4135 INICHAR

UNDERGROUND GASIFICATION IN VARIOUS COUNTRIES - INICHAR REPORT (CONT

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TRANSLATION

Foreign

TITLE : – as translated into ENGLISH

UNDERGROUND GASIFICATION IN VARIOUS COUNTRIES
INICHAR REPORT
(CONTINUATION)

II. EXPERIMENTS IN ITALY
III. EXPERIMENTS AT GORGAS (U.S.)

– as translated from FRENCH

La gazéification souterraine dans les divers pays
RAPPORT D'INICHAR
(SUITE)

II. — Experimenten in Italië.
III. — Experimenten te Gorgas (U.S.A.)

AUTHOR/S/ : INICHAR

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UNDERGROUND GASIFICATION IN VARIOUS COUNTRIES

INICHAR REPORT

(CONTINUATION)

II. EXPERIMENTS IN ITALY

III. EXPERIMENTS AT GORGAS (U.S.)

INICHAR

II. ITALIAN TESTS

A. Valdarno. 12 July-30 August 1947. (Castelnuovo dei Sabbioni)

1) *Arrangement of the site*

The test was performed in the lenticular deposit of lignite at Banco-Casino, covered by a layer of 12 to 30 m of dead ground.

The seam had a dip of 12° and a thickness varying from 5 to 20 m. It had been worked previously in its upper portion, so that the roof formed by the stopes of the old working were highly irregular.

The stratification of the lignite is very capricious and many seams of clay contaminated the coal seam. The floor of the coal seam consists of eocene sand and the roof of pliocene clay. The distillation of this clay releases a certain amount of gas consisting especially of H_2, CO and CH_4 and able to enrich the gas formed in the seam during the test.

The lignite ashes are softened at 1030-1100° and fuse at 1270°.

The composition of the fuel, also very variable, follows from the analyses below:

1 Analyse immédiate					2 Analyse élémentaire					
3 PCS	4 Humidité	5 Cendres	6 MV	7 CF	C	H	O	N	S	8 PCI
9 tout venant		10 sur sec	11 sec, sans cendres		12 sec, sans cendres					
2.288	50.55	25	53.3	46.7	50.7	6.5	39.4	1.7	1.0	5.850

<u>KEY</u>

1. Proximate analysis
2. Ultimate analysis
3. g.c.v.
4. Moisture
5. Ash
6. Volatile matter
7. Fixed carbon
8. n.c.v.
9. Run-of-mine
10. Relative to dry
11. Dry, without ash
12. Dry, without ash

This lignite, dried to 37% moisture, yields an air-gas of the following mean composition in a Körpell gas generator:

CO_2	O_2	CO	H_2	CH_4	N_2	g.c.v.	n.c.v.
0.7	1.0	25.0	11.0	1.5	54.8	1.250	1.105

By distillation at 600-900°, it supplies per ton:

560 liters of water (of which 500 correspond to the moisture in the fuel),

20 kg of tar,

1.7 to 2.8 kg of benzine (according to the distillation temperature),

1 to 1.5 kg of ammonia,

270 kg of coke,

145 to 180 m^3 of gas having a composition varying little with the distillation temperature:

CO_2	O_2	CO	H_2	CH_4	C_nH_m	N_2	g.c.v.	n.c.v.
20.7	1.2	13.6	15.1	13.9	1.0	1.5	5.200	2.860

The lignite in place contained 50% moisture and represented only 1870 cal/kg of net calorific value. Moreover, there was a variable but continuous inrush of water into the fire passage.

Furthermore, old workings performed in the seam decreased its leaktightness.

The site had a horseshoe shape. Two descending passages, designed for air inlet and gas return, had been hollowed out starting from the overburden, following the dip of the seam (about 12°) over a length of 60 m (Fig. 21).

A transverse passage (60 m) called the fire passage connected the ends of the two preceding passages, thereby delimiting a panel of trapezoidal shape containing about 7200 tons of lignite.

Another passage parallel to the gas passage was used as a pump room.

This network of passages had been constructed so as to leave a thickness of 2 to 4 m of lignite below the bottom level and a thickness of 3 m at the crown. In fact, the latter thickness varied between 0 and 10 m due to irregularities in the seam.

The inlet and outlet passages were each blocked 20 m from the inlet by a dam which traversed the air and gas conduits.

Twenty-eight thermocouples had been placed along the fire passage (no. 1 to 18) and gas passage (no. 18 to 26), either in the gas stream or in the walls or roof of the lignite at a depth of 1 m.

