underground in a primary crusher, and then transported to surface by means of a conveyor belt that will be installed in a special decline.

Another decline from surface, with a larger cross-section, will serve for vehicle traffic, personnel transport, as well as a ventilation "shaft".

### *2.1 - Size of the drifts*

In order to reduce production costs to a maximum, mining should be mechanized as much as possible, using for instance diesel LHDs with a 7 to 10 tonne capacity, which should be compatible with roof-support and ground-strength conditions.

The drifts in waste rock should be 15 to 20 $m^2$ in section, the latter dimension having been adopted for the time being. It is already known that 20-$m^2$-section drifts in the ore will pose no particular roof-support problems.

### *2.2 - Subdividing the orebody*

The haulage distance for LHDs should not exceed 200-250 m for reasons of efficiency. this means that an ore landing station can serve an ore block of about 400 m; consequently the orebody was divided in two blocks:

- the North Block, at relatively shallow depth and well-explored by means of drifts and boreholes, which should be mined first;
- the South Block, which is deeper and less-well explored and should be mined second.

### *2.3 - Planning of the lay-out*

A conscious effort was made to site the maximum of development works in the basaltic units of the foot-wall, which are homogeneous and show excellent geotechnical conditions, and to drift as little as possible in the rhyolitic tuffs of the foot-wall that are more fractured.

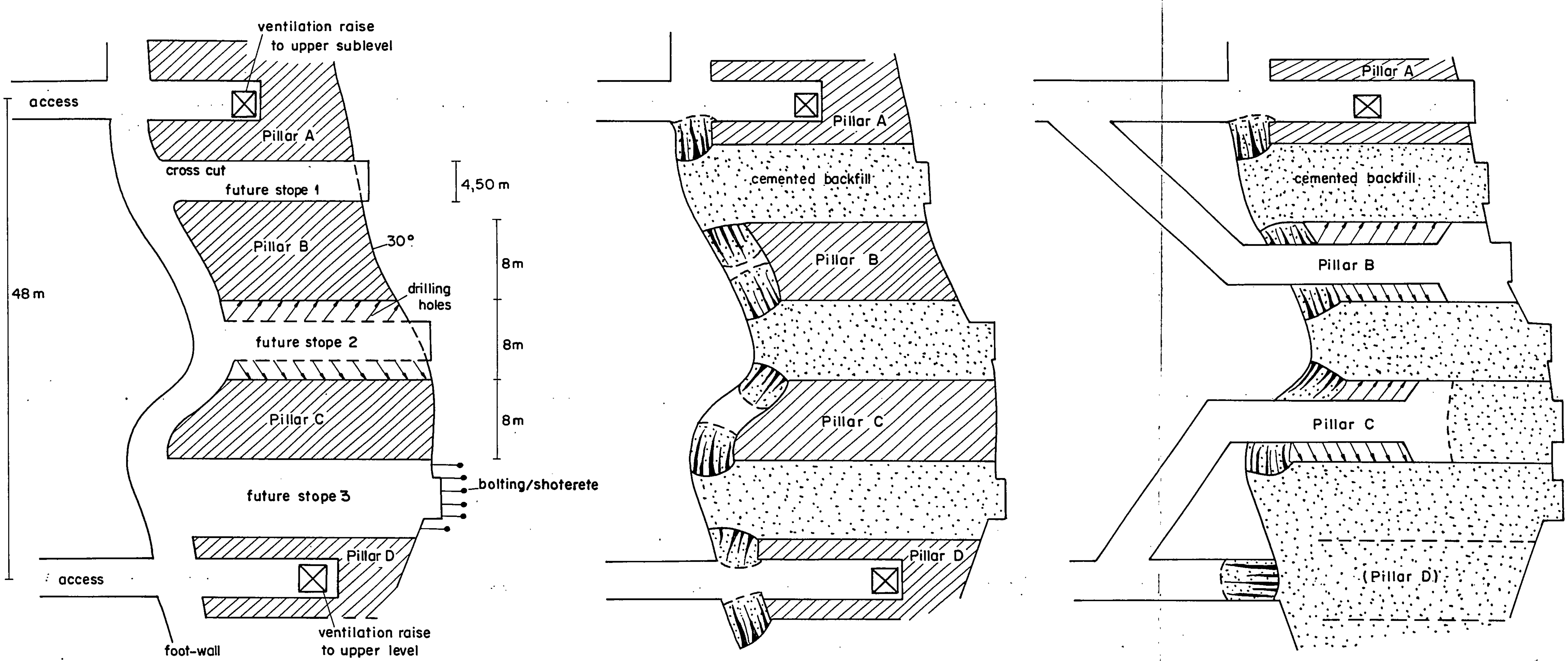
### *2.4 - Haulage*

In the North Block, several types of access (from the landing-station exit to the stope) were studied and designed to reach the ore.

CHESSY

Mining Method

Plan




PHASE 1

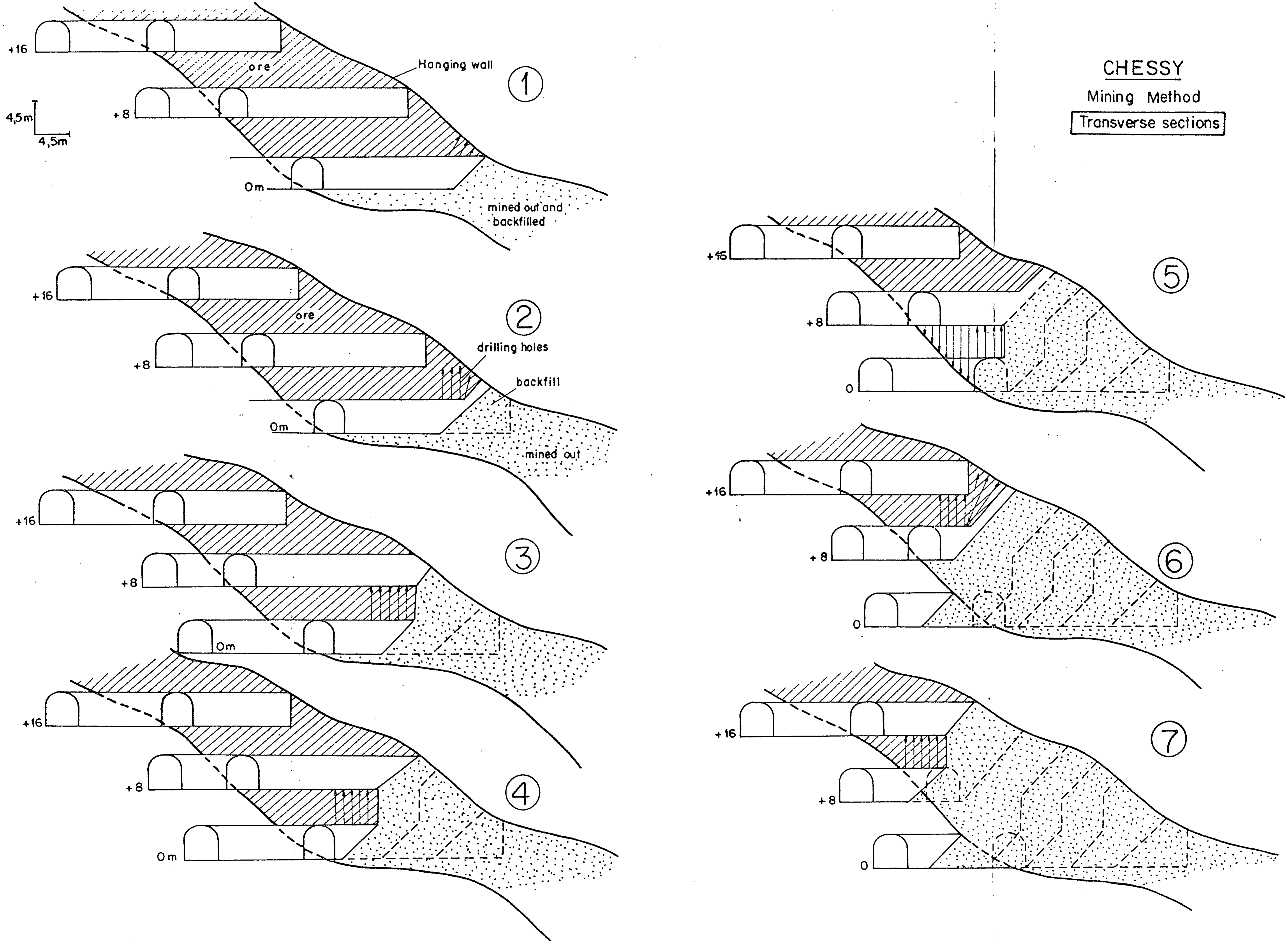
Preliminary stoping

PHASE 2

PHASE 3

Pillar recovery

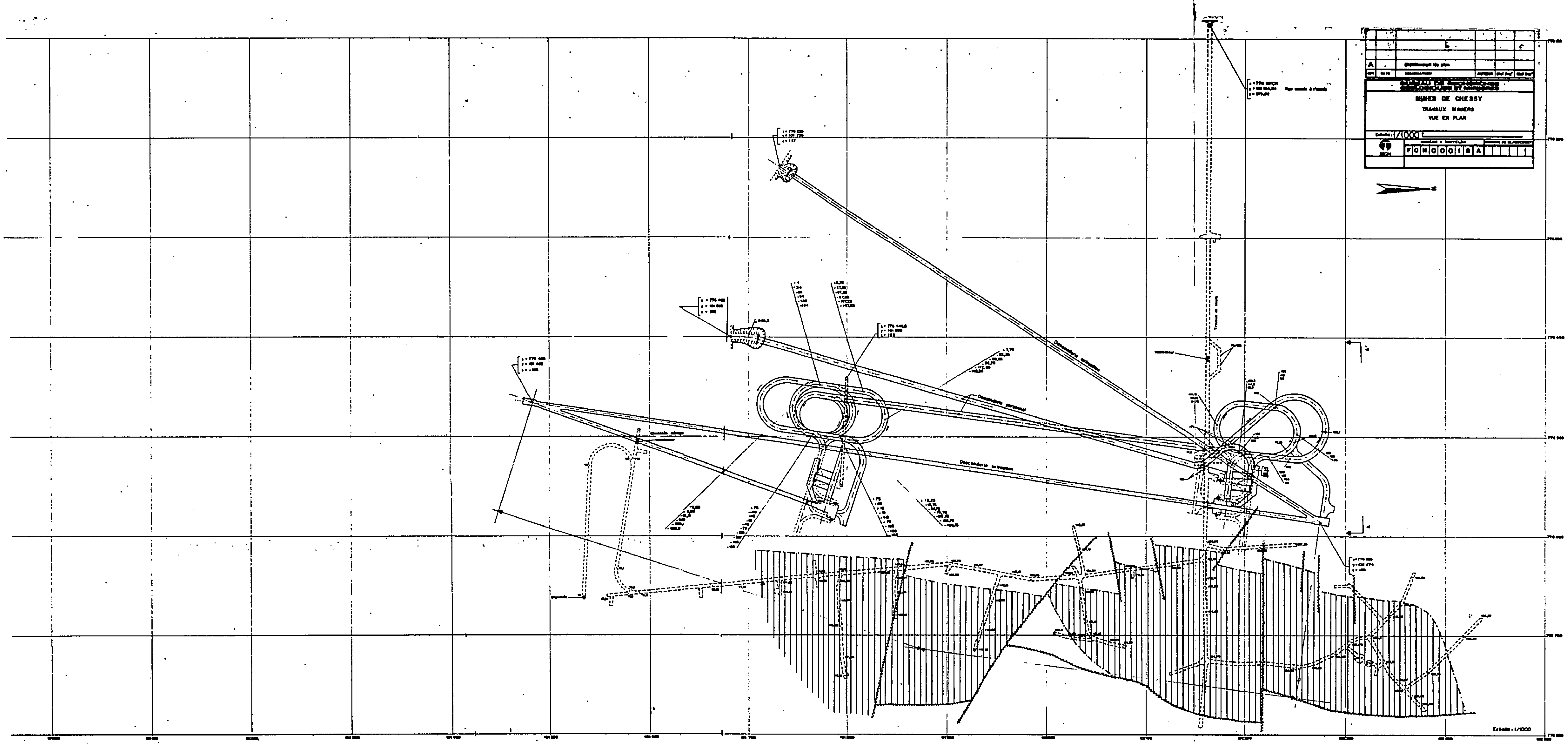
2




CHESSY

Mining Method

Transverse sections


MINES DE CHESSY
TRAVAUX MINIERS
VUE EN PLAN
Echelle : 1/1000
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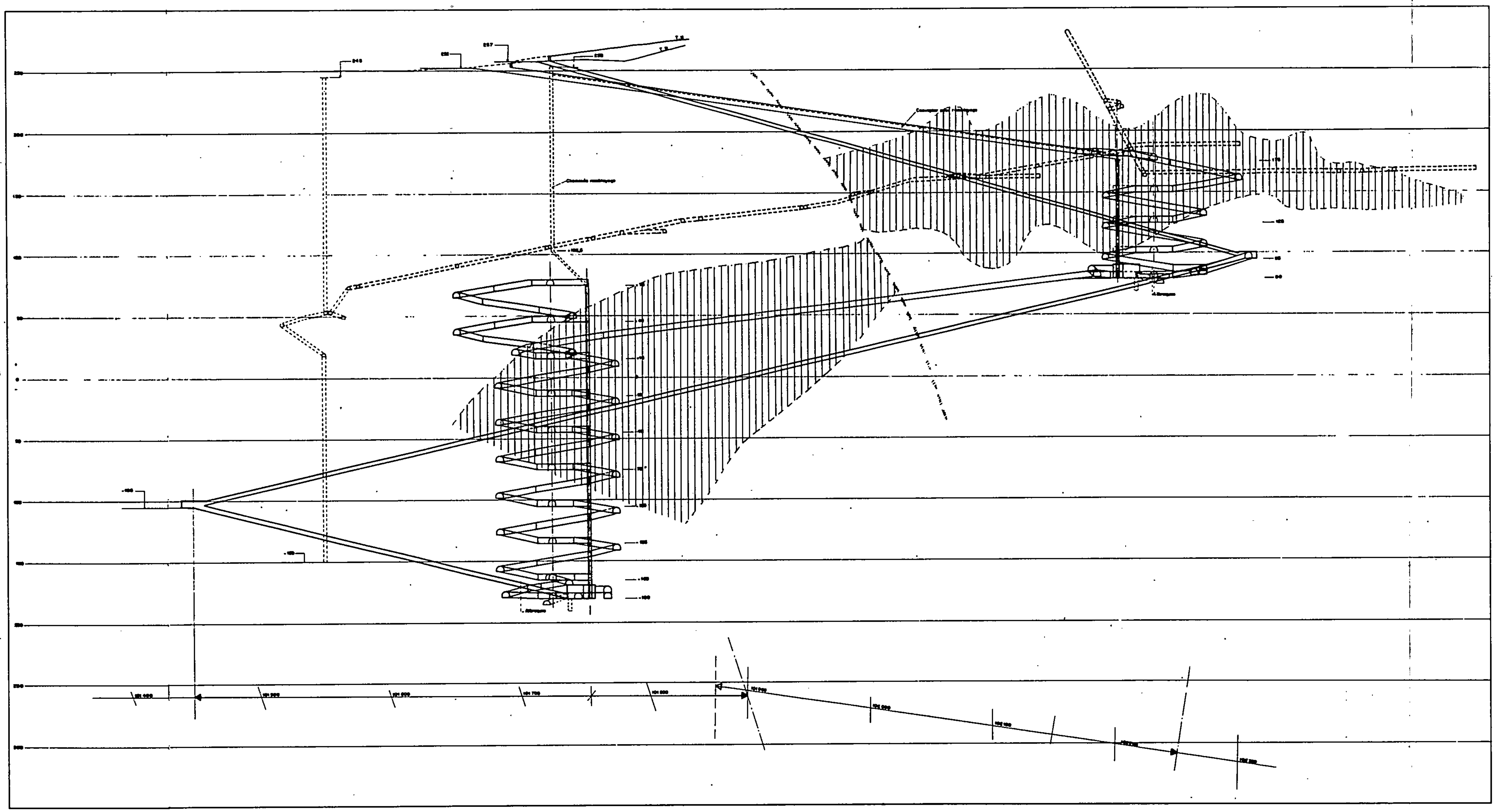