To prevent the spreading of the fire beyond the panel delimited by the three passages, the outside wall of those passages

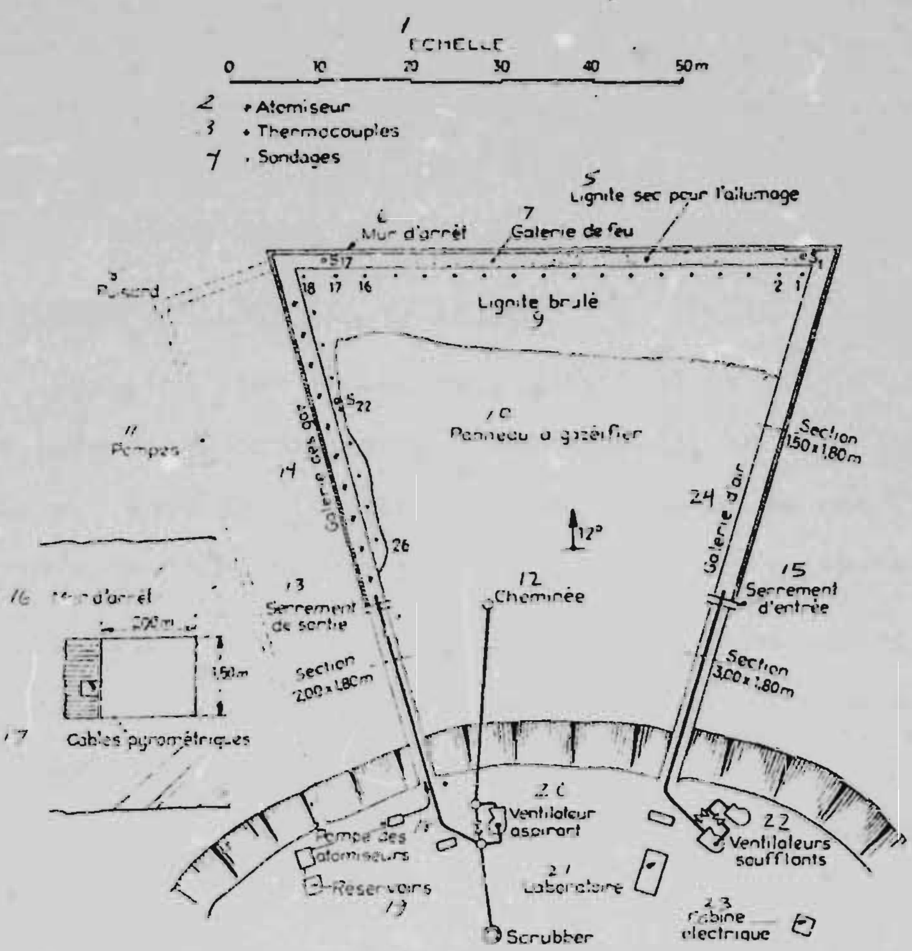


Fig. 21

KEY

1. Scale
2. Atomizer
3. Thermocouples
4. Boreholes
5. Dry lignite for ignition
6. Stop wall
7. Fire passage
8. Sump
9. Burnt lignite
10. Panel to be gasified
11. Pumps
12. Chimney
13. Outlet dam
14. Gas passage
15. Inlet dam
16. Stop wall
17. Pyrometric cables
18. Pump for the atomizers
19. Tanks
20. Aspirating ventilator
21. Laboratory
22. Blowers
23. Electrical cabin
24. Air passage

had been covered by a stop wall. In this wall there had been cut a notch to accomodate the pyrometric cables. Atomizers cooled the gas in the return passage in order to protect the conduit and dam. They could possibly serve as gas taps for analyses.

2) *First ignition (12-17 July 1947)*

Ignition took place on 12 July 1947 at 17 h 25, by means of three incendiary bombs ignited electrically at intervals of a few minutes. Some dry lignite had been piled up in the fire passage up to 10 m from the gas passage in order to initiate combustion.

As soon as the bombs ignited, smoke appeared at the chimney. At around 19 hours, the pyrometers in the fire passage began to detect temperature elevations.

When ignition of the bombs was completed, the flow rate was raised to 10,800 m^3/h. This intense stream produced too rapid combustion of the dry priming lignite and immediately drove the fire downstream to the gas passage.

This was manifested by the fast rise of pyrometer 17 (from 110° at 3 hours in the morning of 13 July to 880° at 6 hours) and by the increasing temperature of the gas leaving the site (from 37° at 3 hours to 350° at 9 hours). This gas contained 16 to 18% CO_2 and 2% O_2 and a little CH_4 and nitrogen.