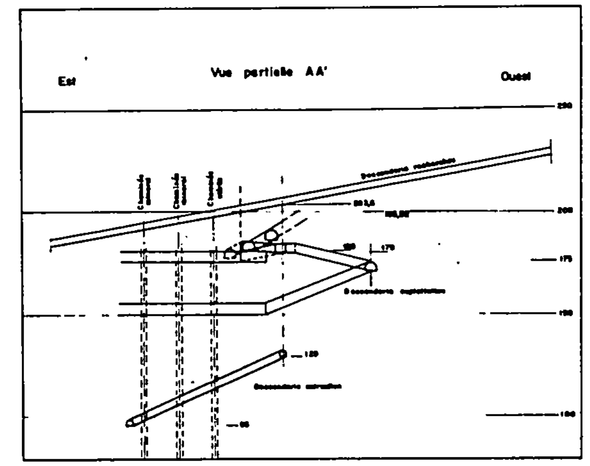


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BUREAU DE RECHERCHES GÉOLOGIQUES ET MINIÈRES

MINES DE CHESSY

TRAVAUX MINIERS

VUE FRONTALE

Echelle 1/1000

| NUMERO A RAPPELER | NUMERO DE CLASSEMENT |
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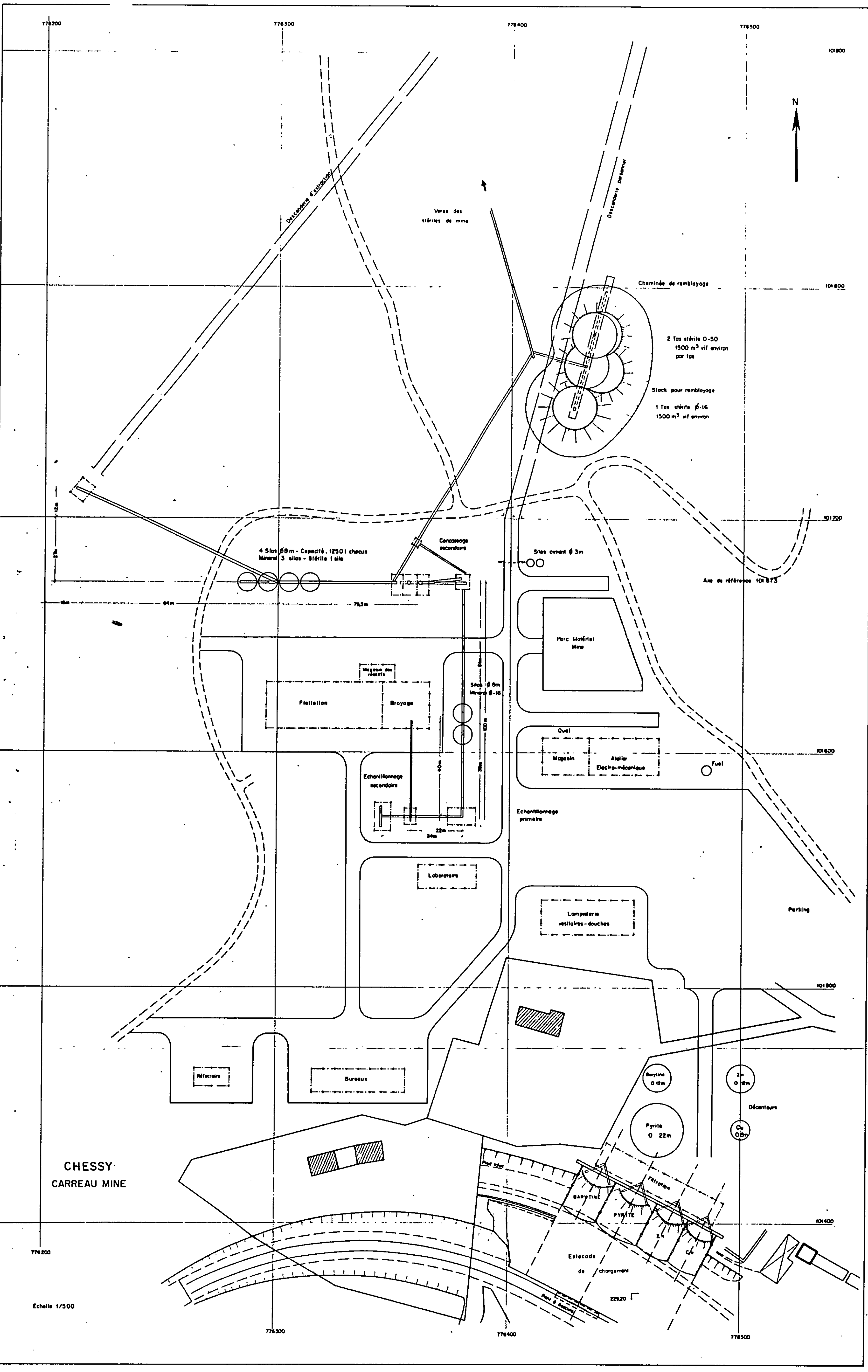


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| REV | DATE | DESIGNATION | AUTEUR | Chef Proj' | Chef Dep' |

BUREAU DE RECHERCHES
GÉOLOGIQUES ET MINIÈRES

CHESSY

PLAN DE MASSE

Echelle 1/500

NUMERO A RAPPELER: J O U 0 0 0 0 5 A

NUMERO DE CLASSEMENT

For the sub-levels, which are spaced every eight vertical meters, in principle 0.0027 m of drift in waste rock are necessary for each tonne of run-of-mine ore. In practice, this value is raised by 30 % to take into account any unforeseen cross-cuts. For a production of 1,500 t/d, one must therefore reckon with 5.20 m drifting in waste rock per day, which, at a maximum cross-section of 20 $m^2$ and a specific gravity of 2.7 $t/m^3$, represents 0.19 tonne for each tonne of ore.

It is estimated that 25 % of the access drifts will be of good strength (bolted and protected with wire netting) and 75 % will be of poor strength (shotcrete or grout over wiremesh).

### *2.5 - Landing stations and orechute*

For efficiency reasons, landing stations are spaced each 24 m vertical meters, each in connection with the main spiralling decline. Two vertical chutes, spaced at 17.40 m, are planned, one for the ore and the other for waste. These chutes will be drilled by raise borer of 2.40 m diameter. To avoid any clogging, each tipping station will be equipped with a screen with a 0.40 × 0.40 m mesh. The blocks that remain on the screen will be broken by means of a mobile hydraulic breaker.

### *2.6 - Underground crushing*

For the North Block, the lowest landing station will be at the +95 level and the crusher wil thus be installed on the +80 level. This primary crushing with a 250 t/hr capacity is justified by the fact that the grainsize should be reduced to 100-150 mm for the material to be transported to surface by conveyor belt.

### *2.7 - Conveyor-belt transport*

A comparative study was made of the various haulage means (truck, skip and conveyor belt); the conveyor-belt solution is a well-proven one for relatively-shallow orebodies (to 300 m depth). Its use is marked by great flexibility, it can be lengthened at will, and it allows spreading in time of capital investment costs: for these reasons the conveyor-belt solution was retained. The recent appearance on the market of electric haulage trucks of 50-tonne capacity, such as the Kiruna Electric, may invalidate some of the more traditional haulage solutions and should be further studied.

### *2.8 - The access decline*

This decline, which will serve for personnel transport and vehicle movements, presents a free cross-section of 20 $m^2$ and a slope of 15 % (to 18 % in certain sections). Excavated in the foot-wall basalt, it links the mine portal (252 level) with the upper landing station on the 175 m level of the North Block, to descend as deep as the crushing station on the +80 level. The decline was designed to be extended towards the South Block.

### 2.9 - *Ventilation*

The access decline doubles as ventilation "shaft" and fresh air is further brought in through a raise at the northern end of the North Block. The work faces will be ventilated through the landing stations and the main cross-cuts. Air then ascends through short raises opened in the axes of certain ore pillars between the sub-levels, and is collected in the upper part of the mine by drifts of 9 to 11 $m^2$ section.

The used air is extracted:

- through the 9-11 $m^2$ section BRGM decline in which a fan is mounted;
- through the end of the BRGM southern decline, extended to the west until the basalt and connected to surface by a 195-m-long raise (from the +52 to the +247 level), which is called the "South Raise";
- through the crushing room and the haulage decline, to get rid of dust and avoid entry of smoke in case one of the belts catches fire.

Air circulation, calculated on the basis of 30 l/s/hp installed, or 50 l/s/hp used, will be about 100 $m^3$/h.

### 2.10 - *Dewatering*

The water inflow rates recorded until present, hardly ever went beyond 20 $m^3$/hr, even before plugging those boreholes that caused the strongest inflow.

The planned dewatering installations cover a pumping rate of 200 $m^3$/hr, with a possibility to extend this to 300 $m^3$/hr.

## 3 - BACK-FILLING

### 3.1 - *Definition of requirements*

For a production rate of 1,500 t/d run-of-mine ore, 360 $m^3$/d of back-fill should be stowed.

### 3.2 - *Back-fill material available on site*

- **Waste rejected by the mill**

In the knowledge that the mill will have to reject on average 20 % of the ROM ore as waste, the usable quantity (grainsize above 50 micrometers) only represents 7 % of ROM, or 20,000 tonnes per year, which is very much insufficient.

**- Waste rock from mine-infrastructure work**

Most of the waste rock that comes from infrastructure work on the North Block, will be produced before the mine starts production and is reserved for construction of the tailings dam.

On the other hand, during production, the infrastructure development work related to opening-up the South Block will generate a large tonnage of waste rock.

**- Waste rock from mine-development work**

Such waste rock will be produced at a daily rate of 0.19 tonnes waste per tonne ROM, or 285 t/d and 63,000 t/y. Most of this is basalt and rhyolite from the foot-wall, which, after primary crushing underground, will have a grainsize from 0-150 mm.

**- "Undefined" waste**

A provision has been made for rounds of unforeseen waste, such as rounds of too-diluted ore, dykes to be traversed, etc.

**- "Dilution" waste**

The possibility has been foreseen, to place in the mill, above the comminution circuit, a "gravimetric trap" (such as jigs), destined to separate a major part of the diluting material from the ore and to regulate the grade and hardness of the ore to be processed. The grainsize of such waste material would be 0-20 mm.

### *3.3 - Mechanical characteristics of the back-fill*

The first tests to manufacture back-fill (54 test slabs) indicate the following strengths after 28 days:

| | Rc |
|---|---|
| - 0-50 mm grainsize with 150 kg/m$^3$ cement | 4.5 MPa |
| - 0-16 mm grainsize with 150 kg/m$^3$ cement | 4.6 MPa |
| - 3-16 mm grainsize with 150 kg/m$^3$ cement, to which 10 % mill waste (50-400 micrometers) has been added | 5.6 MPa. |

The objective to be reached is a back-fill whose strength after 90 days is Rc = 8-10 PMa. In view of the first results and the progress that, undoubtely, can be made by fine-tuning the grainsize of the aggregate, the objective of dosage with 100 kg/m$^3$ cement must be realistic.

### 3.4 – *Waste-rock requirements*

One $m^3$ of back-fill (measured density 2 $t/m^3$) contains:

1,780 kg waste-rock (all grainsizes combined)
100 kg cement
120 kg water

———

2 000 kg total weight.

Waste-rock requirements are thus 360 × 1,780 = 641 t/d, or 141,000 t/yr. Mine development work can provide 44 % of these requirements; the balance might have to come from a small quarry to be opened in the upper part of the portal area (the incised part of the "La Goutte Grange" valley).

### 3.5 – *Stowing the back-fill*

The solution adopted, is to stow the back-fill by means of projection, successfully used by Metallgesellschaft in its Meggen mine. This technique presents great advantages in flexibility over the traditional pumped concrete, as projection can go as high as 8 m and, among other points, presents excellent adherence to and strengthening of the roof at the far ends of the excavated rooms.

### 3.6 – *Hauling back-fill material*

Waste from the mine, with a grainsize of 0-150 mm, is first stored in an 800-t silo and then passed through the secondary and tertiary crushing circuits to arrive at the desired grainsizes of 0-50 mm and 0-16 mm. This material is then sent by conveyor belt to the back-fill storage site that is close to the concrete factory.

### 3.7 – *Cement supply*

the test slabs confirmed that the Val d'Azergues cement works, situated at 4 km from Chessy, sell a cement of excellent quality.

### 3.8 – *Fabricating the cemented back-fill*

The cemented back-fill will be prepared in concrete plant at surface, close to the mine portal. In this way quality control is guaranteed, but its operation can be confided to a sub-contractor, if so desired.

### *3.9 - Transporting and stowing the back-fill*

The principle, which is entirely standard, consists of sending the back-fill down the mine by means of a borehole that allows the filling of 9-$m^3$ trucks, fitted with a scraper blade and, at the rear, with high-speed slinger belt that is capable of projecting the back-fill to a distance of 14 m and a height of 8 m.

However, for the North Block that is relatively shallow, instead of transporting the back-fill by truck to the filling orifice of the borehole at surface, if could be envisaged to truck the back-fill from the plant directly to the stope.

For the South Block, the borehole orifice will be directly besides the concrete plant.

### *3.10 - Truck requirements*

In view of the fact that back-filling requires moving 360 $m^3$/d and taking any irregularities into account, the trucking equipment should be able to handle 450 $m^3$/d.

With an average availability of trucks of 5.2 hr/shift and for the North Block, a truck cycle is evaluated at 37 minutes. With 9-$m^3$ trucks, 3 trucks are needed for 2 shifts, or 2 trucks for 3 shifts.

## 4 - COMMON SERVICES

### *4.1 - Fuel requirements*

Fuel requirements are evaluated at about 2,000 litres/d, of which more than half is for the LHDs and the trucks. All vehicles that should return to surface at the end of a shift, can be refuelled up top.

The other, less mobile, equipment such as jumbos, will require 300-400 litres/shift, which can be transported by a specialized tanker truck.

### *4.2 - Supply and storage of explosives*

An explosives warehouse, at surface or buried, can be sited close to the mine portal. This warehouse will be protected by closed-circuit TV, monitored from the mill control-room. Daily explosives consumption is estimated at between 500 and 1,000 kg.

*4.3 – Maintenance*

No underground maintenance work-shop is planned for the North Block. All vehicles, such as LHDs and trucks, return to surface at the end of each shift. The other engines, less mobile like the Jumbos, or less vulnerable such as the engines with elevating platform, will receive routine maintenance on site ans will come up to surface only for periodic overhaul. Maintenance could be largely subcontracted.

## 5 - CAPITAL INVESTMENT EVALUATION

INVCHY3

| CHESSY - CAPITAL INVESTMENT COST | CLASSES OF DEPRECIATION | | | | | TOTAL |
|---|---|---|---|---|---|---|
| Designation | Pre-Prod. Expenses | Construct Developm. | Mobile Equipment | Equipment Tools | Installat -ion | COSTS * 1000 F |
| MAIN DECLINE | 5229 | 16345 | 0 | 116 | 162 | 21852 |
| LANDING STATIONS | 3854 | | | | 164 | 4018 |
| BACK-FILL INSTALLATION | | 4727 | 3935 | 2701 | 722 | 12085 |
| UNDERGROUND PRIMARY CRUSHING | | 1073 | | 4349 | 1745 | 7167 |
| ORE HAULAGE | | 6176 | | 5944 | 981 | 13101 |
| MINE DEWATERING | | 1105 | | 801 | 600 | 2506 |
| MINE VENTILATION | | 5564 | | 550 | 620 | 6734 |
| ELECTRICITY SUPPLY | | | | 4145 | | 4145 |
| MOBILE EQUIPMENT | 526 | | 20963 | 105 | | 21594 |
| S/T MINE | 9609 | 34990 | 24898 | 18711 | 4994 | 93202 |

## 6 - EVALUATION OF OPERATING COST OF MINE

Basic date for a shift are:

- shift time: 8 hrs
- availability of manpower: 6.5 hrs
- availability of diesel equipment: 5.2 hrs

For a production of 330,000 tonnes ore/year, or 1,500 t/d over a 5-day week, expenses are distributed as follows:

| SUMMARY STATEMENT OF MINE OPERATING COST | Personnel | Sub-contractors | Electrical energy | Consu-mables | TOTAL FF/t |
|---|---|---|---|---|---|
| Staff | 5.67 | - | - | 6.27 | 11.94 |
| Mine development<br>. linking up chutes and stopes<br>. access for mining pillars | 6.42 | - | - | 4.03 | 10.45 |
| Ore production<br>. from drifts 600 t/d<br>. from stoping 900 t/d<br>. wall support | 11.63 | - | - | 18.59 | 30.22 |
| Back-filling<br>. preparation<br>. transportation and stowage | 5.04 | 5.68 | 0.48 | 16.45 | 27.65 |
| General services | | | | | |
| . chutes and ore passes | 3.10 | - | - | - | 3.10 |
| . primary crushing | 1.21 | - | 0.52 | 1.02 | 2.75 |
| . haulage | | | 0.81 | 1.02 | 1.83 |
| . dewatering | | | 0.37 | 0.02 | 0.39 |
| . ventilation | | | 1.45 | 0.40 | 1.85 |
| . road maintenance | | | - | 0.40 | 0.40 |
| . liaison vehicles | | | - | 0.37 | 0.37 |
| . explosives warehouse | - | - | - | - | - |
| . water and power distribution | | | 0.17 | - | 0.17 |
| . geotechnical studies | | 1.45 | - | - | 1.45 |
| . maintenace | 8.23 | | - | - | 8.23 |
| For North Block TOTAL FF/t | 41.30 | 7.13 | 3.80 | 48.57 | 100.80 |
| For South Block (additional haulage and dewatering costs) | | | | | |
| . if crushing at - 40 m : FF/t | 41.30 | 713 | 494 | 48.77 | 102.14 |
| . if crushing at - 140 m : FF/t | 41.30 | 7.13 | 6.08 | 48.97 | 103.48 |

## IV - THE MILL

The mill was was designed around a production rate of 1,080 t/d or 330,000 t/yr, based on a 7-day working week with 3 shifts per day.

### 1 - STORAGE OF ROM ORE AND WASTE

As storage of ore and waste on the ground takes up excessive room, it was envisaged to build four silos of 8-m diameter, or one 800 t silo for waste and three 1,250 t silos for ore, which corresponds to 3.5 days production.

### 2 - SECONDARY AND TERTIARY CRUSHING

This circuits will crush ore and waste (for back-fill) at the rate of 5 days/week and 2 shifts/day.

Designed for a through-put of 191 t/hr (average hourly through-put 153 tonnes), this is a standard-type circuit, with reduction to 50 mm and then to 16 mm carried out by two Symons cone crushers, the second (short head) being in a closed circuit on a screen.

### 3 - STORAGE OF CRUSHED ORE AND WASTE

The ore, crushed to about 16 mm, is stored in two silos with a total capacity of 2,500 tonnes, or two days mill through-put.

The waste from the mine, as well as that from the special quarry, does, in principle, not need any washing and will stored in heaps according to grainsize, close to the concrete plant.

### 4 - GRINDING AND CLASSIFICATION

Liberation of sulphides requires grinding to $d_{80}$ = 75 micrometers, all the while taking care that the quite fragile barite is not over-ground. For this reason, the classical two-stage grinding circuit was retained:

- grinding in a rod mill with rods of 2.30 m diameter and 5 m length;
- classification by cycloning rather than by a screen classifier;
- re-grinding of the oversize in a ball mill of 3 m diameter and 5 m length.

This circuit is designed for 67 t/hr, to handle an average through-put of 52 t/hr. The hardness of the ore being variable, depending partly on the pyrite content and partly on the silicates from dilution, the grinders are fitted with variable-speed motors that are commanded by the central mill computer.

The grinding building is designed to house a gravimetric separator, such as a jig, which, by using the density difference between ore and waste, could eliminate a major part of the diluting material. However, such gravimetric separation has not yet been tested and the conomic impact therefore is as yet unknown, but this optional installation could bring about a considerable improvement in run-of-mill ore specifications.

## 5 - FLOTATION

Flotation will be in the following sequence:

- chalcopyrite (with galena if present),
- sphalerite,
- pyrite,
- barite.

There will thus be four rows of flotation cells with for each, of course, roughers, scavengers and cleaners. The capacity of these cells and their number were calculated based on the parameters provided by pilot testing.

Separation of the pyrite is preceded by a desliming phase, in order to increase the flotation velocity and to avoid over-consumption of reagents, this for by-products on which the project economics do not depend.

The tailings pumped toward the ponds, consist therefore of two components:

- the overflow from desliming,
- the rejects from floating the barite.

Modern and continuous pulp-analysis techniques will permit automatic distribution of reagents and to follow the variations in grade of the various minerals.

The project takes into account:

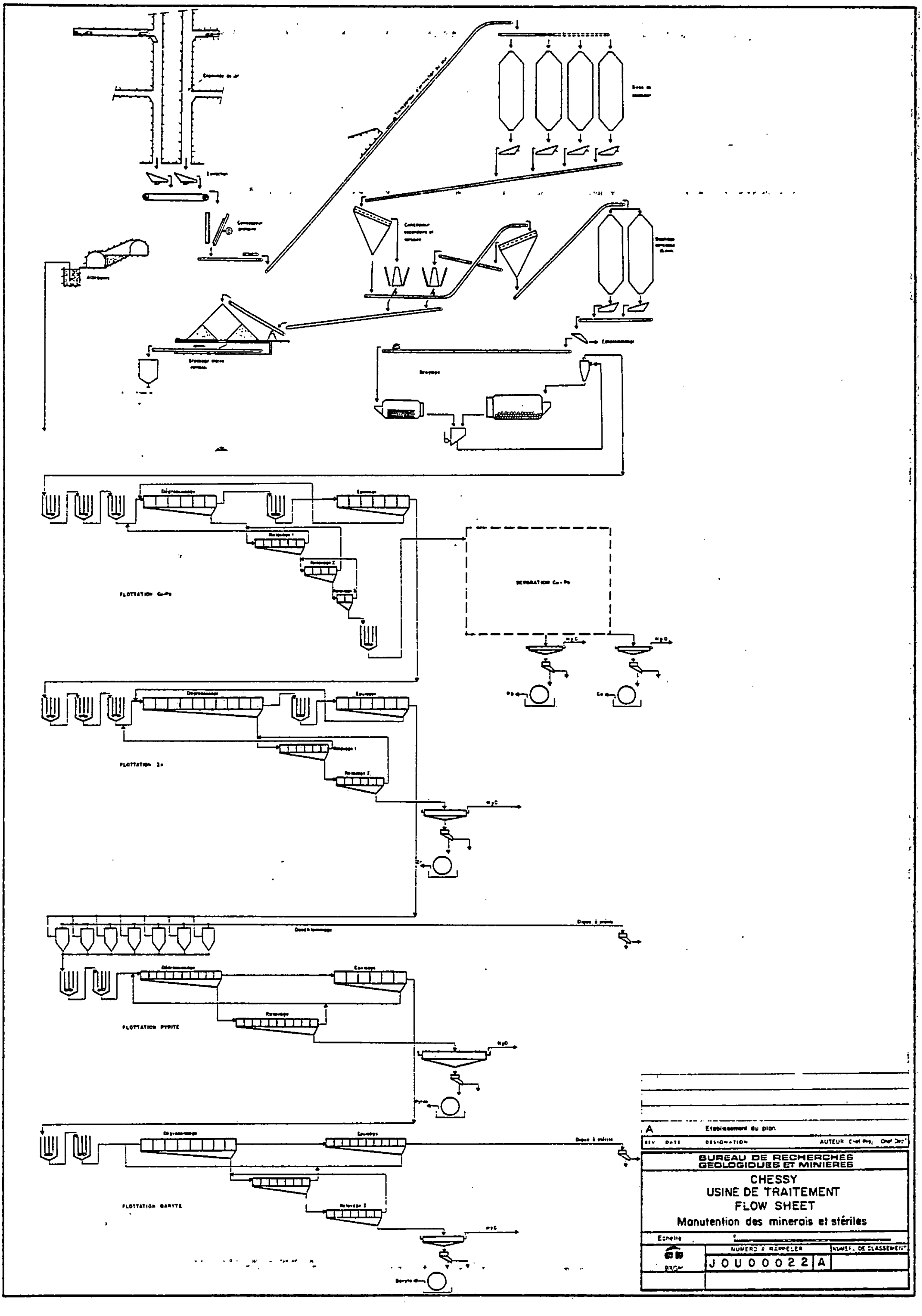
- continuous solids-analysers (XRF process sold by Outokumpu Oy) for Cu, Pb, Zn, Fe and Ba;

- four continuous liquids-analysers ("Pirana" process by BRGM) for organic reagents and minerals;

- a central steering unit (Proscon process automatics by Outokumpu Oy), which gathers all information, processes it and sends instructions to the reagent distributors.

## 6 - THICKENING AND FILTRATION

In view of the fact that 4 to 5 concentrates will be produced, it is logical that this circuit will be at the head of the concentrate-storage and -loading installations. In this study, the solution was adopted of a private railway spur for concentrate transport, for which reason the filtration unit is located at several hundreds of meters from the flotation circuit.




A Etablissement du plan

| REV | DATE | DESIGNATION | AUTEUR | Chef Proj. | Chef Dept |
|---|---|---|---|---|---|

**BUREAU DE RECHERCHES GÉOLOGIQUES ET MINIÈRES**

**CHESSY**
**USINE DE TRAITEMENT**
**FLOW SHEET**
**Manutention des minerais et stériles**

Echelle

BRGM | NUMERO A RAPPELER: J O U 0 0 0 2 2 A | NUMERO DE CLASSEMENT

The four thickeners and filters were designed based on the pilot-test data; the filters are standard drum filters. Complementary tests will have to determine the economic interest of adding press-rollers to decrease the moisture content of certain concentrates even further.

## 7 - SEPARATION OF COPPER AND LEAD IN THE BULK CONCENTRATE

The overall Pb grade of the orebody is low (about 0.30 %) and its presence in the copper concentrate has little impact on the market value of the latter. However, it seems that the position of the concentrate producer as against that of the smelter will be stronger if the Pb + Zn grade of the concentrate were to be limited to 5.5 %; this is behind the idea to extract from the bulk concentrate a purified copper concentrate as well as a galena concentrate.

By assuming for the copper concentrate a residual Zn grade of 4 % (which can be improved during industrial exploitation), the residual lead grade should be brought to about 1.5 %. On the other hand, care must be taken that a minimum amount of copper ends up in the galena concentrate.

A first series of laboratory tests has shown that several procedures can lead to the desired result. The definitive choice has not yet been made, as one hesitates between:

- gravimetric separation (shaking table), which should be a continuous operation that recovers about 60 % of the galena and 3 % of the copper in a concentrate grading 55 % Pb;
- high-intensity wet magnetic separation;
- separation by hot-pulp flotation with dextrine, a slightly-expensive process that, on the other hand, functions only intermittently, recovering 85 % of the galena and 5 % Cu in a concentrate grading 50 % Pb. Such an installation would only function above a cut-off grade of 0.40 % Pb, or about 32 % of the total tonnage processed;
- a simple grainsize classification towards 60 microns, which would allow to obtain, in the oversize fraction, a copper concentrate with a very markedly reduced lead content.

The present BRGM evaluations take the first (gravimetric) solution into account, which is easy to install, uses no additional reagents, and reduces Cu losses to a maximum. The tonnage to be processed is low, about 3 t/hr.