Several maneuvers were attempted to return the fire upstream: reducing the flow rate, starting the atomizers, igniting the ignition bombs remaining at the head of the site, but they had no effect other than causing the appearance of an oxygen excess in the outgoing gas at the expense of the CO_2 percentage.

It was not possible to eliminate this residual oxygen, even by re-establishing the original flow rate of 10,000 m^3/h.

The fire continued to develop in the gas passage and reached 6.50 m from the outlet dam.

It was then decided to extinguish the site by keeping the atomizers in operation and allowing the water to rise in the passages and by not halting the blowers until complete flooding had occurred.

On Wednesday, 16 July, it was found that the water was at a level 0.50 m higher than the initial roof of the fire passage. However, an air flow rate of 700 m^3/h passed under a pressure of 120 mm, while the pyrometers in the gas passage always indicated the presence of fire. Thus, it is probable that the fire had encroached upon the roof of the passages and had caused cave-ins there providing a passage to the gas stream.

The blowers were stopped at 18 h 10.

The period of forced rest from 17 to 22 July was put to good use to perform auxiliary operations and repairs. A leak at the outlet dam was plugged. A borehole (S_1) designed for reignition of the site was made at the entrance of the fire passage. Another borehole (S_{22}, close to pyrometer 22) was made into the gas passage to send water there. This latter borehole cut across an excavation originating from an old working.

Despite the sending of water and the injecting of 200 m^3 of CO_2 into the gas passage, the pyrometers always indicated (in a highly irregular manner) a certain temperature. It is probable that their indications were inexact (false contacts between wires). In the fire passage, increasing temperatures were also noted during this period (ascent of the fire?).

3) *Second ignition*

The reignition occurred on 23 July around noon through borehole S_1. It was difficult and succeeded only after several failures (Bickford cord, gasoline, thermite bomb). Success was finally achieved by igniting some dry lignite soaked with gasoline by means of a cloth ignited outside the hole.

Since the ground was very moist (due to a heavy rainfall some days before), it was necessary to supply this pile continuously with dry lignite thrown into the hole. A great deal of steam emerged from the borehole.

The air flow rate was 700 m^3/h.

The reheating of the site was very slow and lasted until 28 July. During this period, the flow rate was increased gradually, which had the effect of increasing the CO_2 content of the fumes (at the expense of O_2) and causing the gas outlet temperature to rise.

	Q air m^3/h	CO_2 %	1 Temp. sortie	
7-23	700	—	—	
7-24	700-1.000	4.4	—	2 2.250 m^3/h pendant 4 heures.
7-25	1.500	6.4	70°	3 Les pyromètres commencent à monter — A partir de 18 h : coups d'air périodiques de 3.000 m^3/h.
7-26	1.800	16.6	—	4 Coups d'air de 6.500 m^3/h.
7-27	3.500	4.9	—	
	4.500	19.4	100°	5 Coups d'air toutes les 20 min.
7-28	3.000	—	300°	6 Gaz à 440 cal.

KEY

1. Outlet temperature
2. 2250 m^3/h for 4 hours.
3. The pyrometers begin to rise. Beginning at 18 h: periodic blasts of air at 3000 m^3/h.
4. Blasts of air at 6500 m^3/h.
5. Blasts of air every 20 minutes.
6. 440-cal gas.

On 26 July, there was installed at the bottom of borehole S_1 a gas oil burner intended to activate the pile.

On 28 July after several pluggings of the borehole and burner, the burner was deactivated and the supply to the pile was halted. A new borehole, S_{17}, was drilled close to the downstream end of the fire passage.

On 27, 28 and 29 July, the gradual increase of the air flow rate sent into the mine produced a continual improvement of the gas. Unfortunately, it was necessary to restrict this flow rate due to the poor strength of the pipes.

On 29 July, a g.c.v. of 700 cal/m^3 was attained, while the daily mean was established at 550 cal/m^3 at a flow rate of 6000 m^3/h. This gas did not burn, however, even after having been stripped of its moisture and CO_2.

From that day on, the residual oxygen disappeared completely from the analyses (see graph in Fig. 22).