## 8 - PREPARING THE REAGENTS

Eleven different reagents will have to be used, to a total of 5.5 t/d.

| REAGENTS IN KG/TONNE ORE | Grinding | Flotation | | | | |
|---|---|---|---|---|---|---|
| | | Cu | Zn | py | BaSO4 | TOTAL |
| Lime Ca(OH)2 | 540 | 170 | 1 780 | - | - | 2 490 |
| Sulphuric acid | - | - | - | 415 | - | 415 |
| Aerofloat 238 | - | 30 | 140 | - | - | 170 |
| Amylxanthate K (Aero 350) | - | - | - | 100 | - | 100 |
| Methylisobutyl Carbinol (MIBC) | - | 15 | 5 | 20 | - | 40 |
| Melioran B109 | - | - | - | - | 415 | 415 |
| Zinc sulphate | - | 250 | - | - | - | 250 |
| Copper sulphate | - | - | 330 | - | - | 330 |
| Sulphidic anhydrite gas ($SO_2$) | - | 415 | - | - | - | 415 |
| Hexametaphosphate | - | 30 | - | - | - | 30 |
| Sodium silicate - | - | - | | - | 415 | 415 |

## 9 - CAPITAL INVESTMENT IN THE ORE-PROCESSING PLANT

INVCHY3

| CHESSY - CAPITAL INVESTMENT COST | CLASSES OF DEPRECIATION | | | | | TOTAL |
|---|---|---|---|---|---|---|
| Designation | Pre-Prod. Expenses | Construct Developm. | Mobile Equipment | Equipment Tools | Installat -ion | COSTS * 1000 F |
| ORE & WASTE HANDLING | | 1253 | | 836 | 3223 | 5312 |
| SECONDARY & TERTIARY CRUSHING | | 794 | | 3088 | 1553 | 5435 |
| CRUSHED ORE STORAGE | | 2031 | | 873 | 248 | 3152 |
| ORE GRINDING | | 5335 | | 7003 | 686 | 13024 |
| FLOTATION | 231 | 5907 | 208 | 45 | 10862 | 17253 |
| FILTRATION | | 1922 | 147 | 4830 | 1532 | 8431 |
| REAGENT PREPARATION | | 1901 | | 1787 | 531 | 4219 |
| Pb-Cu SEPARATION | | 446 | | 2159 | 840 | 3445 |
| S/T MILL | 231 | 19589<br>0 | 355 | 20621 | 19475 | 60271 |

Capital expenditures for equipment replacement are evaluated at 5 %/year the total equipment value, over the entire mine life exept the last two years.

## 10 - OPERATING COST OF MILL

| MILL<br>330,000 t/yr or 1,080 t/d | Personnel | Sub-contracting | Electric energy | Consu-mables | TOTAL FF/t |
|---|---|---|---|---|---|
| Secondary and tertiary crushing | 3.51 | - | 0.73 | 2.30 | 6.54 |
| Grinding | | | | | |
| . balls and rods | - | - | - | 4.40 | 4.40 |
| . liner and others | - | - | 6.75 | 3.11 | 9.86 |
| Flotation | | | | | |
| . reagents | - | - | - | 12.95 | 12.95 |
| . other | 6.43 | - | 6.98 | 4.20 | 17.61 |
| Filtration | - | - | 0.92 | 0.60 | 1.52 |
| Pb/Cu Separation | - | - | 0.15 | 0.10 | 0.25 |
| Reagent preparation | 2.37 | - | 0.04 | 0.56 | 2.97 |
| Maintenance | 8.14 | 5.00 | - | - | 13.14 |
| Staff | 5.85 | - | - | - | 5.85 |
| Various | 1.19 | - | - | - | 1.19 |
| TOTAL FF/t | 27.49 | 5.00 | 15.57 | 28.22 | 76.28 |

## V - TAILINGS DISPOSAL

The maximum of possibilities, covering about ten sites, was studied and compared, of which four (sites 5, 6, 7 and 8) were shortlisted, and of which site n° 6 was retained for cost evaluation as it was the most throroughly studied (trenches, geotechnical sampling and testing, calculating the stability conditions of the dam).

### 1 - PRINCIPLES

- A tailings disposal area must be selected, whose capacity is sufficient for the planned mine life, if not more; if possible, two areas should be envisaged, which gives more flexibility to the exploitation.
- The long-term stability of the tailings dam should be ensured, as well as its protection against erosion by run-off or by floods, which invalidates the classical solution of constructing the dam with the largest sand-size fraction of the flotation waste. It is proposed here, to construct a dam of rip-rap.
- The cheapest rip-rap is the waste from the infrastructure-development work of the mine, extracted before start-up of exploitation. It should thus be verified that such material will be available in sufficient quantities to construct the dam.
- It should be avoided to encroach upon the vineyards and, in general, those solutions must be selected that have the least influence upon the environment.
- The technical agreement of the Permanent Technical Commithee of Dams (CPTB) should be obtained; this committee can ask to be informed and consulted as soon as the height of a dam exceeds 10 m, and requires this as soon as the height rises above 20 m.

### 2 - SPECIFICATIONS OF THE TAILINGS

- The tailings will consist of a sand composed of:
  - 50 % desliming product with a grainsize less than 40 microns,
  - 50 % with a grainsize between 40 and 150 microns.
- The dry density of the tailings is 1.75 t/m$^3$.
- The humidity of the drained tailings should be about 20 % (of the wet weight).
- The wet density of the drained tailings varies between 2.05 and 2.20; the value of 2.10 t/m$^3$ was retained for calculations.
- The tailings are pumped and dumped in a basic environment (pH = 9 to 12).

## 3 - ANNUAL QUANTITY OF TAILINGS

In view of the silicate gangue in the massive ore, of the dilution during mining, and of the imperfect recovery during flotation, the tailings will present 15 to 25 % (20 % was used for calculations), of the run-of-mill ore. One should thus dispose of:

| Mineables Reserves | Dry Tailings | Wet tailings ($H_2O$ = 25 % net weight or 33 % of dry weight) | Volume to be disposed (d = 2.10) |
|---|---|---|---|
| N + S blocks = 3.645.000 t | 729.000 t | 972.000 t | 463.000 m3 |
| Unités N + S + potentiel = 5 645 000 t | 1.129.000 t | 1.505.000 t | 717.000 m3 |

### *3.1 - Solution 6: De la Ronze valley: dam below the "Etang du Maire"*

* Damsite between the "Etang du Maire" (the Mayor's Pond) and the axis of the exploration decline.

* Dam:

  - generally straight, but with wings oriented slightly stream-upwards;
  - height 20 m (274 to 294 a.s.l.) above the present ground surface;
  - length of crest 320 m;
  - width of crest 6 m;
  - upstream and downstream slopes 26.5°;
  - required volume of rip-rap 95,000 $m^3$;
  - foundation: granite sand on left bank and in valley axis, weathered volcanic rock on right bank. The bottom of the valley shows over 40 m width a silty, water-logged fill that is 4 m thick and has mediocre mechanical characteristics.

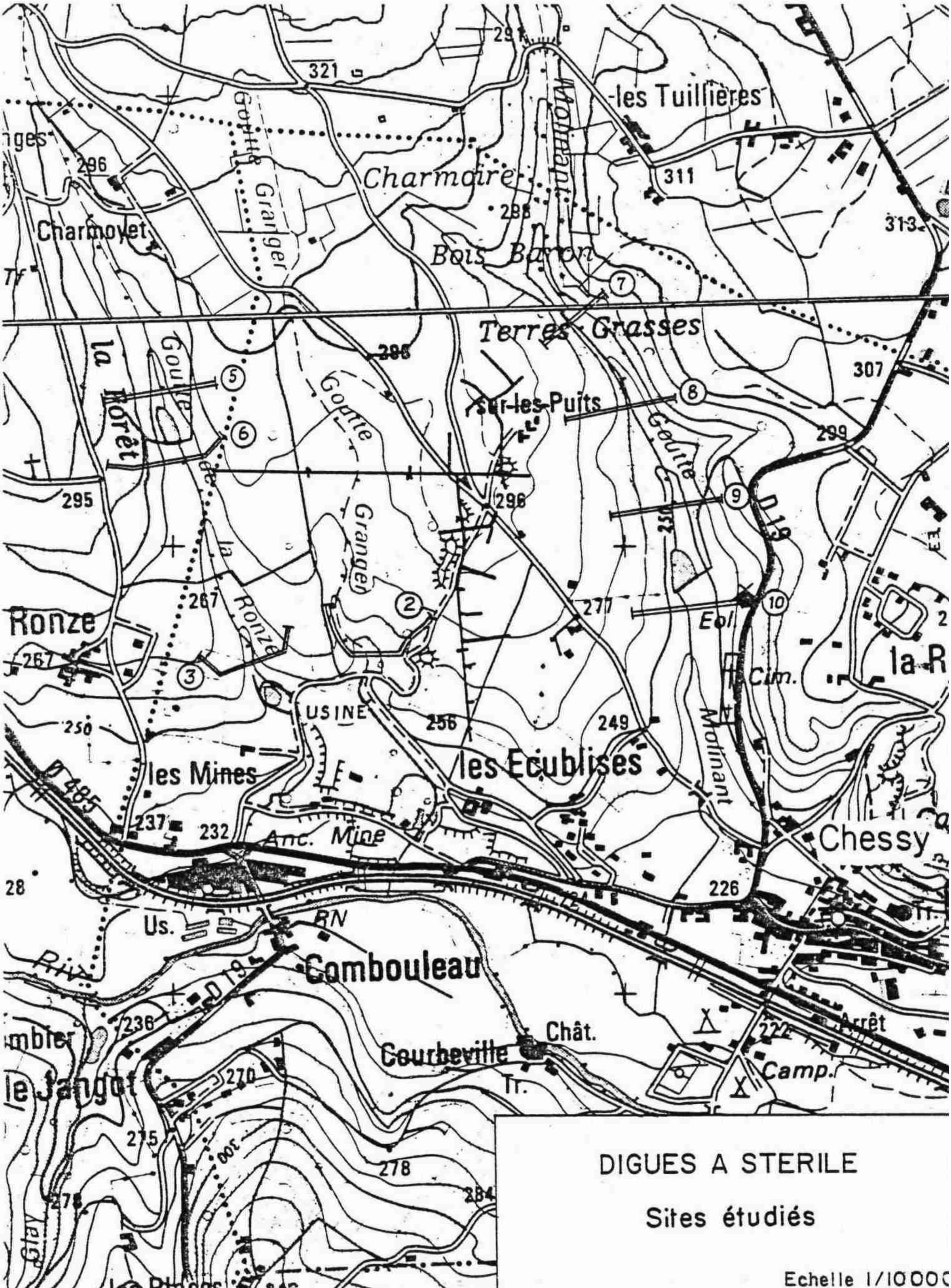
* Dam capacity is 560,000 $m^3$ from the 274 to the 293 m levels, which is more than the planned exploitation requirements that are 463,000 $m^3$ (293 is the level of the spill-way).

* Environmental impact of dam and tailings pond:

  - occupied ground is mainly grass-land and forest, and a few hundred $m^2$ of vineyard below the spill-way;
  - the community of Breuil is not subject to the national zoning regulations (Plan d'Occupation des Sols or Soil-Use Planning).

* Connection between mill and dam:

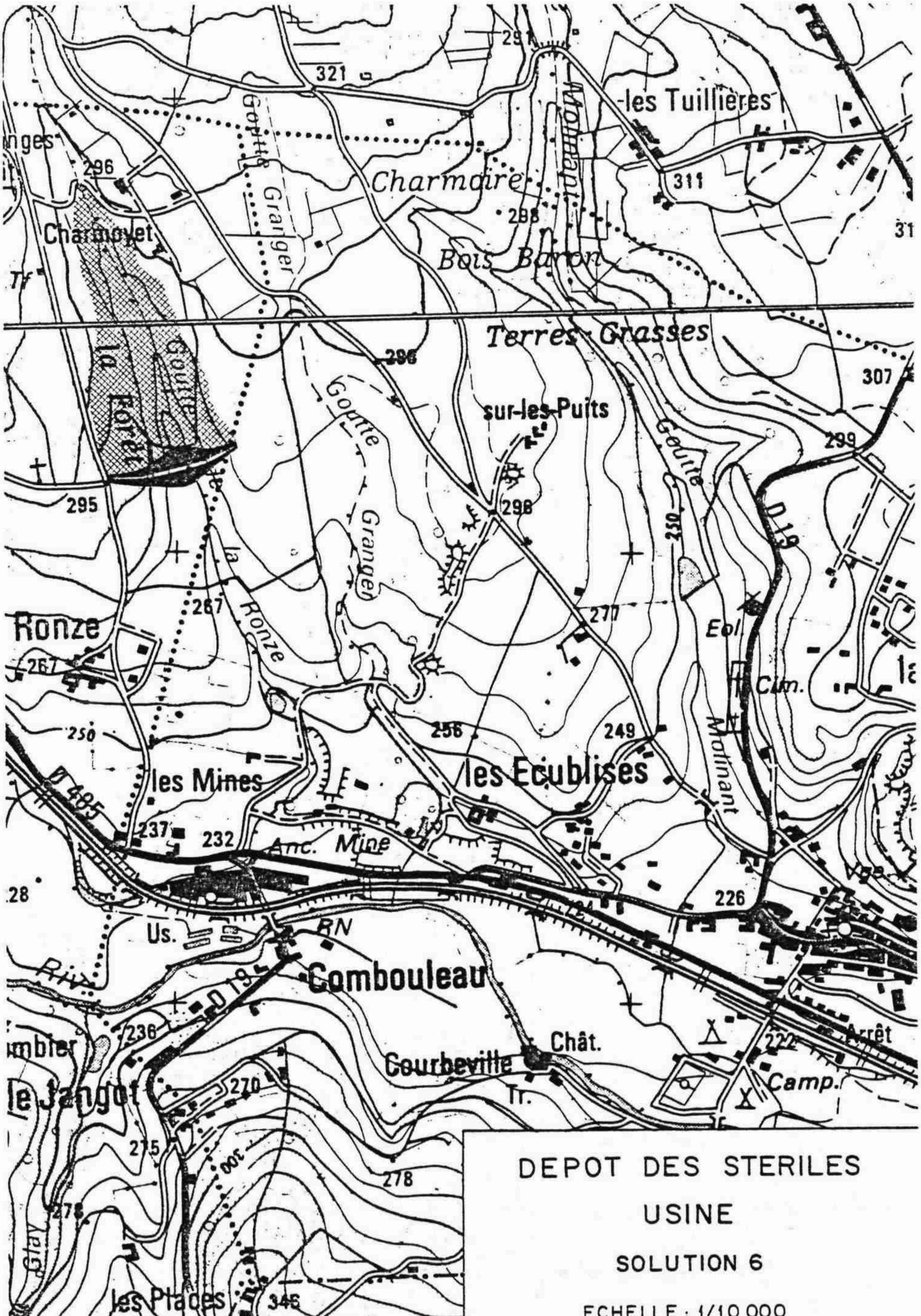
  - distance is 750 m,
  - pumping height is 50 m,




DIGUES A STERILE

Sites étudiés

Echelle 1/10000




DEPOT DES STERILES

USINE

SOLUTION 6

ECHELLE : 1/10 000

- a connecting track will have to be built along the Ronze valley.

* Studies and works carried out on this solution:

  - plans and sections at scale 1:1,000, which allowed to calculate the retaining capacity of the pond for various heights, as well as the dam volume itself;

  - reconnaissance work by means of 11 trenches to 5-m depth, along the E-W and N-S axes of the damsite;

  - sampling of undisturbed soils for shear testing in the geotechnical laboratory of BRGM Orleans.

* Advantages of solution 6:

  - capacity is 20 % more than the planned production,
  - the ratio rip-rap/capacity volume is the best for the Ronze valley.

* Disadvantages of solution 6:

The crown of the dam comes up to road level; further incrase of the retaining capacity remains possible but limited.

* Conclusion: solution 6 is the most attractive one and has served as calculating basis for the Chessy mining project.

## 4 - OPERATING COST OF TAILINGS DISPOSAL

Not counting manpower costs, the operating costs are as follows:

| | FF/t |
|---|---|
| - Electrical energy 0.81 kWh/t x 0.42 FF | 0.34 |
| - Consumables, such as pump fittings and parts | 1.15 |
| | 1.49 FF/t |

## VI - SUPPLY OF INDUSTRIAL WATER

### 1 - RECAPITULATION OF REQUIREMENTS

For a production capacity of 1,080 t/d, the water requirements are 80 $m^3$/hr, which are to be obtained in the following manner:

- 20 $m^3$/hr from dewatering of the mine (which should have a basic pH because of the use of censented back-fill;
- 60 $m^3$/hr from the Triassic-sandstone aquifer.

### 2 - TAPPING THE AQUIFER

The static water level is at 234 m a.s.l. within borehole S178 and at level 220 m in the lowest point of the Azergues valley.

It is proposed to sink a vertical well by percussion drilling to 100 to 160 m depth, rather than drilling a cored and horizontal borehole from the mine workings.

### 3 - WELL-SITE SELECTION

As a first approach, the local geology imposes a site in a strip that is parallel and 200-400 m east of the N20 fault. It is important that the borehole intersects the Triassic sandstone and reaches the paleogeographical contact between sandstone and siltstone, which is a discontinuity that is favourable to water circulation.

The proposed well site lies in the valley, at the angle between the old departmental highway D485 and the communal road n° 106, called locally "of Saint Gobain".

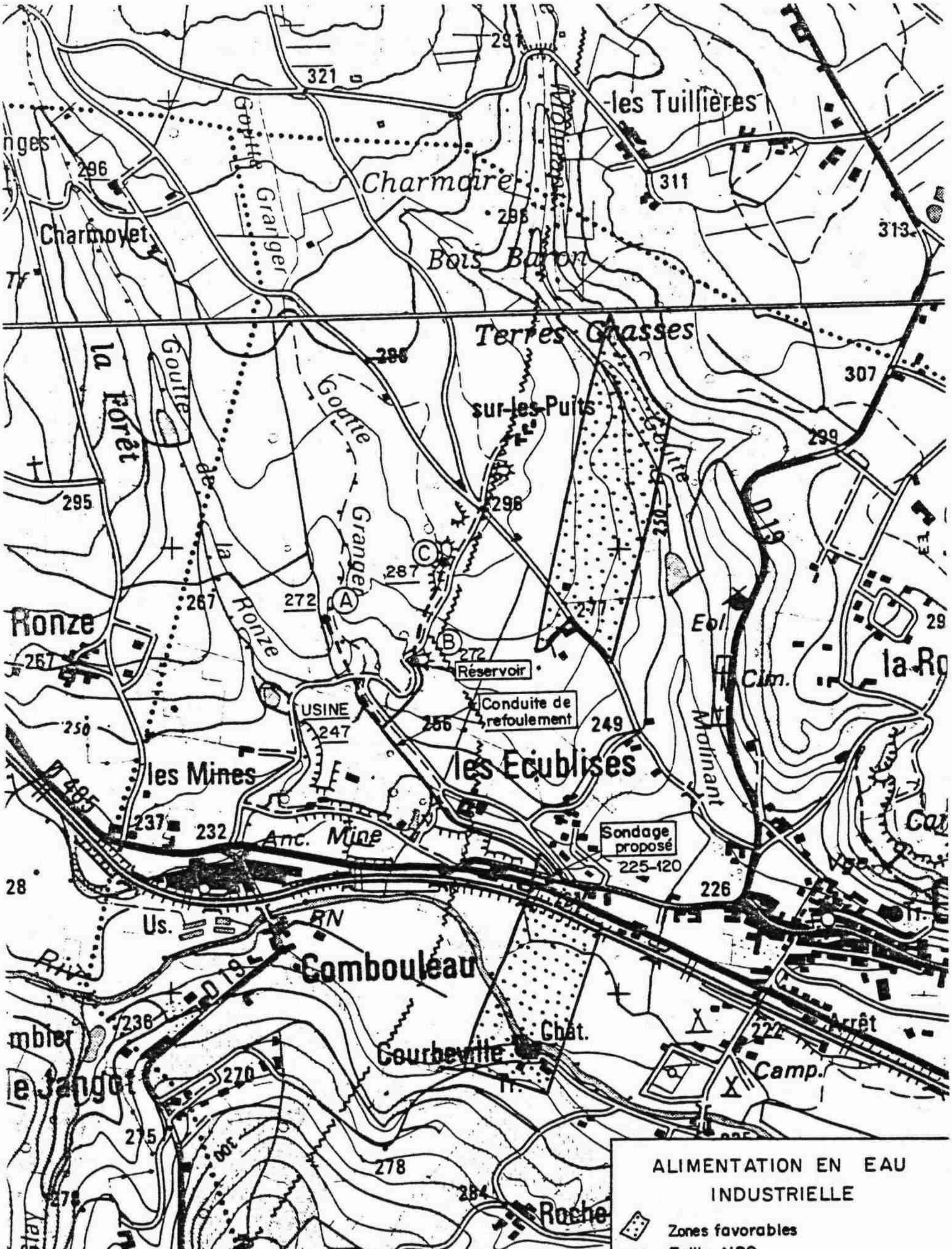
### 4 - WATER RESERVOIR

As the well collar will be at 225 m a.s.l., the water must be pumped to a reservoir where recycled water, mine water, and well water will be stored. This reservoir should provide water under pressure to the mill, whose ground level will be at 247 m. Three favourable sites are available, which are shown on the 1:10,000-scale plan hereafter. Of these, site B was arbitrarily selected.

To provide water autonomy of about 10 hours, the reservoir capacity should be about 800 $m^3$.

## 5 - OPERATING COST OF WATER SUPPLY

| | | |
|---|---|---|
| - Electrical energy 1.15 kWh/t x 0.42 FF | | 0.48 FF/t |
| - Spare parts | | 0.16 FF/t |
| | Total | 0.64 FF/t |



ALIMENTATION EN EAU INDUSTRIELLE
Zones favorables
Faille N20
287 Altitude
Echelle 1/10 000
les Tuillières
Charmaire
Charmoyet
Bois Baron
Terres Grasses
sur-les-Puits
Goutte Granger
Goutte de la Ronze
la Forêt
Ronze
Réservoir
Conduite de refoulement
USINE
les Ecublises
les Mines
Anc. Mine
Sondage proposé
225-120
Molinant
Combouleau
Courbeville
Chât.
Camp.
Arrêt
le Jangot
les Places
Roche
Cim.

## VII - DISCHARGE OF EFFLUENTS

### 1 - VOLUME OF EFFLUENTS

Discharge into the surface drainage system will be equivalent to the dewatering volume of the mine (20 $m^3$/hr), plus the 60 $m^3$/hr taken from the aquifer, or a total of 80 $m^3$/hr.

### 2 - DISCHARGE POINT

If the tailings dam is built as planned in the Ronze valley, discharge will take place in the stream of the same name or, if necessary, in the nearby Azergues river.

### 3 - DISCUSSION OF DISCHARGE STANDARDS TO ADOPT

In order to avoid any criticism, a self-imposed standard was adopted in agreement with the Departmental Directorate of Agriculture, that corresponds to standard 1B for the Azergues.

The reference minimum flow is 170 l/s or 610 $m^3$/h and thus the standard applies to 80 $m^3$/h of effluent diluted by 610 $m^3$/h of supposedly pure water.

* As for the mineral substances, it can be noted that only the zinc content exceeds the standards, and than only in the flotation effluent of pyrite. This can be corrected in the following manner:

  - by reducing the total zinc-sulphate consumption from 1,600 g/t to 250 g/t;

  - by floating pyrite with a pH of 7.5 instead of 5.8, which considerably reduces the solubility of zinc sulphate;

  - by automatic regulating of the reagents as a function of requirements, which only uses the amount of reagent that is useful for flotation and remains trapped on the grain surface of the concentrates.

* As for the organic substances, one disposes unfortunately not of analyses, but the experience gathered from other mining ventures that process similar ores, confirms that such reagents are broken down when filtering through the tailings dam (example of the Zinkgruvan mine, DB07 content in filtration water is 11 to 67 mg/l, the overflow from the tailings dam has a content of 2-7 mg/l, and the recycled water contains 0.4-6 mg/l. Once diluted, this flux will answer the standard requirement of 3-5 mg/l.

At the exit of the collector that recovers decantation fluids, the water composition will be constantly monitored, especially for those elements that have to be watched, but in principle it seems that no further processing of the water will be necessary. The water can be decanted once more in the existing basins below the exploration decline, before being discharged into the Ronze stream.

## 4 - OPERATING COST OF EFFLUENT DISCHARGE

| | |
|---|---|
| - Analyses for water quality | 0.07 FF/t |
| - Electrical energy 0.11 kWh/t | 0.05 FF/t |
| - Chemical products (provision) | 0.10 FF/t |
| - Tools and spare parts | 0.02 FF/t |
| Total | 0.24 FF/t |

# VIII - ELECTRICAL ENERGY SUPPLY

## 1 - ELECTRICAL POWER REQUIREMENTS

For a production rate of 330,000 t/yr, the electricity use is estimated at 4-4.5 MW. The Vienne office of Electricité de France (EDF - French Electricity Board) that is responsible for the area of the mine site, has been informed of the project in October 1985, so as to be able to integrate these data in their plans to extend the 60 kV network; EDF is periodically informed of progress in the project.

EDF has promised that the required power could be available by 1988.

## 2 - DESIGNING POWER TRANSMISSION

The present power supply consists of a 20 kV line of 9 km that links the sub-station at Civrieux-sur-Azergues with the medium/low voltage transformer close to the Mathelin factory. This line passes directly over the future mine site; one could not be better located!

## 3 - OPERATING COST

The recommended billing is the "Green A Tariff" with five seasonally-adjusted rates. When the development decision has been taken the operator should choose the best tariff: "general" or "long-duration".

Awaiting this, the amount paid by other mines in France has been used as a yardstick, which is on average 0.42 FF/kWh.

# IX - CONCENTRATE TRANSPORT

## 1 - GENERAL POINTS

### *1.1 - Tonnages and specifications of the concentrate to be transported*

For an annual run-of-mill production of 330,000 tonnes ore:

| Mined ore diluted grade % | Recovery at processing % | Concentrate grade % | Dry conc. tonnage | $H_2O$ % | Wet conc. tonnage # |
|---|---|---|---|---|---|
| 2.23 Cu | 92 | 28.3 | 23,923 | 9.5 | 26,000 |
| 9.60 Zn | 94 | 56.0 | 53,177 | 11 | 59,000 |
| 35.22 Py | 83 | 98.8 Py | 97,639 | 8 | 105,000 |
| 28.3 $BaSO_4$ | 77 | 98 $BaSO_4$ | 77,377 | 6 | 78,000 |
| | | | 248,116 | | 268,000 |

The tonnage to be transported is thus close to 270,000 t/yr, or 1,220 t/d on a basis of 220 d/yr, which corresponds to a capacity of:

- 1 full train per day,
- 49 articulated trailer trucks of 25 t per day,
- 5 barges of 250 t (Freycinet type) per day,
- 0.9 barge load of a Rhine-type barge per day.

### *1.2 - Destination of concentrates*

For this report, it was deemed unnecessary to indicate to the future investor and operator which potential client would be best; a large number of possible solutions was regarded that will not be gone into detail here.

## 2 - TRUCK TRANSPORT

Transport of concentrate by trucks was studied and, at an early stage, contacts were established with a transport firm.

The average distance mine-to-consumer is, regardless of concentrate type, about 700 km; supposing a cycle-frequencyof three days, the fleet would require 100 to 150 trucks, which might be a very attractive proposition to many transport firms.

## 3 - RAIL TRANSPORT

Creating a terminal-spur line, allows the application of rates that cover expediting complete trains of 18 to 20 cars with a useful load of 1,200 to maximum 1,340 tonnes. The basic freight rate depends on the type of goods, is lowest for bulk-freight and in fact varies very little between copper, zinc, iron, or barite concentrates.

Various solutions were studied:

### *3.1 - Solution 1: Legny-le-Bois d 'Oingt railway station*

This station, located 4 km from the mine at "Ponts Tarets" along the CD 485 highway on the left bank of the Azergues, presents a railway infrastructure that would only need little modification.

### *3.2 - Solution 2: private spur line to Chessy*

Three variants that are technically possible have been studied in detail, two of which would require a level crossing with the CD 485 highway.