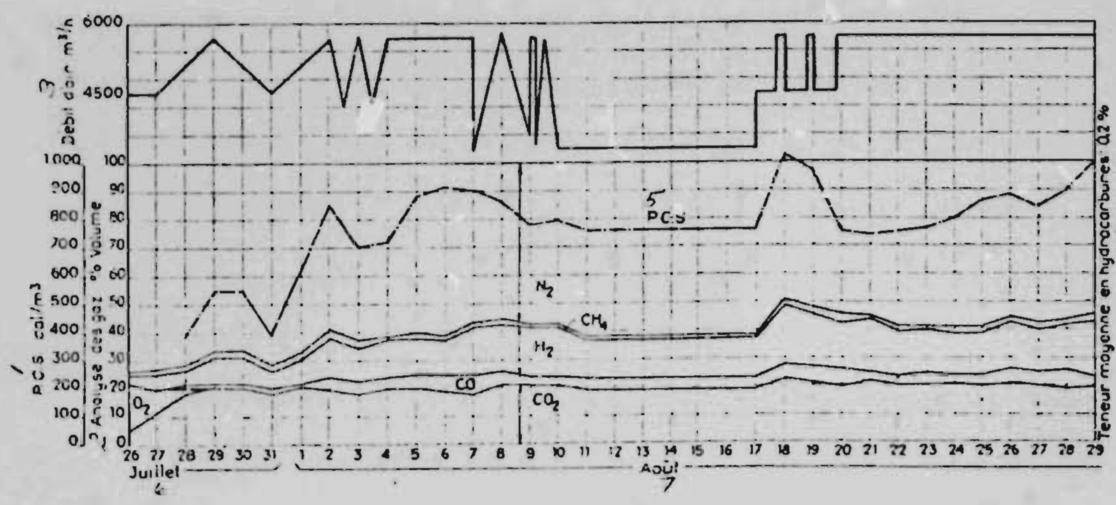


Fig. 22

KEY

1. g.c.v.
2. Gas analysis
3. Air flow rate
4. Mean hydrocarbon content: 0.2%
   Mean oxygen content: 0.2%
5. n.c.v.
6. July
7. August

On the following days it was necessary again to reduce the flow rate due to leaks which caused the formation of parasitic fires in the gas passage outside the dam. It was necessary to close the entrance of the passage by causing a cave-in of the overlying ground. Furthermore, the equivalent orifice of the site had decreased sharply relative to its initial value.

This decrease of the flow rate caused a lowering of the calorific value of the gas to 400 cal/m^3.

Beginning on 31 July it was possible to increase the blowing. The calorific value rose again immediately and on 2 August at 5 h the gas was ignited and continued to burn without interruption until the end of the experiment. Its gross calorific value, calculated according to the analysis, was 859 cal/m^3 at that time.

The equivalent orifice had decreased again: A pressure of 1000 mm of H_2O was necessary to enable 5700 m^3/h to pass through. The greatest portion of this pressure drop occurred in the downstream part of the site (gas passage). Actually, there was a pressure of 700 mm of H_2O at borehole S_{17}.

On 3 August, 928 cal/m^3 was obtained (g.c.v.). The ground began to crack around borehole S_1 where 15 m of dead ground covered 10 m of lignite. From the position of the crevices, the fire seemed to have passed above the protection floor of the fire passage. Around the borehole, the surface of the soil had sagged in the form of a conical basin.

3 and 4 August were devoted to tests with pulsating flow rates (alternately 6000 m^3/h and 3000 m^3/h, with a cycle time of 6 hours for each phase).

Since the results were hardly encouraging (g.c.v. = 700 cal/m^3), the blowing was resumed at the maximum rate and from 5 to 7 August the calorific value oscillated between 900 and 1000 cal/m^3, with approximately 6% CO, 15% H_2 and 2% CH_4.

Starting on 6-7 August, the equivalent orifice stabilized around 1.5 dm^2. The pressures were distributed in a more regular manner:

1 Pression	2 Entrée	3 Au S_1	4 Au S_{17}	5 Au S_{22}	6 Sortie
mm H_2O	1.006	1.006	550	122	40

KEY

1. Pressure
2. Inlet
3. At S_1
4. At S_{17}
5. At S_{22}
6. Outlet

The gas emerged at 84° and contained 385 to 630 g of water per m^3.

On 8 August, a halt of the electric motors to perform work caused a drop of the flow rate to 1800 m^3/h for 20 minutes. The gas which yielded 860 cal/m^3 before the halt displayed the following composition:

CO_2	O_2	CO	H_2	CH_4	C_nH_m	N_2	H_2O	g.c.v	n.c.v.
20	0.5	5.2	16	13.2	0.5	55.8	555g/m^3	2.155	1.018

The return to the normal flow rate of 4500 m^3/h was effected without incident.