### *3.3 - Solution 3: rail loading at the river port of Villefranche*

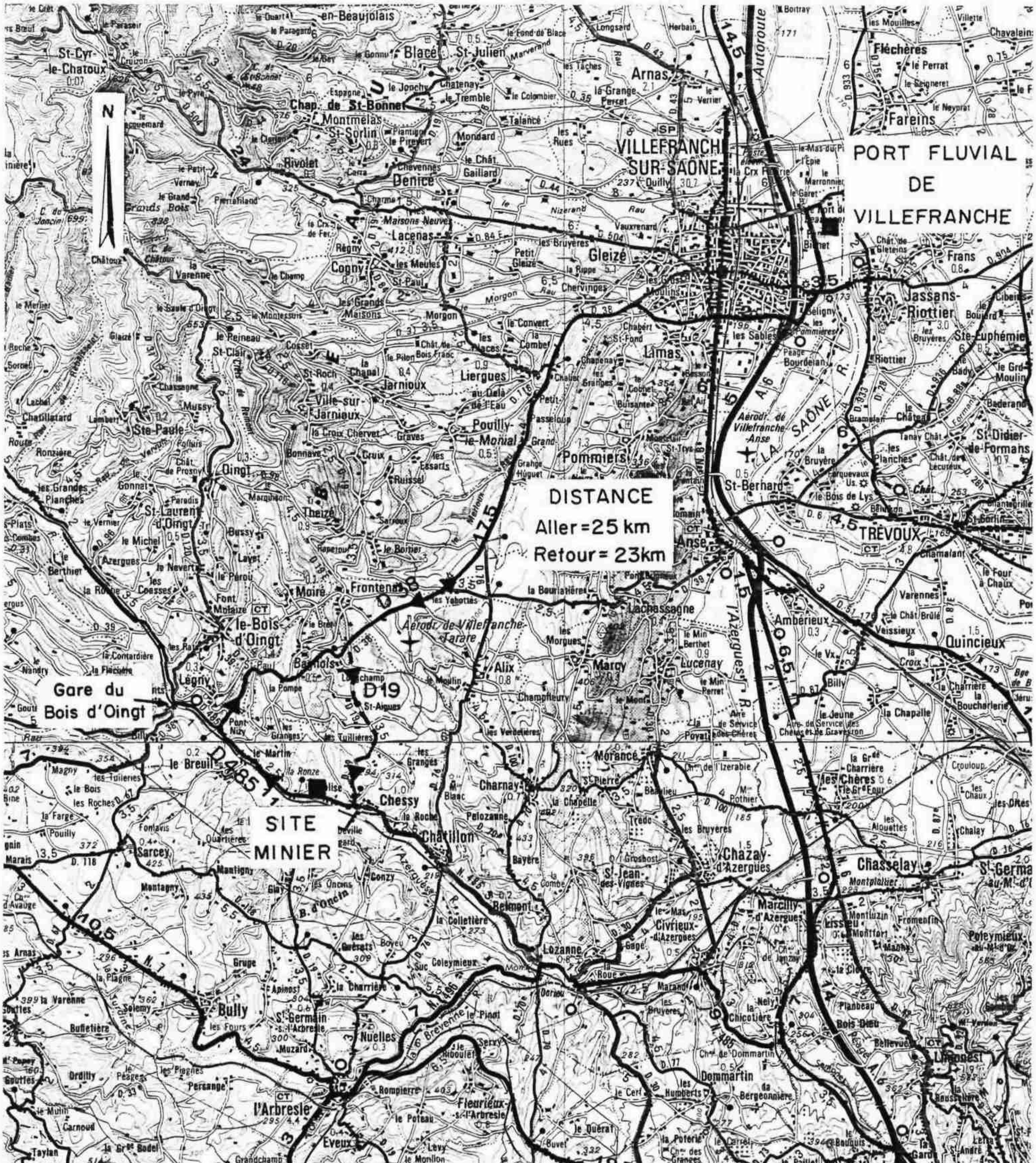
The port of Villefranche on the Saone river, 23 km from the mine site, contains a private spur line that could be extended to the possible concentrate storage installations. The objective would be to saturate as a priority the barge traffic, the complement being assured by complete train loads. In that case, the main storage installations would be in the port.

## 4 - RIVER TRANSPORT

### *4.1 - Dimensions of water ways*

The Saone, like the Rhone, is accessible to large-size barges. On the axis Marseilles-Macon (4,500 t gauge) move 2,000-t barges, pushed barges of 4,000 t, and coastal shipping.

However, for any destinations to the north of Chalons-sur-Saone and in the absence of a large-gauge link between Saone and Rhine, one has to use 250-t barges (Freycinet size) of 38.50 m length.




TRANSPORT DES CONCENTRES

Itinéraire routier de la Mine au Port de Villefranche

ECHELLE : 1/100 000

### *4.2 – The public port at Villefranche-sur-Saone*

The new port presents a dock length of 2 km and an "edge" that can be developed of several hundreds of meters, which has in particular attracted our attention. Present port traffic is 140,000 t/d or t/yr ?

The strong point of the new port is that it is tri-modal, which is to say that it was conceived to permit cargo exchanges between rail, road, and water-way.

One of the advantages of water-way transport is the large tonnage that can be shipped at the same time (up to 4,000 t), in particuliar to the south. For this reason it is to be preferred to have the main storage site in the dock yard, with a minimum storage at the mine site, the two areas being linked by means of road transport over 23 km.

Two possibilities were studied in more detail:

- storage in stalls, concentrate handling by front-end loaders, and barge loading by conveyor belt;

- storage in longitudinal stalls, concentrate handling and barge loading by loading-bridge crane.

Another possibility considered, is storage in a special pit along the quay side and barge loading by travelling crane with a grab.

## 5 - MIXED RAILWAY WATER-WAY TRANSPORT

This solution, applicable to concentrate transport to countries to the north and northeast of France, is a compromise that allows to profit as much as possible of the French rail-transport rates as well as of the best river-transport conditions.

It consists of transporting the concentrate by rail to the river port of Fronard (close to Nancy in NE France) and to transfer the entire train load into a Rhine barge. This solution has been in use successfully since several years, to transport barite concentrate from Chaillac to Cologne.

## 6 - OPERATING COST OF CONCENTRATE TRANSPORT

Operating costs include loading of rail cars as well as rail transport by SNCF:

- Loading cost: loading and weighing are done by one man (one shift), equipped with front-end loader of the Cat 980 type, a radio-controlled track motor car, and an automatic weighbridge. The cost of the consumables is:

| | | |
|---|---|---|
| . Front-end loader | 6 hrs x 262.38 FF/hr = | 1,574 FF |
| . Track engine | 6 hrs x 50.00 FF/hr = | 300 FF |
| | Total/shift | 1,874 FF |

for 220 shifts/yr, or 1.25 FF/t run-of-mill ore.

- Transport costs are accounted for in the pit-head revenues. As opposed to loading costs, transport costs do not form a direct item of operating costs.

The costs of the various transportation means were not negotiated and can be considered as a starting price. The rail-transport costs come from a written proposal, which explains why they were retained for the economic evaluation, but which in no way should be seen as a final choice; this choice should be made by the mine operator.

**COMPARISON OF CONCENTRATE-TRANSPORT COSTS IN 1986 FF/t**
**(not counting capital investment costs for loading installations)**

| Destination / Solution | Hoboken (Belgium) | Noyelles G./Auby (France) | Venice (Italy) | Bad Honningen (F.R.G.) | Homberg (F.R.G.) | Dieuze (Moselle-France) |
|---|---|---|---|---|---|---|
| Annual wet reference tonnage | 26,000 t Cu - Pb conc. | 59,000 t Zn conc. | 105,000 t Pyrite conc. | 78,000 t Barite conc. | Pyrite conc. | Barite conc. |
| **Direct train** | | | | | | |
| - distance in km | (628 + 121 = 749) | (629) | (329 + 504 = 833) | (459 + 191 = 650) | | |
| - train cost | 195.75 | 124.37 | 78.69 + 87.89 | 161.86 | | |
| - empty return | 5.00 | 5.00 | 5.00 | 5.00 | | |
| - premium for running on time | 0 | 0 | 0 | 0 | | |
| - premium for programming | 0 | 0 | 0 | 0 | | |
| Total | 200.75 | 12.37 | 171.58 | 166.86 | | |
| **Train + barge** | Via Nancy (Frouard) | Via Nancy (Frouard) | Via Nancy (Frouard) | Via Nancy (Frouard) | Via Nancy (Frouard) | |
| - train distance in km | (316) | (316) | (308) | (316) | (316) | |
| - train cost | 101.08 | 101.08 | 90.12 | 101.47 | 101.47 | |
| - empty return | 5.00 | 5.00 | 5.00 | 5.00 | 5.00 | |
| - on-time premium | -6.00 | -6.00 | -600 | -6.00 | -6.00 | |
| - programming premium | -4.00 | -4.00 | -4.00 | -4.00 | -4.00 | |
| - port transport | 9.39 | 9.39 | 70 (?) | 9.39 | 9.39 | |
| - barge-transport distance | (?) | (?) | (?) | (?) | (?) | |
| - barge cost | 43.47 | 55.50 (estim.) | 77 (2) | 26.01 | 34.98 | |
| - fixed costs | | | 2 | 5.16 | | |
| Total | 149.04 | 160.97 | 234.12 | 137.03 | 140.84 | |
| Train, when departing from Bois d'Oingt-add. cost | + 6 | + 6 | + 6 | + 6 | + 6 | |
| **Truck + barge** (via Villefranche) | | | | | | |
| - truck transport 23 km | 15 | 15 | 15 | 15 | | |
| - port transit Villefranche | 10 | 10 | 10 | 10 | | |
| - barge transport | 135.75 | 118.20 | 40 | 126.23 | | |
| - insurance | 3.90 | 3.30 | 0.40 | 0.60 | | |
| - maintenance | - | - | 30 (1) | - | | |
| - sea transport | - | - | 77 (2) | - | | |
| - customs duties | 2.00 | 2.00 | 2.00 | 2.00 | | |
| - charter costs | 0.60 | 0.60 | - | 0.60 | | |
| Total | 167.20 | 147.10 | 174.40 | 154.40 | | |
| **Truck alone** | | | | | | |
| - distance to be invoiced | (750) | (616) | (734) | | | (446) |
| - truck transport | 192 | 179.50 (1) | 188 ? | | | 142.20 (3) |
| - customs duties | 28 | - | 28 | | | - |
| Total | 220 | 179.50 | 216 | | | 142.20 |
| **Truck + train** (via Villefranche) | | | | | | |
| - distance to be invoiced | | (594) | | | | |
| - truck transport 23 km | | 15 | | | | |
| - port transit Villefranche | | 10 | | | | |
| - train transport | | 191 | | | | |
| - bonus | | -42 | | | | |
| - connection tax | | 0.35 | | | | |
| Total | | 174.35 | | | | |
| **Truck + water transport** (via Villefranche) | | | | | | |
| - distance to be invoiced | | | 15 | | | |
| - truck transport 23 km | | | 10 | | | |
| - port transit Villefranche | | | 275 | | | |
| - river-sea transport | | | 0.4 | | | |
| - insurance | | | 2 | | | |
| - customs duties | | | | | | |
| Total | | | 302.4 | | | |

(1) Transfer from hold to hold. If to be stored under cover add 40 FF
(2) 5 000-t vessel
(3) Official 1985 cost, after application of maximum reduction of 23.1 %

# X - OTHER TECHNICAL SERVICES

Under Technical Services were grouped:

- storage of mill tailings and waste (described earlier in detail),
- industrial water supply (described earlier in detail),
- effluent discharge (described earlier in detail),
- electrical supply and distribution (described earlier in detail),
- stockage and loading of concentrates (described earlier in detail).

However, this section also covers several other activities:

- maintenance workshops: for ease of calculation, the workforce was distributed on paper between mine and mill, except for the common staff, consisting of a workshop superintendant and a foreman;
- the laboratory: ways and means were defined by the MGA laboratory of BRGM at Orléans;
- research and development of the mill;
- exploration/surveying/drafting section.

Operating costs of all these services combined are as follows:

| Summary of all technical services in FF/t | Personnel | Sub-contracting | Electrical energy | Consum-mables | TOTAL FF/t |
|---|---|---|---|---|---|
| Tailings and waste disposal of mill | - | - | 0.34 | 1.15 | 1.49 |
| Industrial-water supply | - | - | 0.48 | 0.16 | 0.64 |
| Effluents discharge | - | 0.07 | 0.05 | 0.12 | 0.24 |
| Electricity supply | - | | | | |
| Concentrate loading | 0.48 | - | - | 1.25 | 1.73 |
| Concentrate transport (see revenue calc.) | p.m. | p.m. | p.m. | p.m. | p.m. |
| Workshop (part non accounted for) | 2.10 | - | 0.24 | 0.48 | 2.82 |
| Laboratory | 2.96 | - | 0.11 | 1.20 | 4.27 |
| Mill R & D (1) | 1.14 | - | - | 0.67 | 1.81 |
| Road and service network | - | 0.27 | 0.17 | 0.05 | 0.49 |
| Mineral exploration to extend reserves (1) | 1.31 | 6.06 | 0.05 | 1.73 | 9.15 |
| Exploration geology/drafting/surveying | 3.52 | p.m. | p.m. | p.m. | 3.52 |
| TOTAL in FF/t | 11.51 | 6.40 | 1.44 | 6.81 | 26.16 |

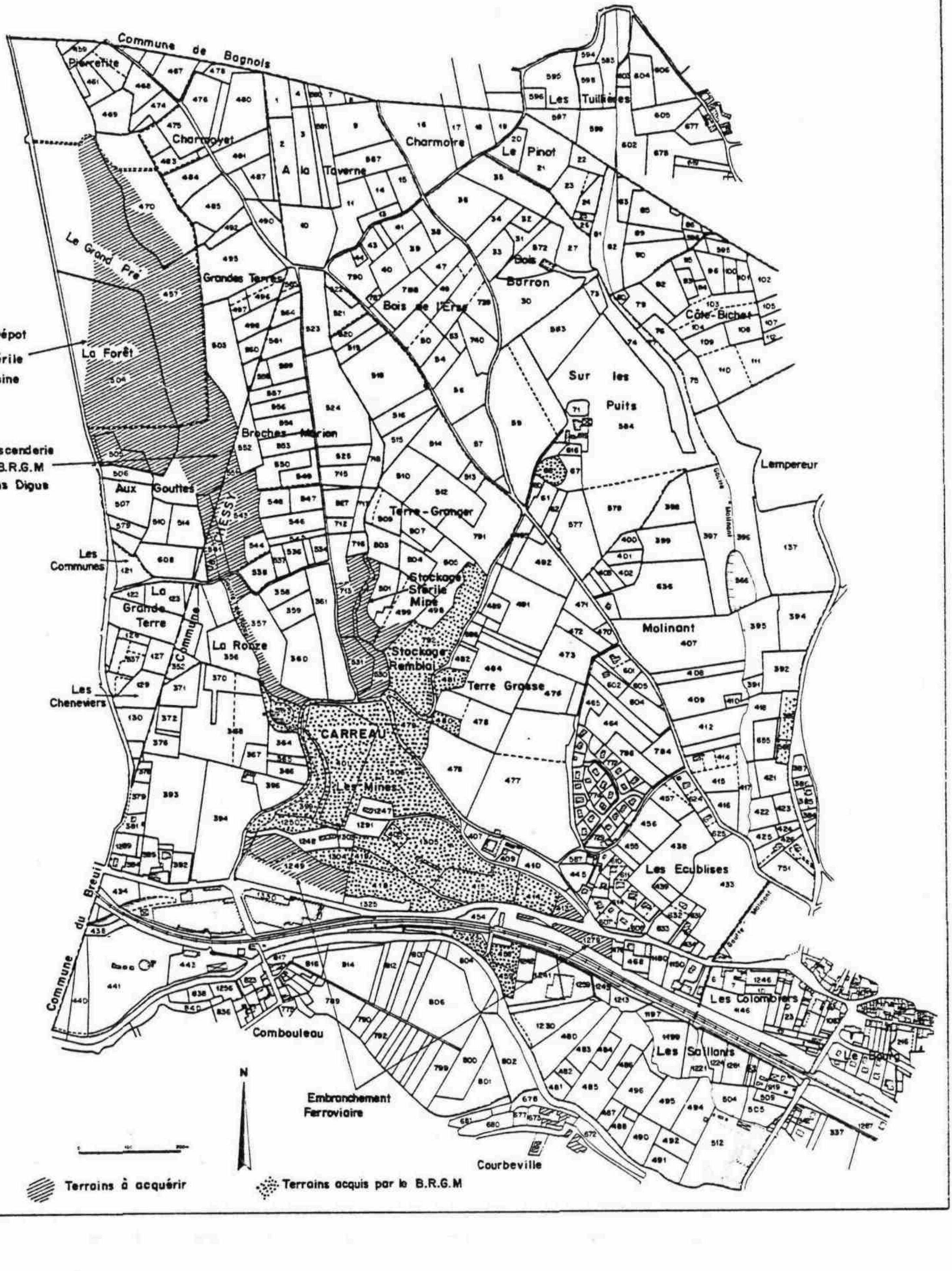
(1) Mill research and development:expenses possible carried by the PRG (Provision pour Reconstitution de Gisement).
p.m. = for the record

The capital costs are as follows:

INVCHY3

| CHESSY - CAPITAL INVESTMENT COST | CLASSES OF DEPRECIATION | | | | | TOTAL |
|---|---|---|---|---|---|---|
| Designation | Pre-Prod. Expenses | Construct Developm. | Mobile Equipment | Equipment Tools | Installat -ion | COSTS * 1000 F |
| COMMON WORKSHOPS | | 1518 | 1002 | 2250 | 963 | 5733 |
| TAILINGS DISPOSAL | | 2565 | | 469 | 509 | 3543 |
| INDUSTRIAL WATER SUPPLY | 72 | 170 | | 597 | 627 | 1466 |
| EFFLUENT CLEANING | | 110 | | 121 | 330 | 561 |
| LABORATOIRY | | 992 | | 987 | 26 | 2005 |
| ELEC. SUPPLY, SUB-STATION | 101 | | 575 | 3220 | 1787 | 5683 |
| ROADS & SERVICE NETWORKS | | 706 | | | 2532 | 3238 |
| CONCENTRATE STORAGE | | 6012 | 1268 | 232 | 700 | 8212 |
| RAILWAY SPUR | | 4974 | 548 | 254 | 1504 | 7280 |
| GEOL., SURVEY, DRAFT SERVICES | | | | 600 | | 600 |
| S/T TECHNICAL SERVICES | 173 | 17047 | 3393 | 8730 | 8978 | 38321 |

# EMPRISE DE L'IMPLANTATION INDUSTRIELLE



## XI - ADMINISTRATIVE AND SOCIAL SERVICES

The various items taken into consideration without further details, are the following:

- health and security,
- buying and stores,
- accounting and personnel affairs,
- concentrate marketing,
- office and shower cleaning,
- office block, canteen etc,
- remote surveillance,
- guards,
- management,
- head-office costs,
- land rental costs.

Broken down, these costs are as follows:

| Summary of Administrative and social services in FF/t | Personnel | Sub-contracting | Electrical energy | Consumables | TOTAL FF/t |
|---|---|---|---|---|---|
| Health and security | 1.49 | - | - | 0.70 | 2.19 |
| Buying and stores | 3.50 | - | 0.01 | 0.33 | 3.84 |
| Common offices | - | 2.73 | 0.10 | 5.36 | 8.19 |
| Concentrate sales | 2.43 | - | - | - | 2.43 |
| Office & shower maintenance | 1.65 | - | 0.16 | 0.95 | 2.76 |
| Office block, canteen etc | - | - | 0.02 | 0.04 | 0.06 |
| Head-office costs | - | 1.52 | - | - | 1.52 |
| Accounting and personnel services | 5.42 | - | - | - | 5.42 |
| Management | 2.41 | - | - | - | 2.41 |
| Land rental cost | p.m. | p.m. | p.m. | p.m. | p.m. |
| TOTAL in FF/t | 16.90 | 4.25 | 0.29 | 7.38 | 28.82 |

p.m. =for the record

| CHESSY - CAPITAL INVESTMENT COST | CLASSES OF DEPRECIATION | | | | | TOTAL |
|---|---|---|---|---|---|---|
| Designation | Pre-Prod. Expenses | Construct Developm. | Mobile Equipment | Equipment Tools | Installat-ion | COSTS * 1000 F |
| OFFICES | | 1890 | | | 1061 | 2951 |
| HEALTH & SECURITY | | | | 390 | | 390 |
| STORES | | 1307 | 264 | | 99 | 1670 |
| SHOWERS & CLEANING | | 1553 | | 58 | 1185 | 2796 |
| ADMIN. & SOCIAL SERVICES | | 735 | | | 210 | 945 |
| REMOTE SURVEILLANCE | | | | | 500 | 500 |
| S/T SOCIAL & ADMINISTRATION | | 5485 | 264 | 448 | 3055 | 9252 |

## XII - PERSONNEL

### 1 - LEGAL WORKING TIME

As said before, for the mine, the number of paid 8-hour shifts, covers about 245 days, which after deduction of the paid holidays of 25 working days is reduced to 220 shifts/man/year.

A normal working week in France has 39 hours, but is 35 hours for personnel working at a continuous rate.

### 2 - EVALUATION OF PERSONNEL COSTS

Pay will be according to the miners' statutes and is divided over two pay scales, one for surface personnel and the other for underground personnel.

Calculating the employers-charges/gross-remuneration for a month gives the following ranges:

| | |
|---|---|
| labourer | : 56 to 60 % |
| clerk/foreman/technician | : 50 to 60 % |
| staff | : 40 to 46 %. |

Reference scales were drawn up for personnel costs based on April 1986 rates, which include absenteeism that is directly covered by the employer, such as paid holidays, training and legal activities, whose rate was evaluated at 13.5 %.

Between April 1986 and July 1987 the reference scale increased by 1 %.

## 3 - PERSONNEL REQUIRED

The personnel requirements were determined for a working rate as follows:

- 5-day week (220 d/yr) for mine, and the technical, social and administrative services,
- 7-day week (320 d/yr) for the mill.

| DESIGNATION | Staff | Foreman technic. clerks | Labourer | TOTAL |
|---|---|---|---|---|
| MINE (61) | | | | |
| . Production/Preparation/Back-filling | 1 | 5 | 43 | 49 |
| . Electro-mechanical maintenance | 2 | 10 | 12 | |
| MILL (41) | | | | |
| . Crushing, griding, flotation, filtration | 1 | 6 | 22 | 29 |
| . Electro-mechanical maintenance | | 6 | 6 | 12 |
| TECHNICAL SERVICES (14) | | | | |
| . Laboratory | 1 | 1 | 2 | 4 |
| . Mill R & D | 1 | | | 1 |
| . Exploration/drafting/surveying | 1 | 3 | 2 | 6 |
| . Concentrate loading | | | 1 | 1 |
| . Workshop (excl. mine + mine maintenance) | 1 | 1 | | 2 |
| ADMINISTRATIVE AND SOCIAL SERVICES (23) | | | | |
| . Security | | 1 | 1 | 2 |
| . Buying and stores | | 4 | 2 | 6 |
| . Concentrate sales | 1 | 2 | | 3 |
| . Various maintenance & cleaning | | | 3 | 3 |
| . Accounting and personnel services | 1 | 6 | | 7 |
| . Management | 1 | 1 | | 2 |
| TOTAL | 9 | 38 | 92 | 139 |

## 4 - RECAPITULATION OF ALL PERSONNEL COSTS

| For a production rate of 330,000 t/an | FF/t | TOTAL |
|---|---|---|
| * MINE | | |
| . Staff | 5.67 | |
| . Development work | 6.42 | |
| . Stoping | 11.63 | |
| . Back-filling | 5.04 | |
| . Primary crushing, haulage, various | 4.31 | |
| . Maintenance | 8.23 | |
| Sub-total | | 41.30 |
| * MILL | | |
| . Staff | 5.85 | |
| . crushing II et III | 3.51 | |
| . Grinding/flotation/filtration | 8.80 | |
| . Various | 1.19 | |
| . Maintenance | 8.14 | |
| Sub-total | | 27.49 |
| * TECHNICAL SERVICES | | |
| . Laboratory | 2.96 | |
| . Mill R & D (1) | 1.14 | |
| . Concentrate loading | 0.48 | |
| . Exploration for increasing reserves (1) | 1.31 | |
| . Geology/surveying/drafting | 3.52 | |
| . Workshop (not counting mine & mill maintenance) | 2.10 | |
| Sub-total | | 11.51 |
| * ADMINISTRATIVE AND SOCIAL SERVICES | | |
| . Security | 1.49 | |
| . Buying and store | 3.50 | |
| . Concentrate sales | 2.43 | |
| . Various maintenance/cleaning | 1.65 | |
| . Personnel and accounting services | 5.42 | |
| . Management | 2.41 | |
| Sub-total | | 16.90 |
| GRAND TOTAL in FF/t | | 97.20 FF/t |

(1) expenses possibly covered by the PRG.

# *CHESSY*

# *SYNTHESIS OF COSTS*

******

# 1 - RECAPITULATION OF TECHNICAL DATA

## *1.1 - Basic data*

The economic evaluation was established on the following basis:

- Tonnage:

| | |
|---|---|
| . proven and extractable (1) | 3.645 Mt (run-of-mill) |
| . immediate potential | 2.0 Mt |

- Extraction rate 330,000 t/yr, except for year 1 (start-up) when mining rate will be 247,500 t

- Expected mine life:

| | |
|---|---|
| . Case 1 (proven ore) | 12 years |
| . Case 2 (proven and probable ore) | 18 years |

- Daily production rate:

  . 1,500 t/d run-of-mine, plus crushing I, II and III
  . 1,080 t/d run-of-minll

- Working rate:

| | |
|---|---|
| . mine | 2 shifts/d - 5 days/week - 220 days/yr |
| . mill | 3 shifts/d - 7 days/week - 321 days/yr |
| . administrative and social services | 1 shifts/d - 5 days/week - 220 days/yr |

- Mining method: cut-and-fill, and sub-level stoping with cemented back-fill

- Primary crushing underground and haulage by conveyor belt

- Ore processing:

  . primary and tertiary crushing of ore and waste,
  . two-stage grinding,
  . automated flotation of four products: Cu, Zn, pyrite, barite,
  . gravimetric separation (jigging) of Pb and Cu,
  . filtration
  . concentrage storage of about 15 days' production.

(1) Results from a geological reserve figure of 4.4 Mt, affected by dilution, the recovery factor, and the abandoning of certain panels with insufficient value in the present economic context.

- Concentrate transport by railway from a private spur line
- Tailings dam in the Ronze valley
- Industrial-water supply from pumping of the Triassic sandstone aquifer and mine dewatering

### *1.2 - Mining sequence*

The mining sequence as adopted is the following:

| | |
|---|---|
| years 1 to 4 | mining of panels 10, 20 and 30 (North Block), |
| years 5 to 11 | mining of panel 50 (South Block), |
| year 12 | mining of panel 50 (end) and panel 40 (N.Bl.), |
| years 13 to 18 | mining of southeast potential ore. |

The economic evaluation has shown the interest of mining panel 40 (ore more diluted or with lower grades) as late as possible, which will reduce the mine life of the North Block and will imply an early start of the South Block.

The infrastructure for mining panels 10, 20 and 30 is completed in years -2 and -1.

The infrastructure for mining the South Block (panel 50) is completed in years 3 and 4.

The infrastructure for mining the southeast potential ore is completed in years 11 and 12.

The yearly evolution of tonnage and ore grades of run-of-mill ore, is shown in the table of 1.3, below.