On the next day, a test with slow decrease of the flow rate (from 4550 m^3/h to 1800 m^3/h in 5 hours) gave no results. The calorific value oscillated between 650 and 900 cal/m^3 without exhibiting any specific relation to the flow rate.

On 10 August, a flow-rate decrease test was done again. The flow rate was reduced suddenly to 1000 m^3/h. After a quarter of an hour, the gas flowed back through the air inlet passage. Gradual re-establishment of the normal flow rate caused an explosion.

On 11 August, the same phenomena occurred following a current outage: reflux of gas and explosion when starting up after six hours of halt. It appears that a flow rate of at least 2000 m^3/h was necessary continually to prevent an inversion of the stream and the mixture of air and gas.

The equivalent orifice of the site, which was 2.1 dm^2 on 1 August, had decreased appreciably during the active period from 1 to 12 August.

Around 6 August, it was stabilized at 1.5 dm^2 and, around 12 August, it varied between 1.45 and 1.9 dm^2. Determinations performed with a high flow-rate generally resulted in a larger equivalent orifice than the one obtained with smaller flow rates. This indicates that the flow rate Q does not vary strictly proportionally to the 1/2 power of the pressure difference h applied to the ends of the circuit, but according to a power of h between 0.5 and 1.

From tests performed on 11 August it was possible to deduce the following characteristic (see Table below):

$$Q\ m^3/h = 148\ h^{0.55}_{mm\ H_2O}$$

Date	1 Debit m^3/h	2 Pression mm H_2O	3 Or. Eq. dm^2
1-8	6.500	950	2.1
2-8	5.700	950	1.85
4-8	5.000	950	1.62
•	4.500	950	1.5
9-8	7.250	1.600	1.87
•	5.000	900	1.68
11-8	4.500	757	1.61
•	6.800	1.367	1.89
•	4.500	860	1.58
	5.000	518	1.44

KEY

1. Flow rate
2. Pressure
3. Equivalent orifice

4) *Last period*

From 11 to 22 August, the site was "put on ice" with a flow rate of 3000 m^3/h in order to prevent current outages from causing explosions and to permit the mounting of emergency blowers activated by explosion motors. In this operating mode, the calorific value of the gas was maintained at about 750 cal/m^3. During the night, however, when current outages were less likely, it was possible to send in larger flow rates with corresponding improvement of the gas. On 19 and 20 August, 1000 cal/m^3 was exceeded.

It was found that the ratio of the gas flow rate to the air flow rate had varied: Thus, gas leaks had gained in importance. The equivalent orifice had increased slightly during this rest period: about 2 dm^2.

Normal operation was re-established on 22 August at 25 hours with a flow rate of 6000 m^3/h. The gas emerged from the mine at 85° and contained 437 g H_2O/m^3.

This operating mode was maintained until 29 August. The gas improved throughout this entire period, finally reaching 1000 cal/m^3. Several interruptions of short duration were caused, however, by deficiencies of the electrical power supply. Each time, the combustible gases rose again in the air passage and mixed with the combustion agent to cause explosions when starting up again.

On 29 August at 3 h 20 in the morning, a voltage drop blew the fuses of the installation, thereby immobilizing the operating blowers. The gases immediately flowed back into the air passage and through the regulators so that there was not enough time to put the emergency blowers into operation. Twelve minutes after the halt, a violent explosion occurred.

After the fuses were repaired, reactivation of two blowers caused abundant fumes to emerge from the air passage.

Putting an exhauster into operation at the gas outlet re-established the normal circuit, but caused a second explosion. The lignite burned above the air inlet dam.

Since the exhauster was soon put out of operation, it was decided to halt the site. Steam was sent in while caving in the ground on the air passage in order to plug it and allowing the inrush of natural water to inundate the underground workings.

5) *Inspection of the site*

The extinguishing operations lasted from 3 to 30 September. Unwatering then occupied the period from 15 October to 10 December. It was then possible to penetrate into the air passage which was intact to 25 m beyond the dam. At that location it was filled with an accumulation of ash containing pockets of unburnt lignite which obstructed the entire section. The protection wall of the exterior wall had resisted perfectly. Furthermore, the fire had not descended below the level of the bottom of the passage.

Unfortunately, cave-ins occurring at the entrance of the passage prevented the prospecting from being pushed on further.

On 15 June 1948 it was possible to penetrate into the gas passage which also remained intact to 30 m from the dam. As in the air passage, the external wall and the bottom of the passage were intact.

Thus, the panel had burned over a depth of 10 m (gas passage) to 15 m (air passage).

6) *Conclusions*

a) A combustible gas with a mean g.c.v. of 820 cal/m^3 was produced continuously for 28 days (from 2 to 29 August). During some periods when a sufficient constant air flow rate was maintained successfully, higher mean values were obtained:

from 5 to 7 August	(3 days)	903 cal/m^3
from 18 to 20 August	(3 days)	950 cal/m^3
from 18 to 29 August	(12 days)	860 cal/m^3

If we attempt to set up a balance of materials for the period from 2 to 29 August during which combustible gas was produced and if we start out from the ultimate composition of the lignite given at the beginning of this chapter and the mean composition of the gas for that period as shown in the Table:

CO_2	O_2	CO	H_2	$CH_4 + C_nH_m$	N_2	g.c.v.	n.c.v.
10.7	0.2	4.5	15.0	2.2	57.8	810	725

we obtain the following figures:

<table>
<tr><td rowspan="2">Elements of the balance per Nm^3 of gas</td><td>Inputs</td><td>Fuel
c= 0.406 kg

Combustion agent
a=0.727 Nm^3

H_2O reduced
h= — Nm^3</td><td>Distillation
877 cal

Air-gas
66 cal

Water-gas
cal

Combustion
-218 cal

n.c.v. of gas
725 cal
30.4 %</td><td>Sensible heat of gas
285 cal
12 %</td><td>Total heat extracted
1010 cal
42.4 %</td></tr>
<tr><td>Outputs</td><td>H_2O formed
-h=0.085 Nm^3

Unburnt residues
k=0.065 kg</td><td>Unburnt residues
8100 x 0.065 = 525 cal
22.1 %</td><td>Losses to ground
845 cal
35.5 %</td><td>Heat remaining in the mine
1370 cal
57.6 %</td></tr>
<tr><td colspan="2">Yields</td><td>η_c = 68.5 %

η_{th} = 39 %</td><td>Total potential heat
1250 cal
52.5 %</td><td>Total sensible heat
1130 cal
47.5 %</td><td>n.c.v. of fuel
0.406 x 5850 =
2380 cal
100 %</td></tr>
</table>

The high value of losses to the ground and, consequently, the low value of the thermal yield are due largely to the high content of ash and moisture in the fuel. Evaporation of the fuel required 0.5055 x 640 = 320 calories per kg of rough lignite, i.e. 865 calories per kg of clean lignite and about 350 calories per Nm^3 of gas.

Furthermore, the net calorific value of the fuel, which we used in the calculations, was not calculated as usual from the ultimate analysis. The high oxygen content of this lignite actually made suspect the use of the ordinary "grouping formula". We calculated the n.c.v. from the g.c.v., determined experimentally and corrected as a function of the hydrogen content of the clean lignite. The figure obtained seems abnormally high if we compare it with the composition of the fuel. Since it enters the calculation as a difference of heat losses, it may have influenced the yield.

The low CO/CO_2 ratio indicates that the gasification was not very active and that the temperature was very low, which is hardly surprising, considering the quality of the fuel.

According to the balance indicated above, a total of 1470 tons of clean lignite should have been consumed from 27 July to 30 August, of which 1000 tons would have been gasified or burned completely and the remainder (470 t) only being distilled.

This corresponds to 4000 tons of rough lignite, of which 2720 tons would have been used completely, providing a total gas production of 3,900,000 m^3.

b) The calorific value in the steady-state improves with increasing air flow rate. It can rise up to 1000 cal/m^3 with a flow rate on the order of 6500 m^3/h.

A sudden halt or a sudden decrease of the flow rate can provide a much better distillation gas (2000 cal/m^3), but only temporarily. If the flow rate is maintained at a low value, the calorific value soon drops.

c) An exaggerated flow rate 10,000 m^3/h drives the fire downstream, at least in a passage whose cross-section has been reduced by a piling of lignite in pieces.

d) During halts in blowing, explosions must be anticipated due to the mixing of air and gas. It is necessary to maintain continuously a minimum flow rate on the order of 2000 m^3/h.

e) The equivalent orifice is stabilized around 1.5-2 dm^2. It appears to undergo appreciable variations during halts in blowing. The characteristic of the mine is of the form $Q = K h^a$, where the exponent a varies between 0.5 and 0.85.

f) It is difficult to monitor the displacement of the fire. The pyrometers give hardly any precise information.

B. Terni

Another test was attempted during the following winter at Terni (Umbria), in the lignite deposit at Colle dell'Oro, composed of three seams of xyloid lignite between banks of pliocene clays and sands.