*1.3 – Ore-processing balance year by year*

| Years | | 1 | 2 to 4 | 5 | 6 to 10 | 11 | 12 | 13 to 18 |
|---|---|---|---|---|---|---|---|---|
| ORE | | | | | | | | |
| Tonnage | 1 000 t/yr | 247.5 | 330 | 330 | 330 | 330 | 330 | 330 |
| Cu | % | 2.25 | 2.25 | 2.26 | 2.27 | 2.26 | 2.02 | 2.27 |
| Pb | % | 0.32 | 0.32 | 0.30 | 0.29 | 0.29 | 0.18 | 0.29 |
| Zn | % | 8.14 | 8.14 | 9.83 | 10.68 | 10.53 | 7.26 | 10.68 |
| Ag | g/t | 18 | 18 | 20 | 22 | 22 | 20 | 22 |
| Pyrite | % | 32.99 | 32.99 | 39.35 | 42.58 | 42.63 | 43.78 | 42.58 |
| Barite | % | 28.58 | 28.58 | 26.69 | 25.73 | 25.48 | 19.83 | 25.73 |
| CONCENTRATES | | | | | | | | |
| Bulk Cu/Pb | | | | | | | | |
| Tonnage | dt | 18 117 | 24 156 | 24 189 | 24 222 | 24 090 | 21 318 | 24 090 |
| Cu | % | 28.27 | 28.27 | 28.38 | 28.42 | 28.44 | 28.84 | 28.44 |
| Recovery | % | 92.00 | 92.00 | 92.00 | 92.00 | 92.00 | 92.00 | 92.00 |
| Weight Recov. | % | 7.32 | 7.32 | 7.33 | 7.34 | 7.30 | 6.46 | 7.30 |
| Zinc | | | | | | | | |
| Tonnage | dt | 34 006 | 45 342 | 55 044 | 59 961 | 59 103 | 40 458 | 59 103 |
| Zn | % | 56.00 | 56.00 | 56.00 | 56.00 | 56.00 | 56.00 | 56.00 |
| Recovery | % | 94.48 | 94.48 | 95.07 | 95.31 | 95.28 | 94.52 | 95.28 |
| Weight Recov. | % | 13.74 | 13.74 | 16.68 | 18.17 | 17.91 | 12.26 | 17.91 |
| Pyrite | | | | | | | | |
| Tonnage | dt | 68 854 | 91 806 | 108 438 | 116 886 | 117 084 | 121 407 | 117 084 |
| Pyrite | % | 98.80 | 98.80 | 98.84 | 98.86 | 98.87 | 99.05 | 98.87 |
| Recovery | % | 83.32 | 83.32 | 82.54 | 82.23 | 82.28 | 83.23 | 82.28 |
| Weight Recov. | % | 27.82 | 27.82 | 32.86 | 35.42 | 35.48 | 36.79 | 35.48 |
| Barite | | | | | | | | |
| Tonnage | dt | 55 069 | 74 415 | 70 686 | 68 970 | 68 079 | 50 688 | 68 079 |
| $SO_4Ba$ | % | 98.00 | 98.00 | 98.00 | 98.00 | 98.00 | 98.00 | 98.00 |
| Recovery | % | 77.33 | 77.33 | 78.65 | 79.58 | 79.37 | 75.88 | 79.37 |
| Weight Recov. | % | 22.55 | 22.55 | 21.42 | 20.90 | 20.63 | 15.36 | 20.63 |

(Year 1 = production start up - dt = dry tonnes)

## 2 - RECAPITULATION OF OPERATING COSTS

The table hereafter recapitulates the mining and exploitation costs by type and section.

| OPERATING COST FOR 330,000 T/YR | Labour | Sub-contracting | Electrical energie | Consu-mables | TOTAL F/t |
|---|---|---|---|---|---|
| Mine | 41.30 | 7.13 | 3.80 | 48.57 | 100.80 |
| Mill | 27.49 | 5.00 | 15.57 | 28.22 | 76.28 |
| Technical services | 11.51 | 6.40 | 1.44 | 6.81 | 26.16 |
| Administrative and social services | 16.90 | 4.25 | 0.29 | 7.38 | 28.82 |
| For North block TOTAL FF/t | 97.20 | 22.78 | 21.10 | 90.98 | 232.06 |
| For South block | | | | | |
| . if crushing at - 40 m TOTAL FF/t | 97.20 | 22.78 | 22.24 | 91.38 | 233.40 |
| . if crushing at - 140 m TOTAL FF/T | 97.20 | 22.78 | 23.38 | 91.38 | 234.74 |
| Of which expenses potentially paid by PRG | | | | | |
| . processing research | 1.14 | - | - | 0.67 | 1.81 |
| . exploration for further reserves | 1.31 | 6.06 | 0.05 | 1.73 | 9.15 |
| TOTAL FF/t | 2.45 | 6.06 | 0.05 | 2.40 | 10.96 |

It can be seen in this table that expenses for geological exploration for further reserves have been budgeted, but that the expected increase of ore reserves has not been taken into account.

## 3 - CAPITAL INVESTMENT COST

Capital investment costs are expressed in constant 1986 French Francs.

### *3.1 - Pre-production expenses*

Such expenses include:

- complementary studies to be carried out, such as engineering design and the project team, for a total amount of 39,029,000 FF (see table hereafter) to be sperad over the 2.5 years before production start-up
- preliminary stope-access work, for the amount of 10,013,000 FF.

### *3.2 - Terrain acquisition*

It is estimated that the required ground will cost 1,596,000 FF, which may be revised downwards in view of the location.

### *3.3 - Technical capital investment*

The table hereafter gives the break-down of technical capital investment by type of immobilization. The amount of unforeseen and unspecified items is included at a level of 5 %-10 %.

the total amount comes to 191,033,000 FF, to be spread over two years.

## 4 - REPLACEMENTS AND FURTHER INVESTMENT

Under this heading fall (see table hereafter):

- Infrastructure-development work to continue mining towards the southeast.
- Replacement of rolling stock (expected life span 10,000 to 12,000 hours).
- Annual capital investment of 5 % of the value of the mill equipment.

## 5 - TAXES AND FISCAL REGULATIONS

For the record, a few essential rules are mentioned here.

### 5.1 - *Company taxes*

The present rate is 45 % of the taxable income, but this will be lowered to 42 % on January 1st, 1988.

### 5.2 - *Writing-off regulations*

| | Duration |
|---|---|
| - Ground | no writing off |
| - Founding expenses | 5 years |
| - Civil engineering works and buildings | 20 years |
| - Equipment expenses after start-up | 1 year |
| - Equipment and tools | 10 years |
| - Mining equipment | 5 years |
| - Mill and other surface-installation equipment | 10 years |

### 5.3 - *Provision for further exploration and development work (PRG in French)*

The mining company can create a "provision pour reconstitution de gisement", or PRG, that is subject to a double limitation:

- 15 % of the initial value of the concentrate at the mine site,
- 50 % of the operating profits before taxes.

### 5.4 - *Professional tax*

The Chessy mine will not be subject to this tax, but in its place will be a departmental and communal mining tax whose amount consists of:

- 43.60 FF/t Cu contained in the copper concentrate,
- 11.29 FF/t Zn contained in the zinc concentrate,
- 1.129 FF/t of pyrite concentrate.

INVCHY3

| CHESSY - CAPITAL INVESTMENT COST | CLASSES OF DEPRECIATION | | | | | TOTAL |
|---|---|---|---|---|---|---|
| Designation | Pre-Prod. Expenses | Construct Developm. | Mobile Equipment | Equipment Tools | Installat -ion | COSTS * 1000 F |
| MAIN DECLINE | 5229 | 16345 | 0 | 116 | 162 | 21852 |
| LANDING STATIONS | 3854 | | | | 164 | 4018 |
| BACK-FILL INSTALLATION | | 4727 | 3935 | 2701 | 722 | 12085 |
| UNDERGROUND PRIMARY CRUSHING | | 1073 | | 4349 | 1745 | 7167 |
| ORE HAULAGE | | 6176 | | 5944 | 981 | 13101 |
| MINE DEWATERING | | 1105 | | 801 | 600 | 2506 |
| MINE VENTILATION | | 5564 | | 550 | 620 | 6734 |
| ELECTRICITY SUPPLY | | | | 4145 | | 4145 |
| MOBILE EQUIPMENT | 526 | | 20963 | 105 | | 21594 |
| S/T MINE | 9609 | 34990 | 24898 | 18711 | 4994 | 93202 |
| ORE & WASTE HANDLING | | 1253 | | 836 | 3223 | 5312 |
| SECONDARY & TERTIARY CRUSHING | | 794 | | 3088 | 1553 | 5435 |
| CRUSHED ORE STORAGE | | 2031 | | 873 | 248 | 3152 |
| ORE GRINDING | | 5335 | | 7003 | 686 | 13024 |
| FLOTATION | 231 | 5907 | 208 | 45 | 10862 | 17253 |
| FILTRATION | | 1922 | 147 | 4830 | 1532 | 8431 |
| REAGENT PREPARATION | | 1901 | | 1787 | 531 | 4219 |
| Pb-Cu SEPARATION | | 446 | | 2159 | 840 | 3445 |
| S/T MILL | 231 | 19589 | 355 | 20621 | 19475 | 60271 |
| | | 0 | | | | |
| COMMON WORKSHOPS | | 1518 | 1002 | 2250 | 963 | 5733 |
| TAILINGS DISPOSAL | | 2565 | | 469 | 509 | 3543 |
| INDUSTRIAL WATER SUPPLY | 72 | 170 | | 597 | 627 | 1466 |
| EFFLUENT CLEANING | | 110 | | 121 | 330 | 561 |
| LABORATOIRY | | 992 | | 987 | 26 | 2005 |
| ELEC. SUPPLY,SUB-STATION | 101 | | 575 | 3220 | 1787 | 5683 |
| ROADS & SERVICE NETWORKS | | 706 | | | 2532 | 3238 |
| CONCENTRATE STORAGE | | 6012 | 1268 | 232 | 700 | 8212 |
| RAILWAY SPUR | | 4974 | 548 | 254 | 1504 | 7280 |
| GEOL.,SURVEY,DRAFT SERVICES | | | | 600 | | 600 |
| S/T TECHNICAL SERVICES | 173 | 17047 | 3393 | 8730 | 8978 | 38321 |
| OFFICES | | 1890 | | | 1061 | 2951 |
| HEALTH & SECURITY | | | | 390 | | 390 |
| STORES | | 1307 | 264 | | 99 | 1670 |
| SHOWERS & CLEANING | | 1553 | | 58 | 1185 | 2796 |
| ADMIN. & SOCIAL SERVICES | | 735 | | | 210 | 945 |
| REMOTE SURVEILLANCE | | | | | 500 | 500 |
| S/T SOCIAL & ADMINISTRATION | | 5485 | 264 | 448 | 3055 | 9252 |
| TOTALS | 10013 | 77111 | 28910 | 48510 | 36502 | 201046 |

### *5.5 – Value-added tax (T.V.A.)*

* Concerning capital investment: TVA will be reimbursed within a period of 7 months. For this reason, a treaswry advance corresponding to 18.6 % of 7 months of average capital investments, must be reckoned with. It will be important, in view of the fact that the investment period may stretch beyond 24 months without income, to negotiate this fact with the tax authorities.

The corresponding financing cost for the treasury is about 1.3 % of the capital-investment amount.

* Concerning exploitation costs: TVA to be collected on any concentrate sold in France is levied at the rate of 18.6 %; this is essentially for the zinc concentrate. For exported concentrate it is zero.

It can be seen that the TVA collected (at the prices considered) is always more than the deductible TVA; the treasury thus benefits form a slight extra.

-TABINV9

CHESSY_ ANNUAL CAPITAL COST REQUIREMENT IN THOUSAND FRANCS (1986 value)

| YEAR | TOTAL | -3 | -2 | -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | CUMULE |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| LAND ACQUISITION | 1596 | | 1596 | | | | | | | | | | | | | | |
| PRELIMINARY EXPENSES | | | | | | | | | | | | | | | | | |
| exploration expenses | p.m. | | | | | | | | | | | | | | | | |
| complementary studies / engineering / project team | 39029 | 11555 | 17066 | 10408 | | | | | | | | | | | | | |
| prelim. stope access | 10013 | | 5007 | 5007 | | | | | | | | | | | | | |
| TOTAL | 49042 | 11555 | 22073 | 15415 | | | | | | | | | | | | | |
| TECHNICAL INVESTMENT | | | | | | | | | | | | | | | | | |
| -construction | 77111 | | 38556 | 38556 | | | | | | | | | | | | | |
| -mobile equipment | 28910 | | 14455 | 14455 | | | | | | | | | | | | | |
| -fixed equipment & tools | 48510 | | 24255 | 24255 | | | | | | | | | | | | | |
| -installation | 36502 | | 18251 | 18251 | | | | | | | | | | | | | |
| TOTAL | 191033 | | 95517 | 95517 | | | | | | | | | | | | | |
| EXTENSIONS -REPLACEMENTS | | | | | | | | | | | | | | | | | |
| mine development | | | | | | | | | | | | | | | | | |
| -tunnelling | | | | | 1028 | 1028 | 22093 | 20616 | 2048 | 2048 | 0 | 2048 | 0 | 2048 | 0 | 0 | 52957 |
| -equipment | | | | | 1104 | 1104 | 8877 | 8587 | 290 | 290 | 0 | 290 | 0 | 290 | | | 20832 |
| mobile equipment replac. | | | | | | | | | | | | | | | | | |
| -jumbo minipantofore | | | | | | | | | | | | | | | | | 0 |
| -jumbo horiz.& vert.dril | | | | | 1500 | | 1500 | | 1500 | | 1500 | | 1500 | | | | 7500 |
| -jumbo :bolting | | | | | | | | | | | | | | | | | 0 |
| -scoop CT2501 | | | | | | | | | | | 680 | | | | | | 680 |
| -scoop bucket 7-10t | | | | | 1200 | | 0 | 1200 | | 0 | 1200 | | | 1200 | | | 4800 |
| -elevating platform | | | | | | | 430 | | | | | 430 | | | | | 860 |
| -hydraulic rock breaker | | | | | | | | | | | | 500 | | | | | 500 |
| -slinger belt truck | | | | | | | 1000 | | 1000 | | | 1000 | | | | | 3000 |
| -various vehicles | | | | | | | | 200 | | 200 | | 200 | | 200 | | | 800 |
| -concentr. truck loader | | | | | | | | | | | | 1200 | | | | | 1200 |
| mill | | | | | | | | | | | | | | | | | |
| -5% of equipments | | | | | 1050 | 1050 | 1050 | 1050 | 1050 | 1050 | 1050 | 1050 | 1050 | 1050 | | | |
| Pb/Cu separation | | | | | | | | | | | | | | | | | |
| tailings disposal | | | | | | | | | | | | | | | | | |
| TOTAL | | | | | | | | | | | | | | | | | |
| -depreciation 20 years | | | | | 1028 | 1028 | 22093 | 20616 | 2048 | 2048 | 0 | 2048 | 0 | 2048 | 0 | 0 | 52957 |
| -depreciation 10 years | | | | | 2154 | 2154 | 9927 | 9637 | 1340 | 1340 | 1050 | 1340 | 1050 | 1340 | 0 | 0 | 31332 |
| -depreciation 5 years | | | | | 2700 | 0 | 2930 | 1400 | 2500 | 200 | 3380 | 3330 | 1500 | 1400 | 0 | 0 | 19340 |
| WORKING CAPITAL | | | | | | | | | | | | | | | | | |
| -store stocks | | | | | | | | | | | | | | | | | |
| -concentrate stocks | | | | | | | | | | | | | | | | | |
| -receivables | | | | | | | | | | | | | | | | | |
| -suppliers credit | | | | | | | | | | | | | | | | | |
| -cash | | | | | | | | | | | | | | | | | |
| - | | | | | | | | | | | | | | | | | |
| TOTAL | | | | | 1994 | | | | | | | | | | | | |
| operating cost F/T | | | | | 221 | 221 | 221 | 221 | 222 | 222 | 222 | 222 | 222 | 222 | 222 | 207 | |
| expenses charged on PRG | | | | | 11 | 11 | 11 | 11 | 11 | 11 | 11 | 11 | 11 | 11 | 11 | 11 | |
| TOTAL :OC + PRG F/T | | | | | 232 | 232 | 232 | 232 | 233 | 233 | 233 | 233 | 233 | 233 | 233 | 218 | 2780 |
| SNCF refund 000 F | | | | | -1000 | -1000 | -1000 | -1000 | -1000 | | | | | | | | |
| mining tax F/T | | | | | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | |
| marketing expenses | | | | | | | | | | | | | | | | | |