The seams are separated by 20 m and have a thickness of

1.80 m and a dip of 8 to 10°. They are soiled by shale breaks. It is especially the upper seam that was worked and it is in that seam that a gasification panel was laid out.

A proximate analysis of the lignite yielded:

1 Humidité	2 Cendres	3 MV	4 Carbone fixe	5 PCS
6 tout venant	7 sur sec	8 sur charbon net		
41.1	34.0	63.7	36.3	5.100

KEY

1. Moisture
2. Ash
3. Volatile matter
4. Fixed carbon
5. g.c.v.
6. Run-of-mine
7. Relative to dry
8. Relative to clean coal

The site was established beneath a hill and had a layout similar to that at Valdarno. The air passage measured 57 m with a dip of 10° and was reinforced with wood frames (useful cross-section: 2.6 m^2). The horizontal fire passage was 92 m long and was also furnished with wood frames. The wall opposite the panel was protected by a wall of clay reducing the cross-section to 1.7 m^2. The gas passage was 91 m long and had a dip of 9° and a cross-sectional area of 2.2 m^2. The two walls were protected by clay walls. These passages were excavated through the entire thickness of the seam. Two dams closed the air and gas passages. A third dam isolated the fire passage from an auxiliary passage.

Forty chromel-alumel thermocouples, protected by stainless steel sheaths, had been placed along the gas circuit and in the walls of the passages.

The air and gas flow rates were measured by diaphrams and the gas analyses were performed with the Orsat. Four blowers mounted in series or in parallel had to supply an air flow rate greater than at Valdarno.

Several piles of dry wood had been stacked at the entrance of the fire passage and air passage to provide ignition and, possibly, a return of the fire upstream.

The site was ignited on 25 November 1947 and remained in activity until the second half of December. During that period, such large leaks occurred that it was necessary to suspend the tests. A new panel had to be laid out inside the preceding one.

However, this first phase made it possible to ascertain that:

- The ignition was effected easily.
- The pyrometers in the fire passage made it possible to trace the extension of the fire from the ignition points:
  10 m in three days,
  20 m in eight days,
  40 m in ten days.
- The subsidence of the roof induced in the fire passage favored the production of gas.
- The production of gas seems to be favored by high flow rates.
- A greatly reduced flow rate can cause the fire to retreat upstream.
- The thickness of the lignite protection mass should be large in order to be able to assure the leaktightness

of the site. Likewise, it is necessary to make sure that the lignite does not cause leaks around the dams while drying.

- It is possible to maintain the site "on ice" for a long time; reignition is effected rapidly.

The gas produced during this first period had a mean value of 400 cal/m^3. A maximum of 600 cal/m^3 was obtained with air blowing and even 825 cal/m^3 by allowing the distillation gas to be released. A gas tap taking specimens in the fire passage even provided 1320 cal/m^3 gas in a regular manner. But this gas always deteriorated before reaching the outlet of the site.

* * *

The test was resumed on a small panel cut inside the preceding one during the summer of 1948 and the installation operated for two months. Unfortunately, re-entrances of air due to nearby old workings caused a recombination of the gas in the bottom and it was impossible to obtain anything other than combustion gases at the outlet.

III. GORGAS TESTS (UNITED STATES)

The tests organized by the United States Bureau of Mines and the Alabama Power Company at Gorgas (25 km south of Jasper, Alabama) were performed in the Pratt layer composed of two bands of fat coking coal with a total thickness of 90 cm.

Upper band	10 ... 20 cm
Intercalation	5 cm
Lower band	65 ... 75 cm

The seam is practically horizontal (dip less than 1%). It is finely divided by vertical cleavages separated by about 15 mm. Near the outcrops, these cleavages are soiled with clay. The ashes are distributed in very thin beds (on the order of a tenth of a mm) and closely spaced so that this coal is very difficult to wash.

Composition of the coal (average of 6 specimens):

1 Analyse immédiate				2 Analyse élémentaire					6 PCS	
H_2O	3 Cendres	4 CF	5 MV	C	H	O	N	S		
7 Tout venant	8 sur sec	9 Charbon net							7 Tout venant	9 Charbon net
4.1	14.5	61.9	38.1	84.0	5.5	7.5	1.7	1.5	6.850	8.350

KEY

1. Proximate analysis
2. Ultimate analysis
3. Ash
4. Fixed carbon
5. Volatile matter
6. g.c.v.
7. Run-of-mine
8. Relative to dry
9. Clean coal

Characteristics of the walls and ash:

	First deformation	Softening	Fusing
Roof	1260°	1350°	1470°
Floor	1300°	1400°	1430°
Ash	1380°	1450°	1500°

The roof and floor consist of schists with sandy seams at some distance from the roof. The roof is a little carbonaceous

(10%), but the organic matter is finely divided and dispersed in the mass. There are carbonaceous bands in the floor a few dozen centimeters beneath the seam. The "America" layer is found about 20 m further down.

A. Test no. 1

The location selected for the first test is the peak of a hill where the seam crops out on three sides (east, south, west). The fourth side was separated from the rest of the deposit by a 6-m-wide trench. The outlier thus isolated measured about 120 m in length and 60 m in width and was covered by 10 m of dead ground. Old workings were located 100 m further north, in the Pratt vein, and 25 m to the north, in the America vein.

The panel to be gasified was delimited by two parallel passages excavated from the trench and by a transverse passage. It measured 45 m in length and 12 m in width (Fig. 23). These passages had a cross-section of 1.80 m over 90 cm (height of the seam). However, bags of clay had been piled against the outside wall, leaving only a free passage of 1 m to the roof and 0.60 m to the floor. They were provided with a timbering of 28 kg/m rails set into the coal walls (Fig. 24).

Concrete dams (1 part fast-setting cement, 2 parts sand and 4 parts ground cinders) closed the two passages about 3 m from the entrance. They penetrated about 40 cm into the coal, 30 cm into the floor and 15 cm into the roof of the seam.

After removing the coffers, all the crevices due to shrinkage were plugged by injecting cement.

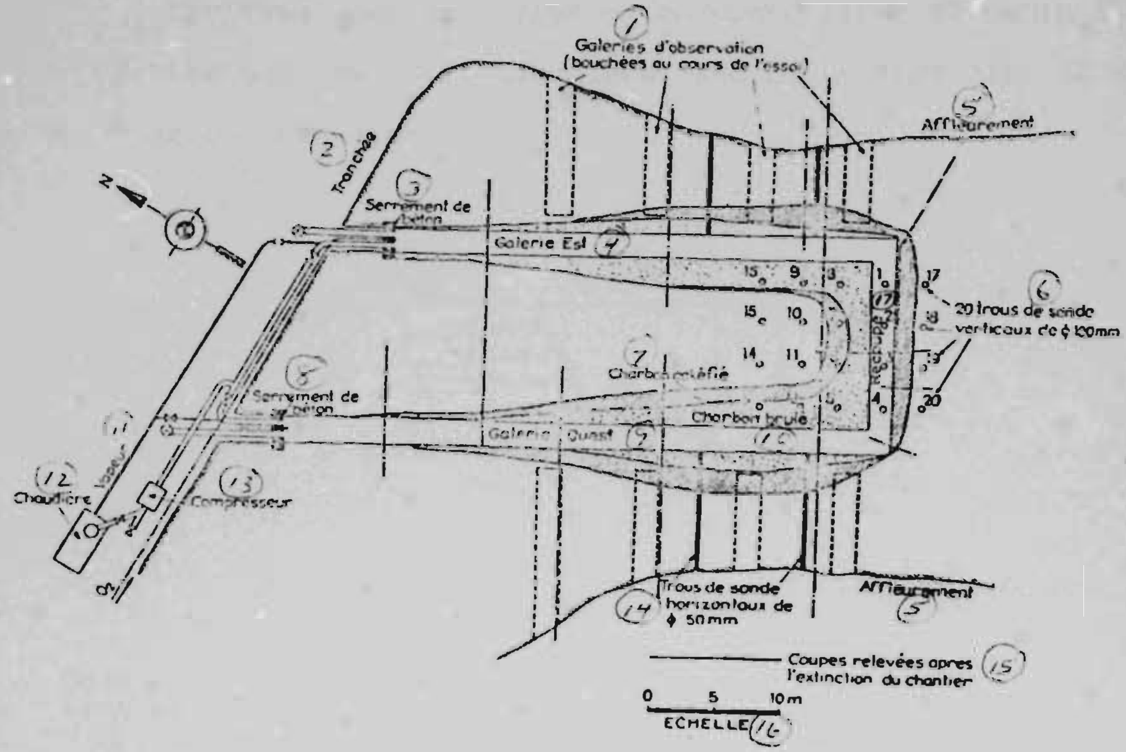


Fig. 23

KEY

1. Observation passages (plugged up during the test)
2. Trench
3. Concrete dam
4. East passage
5. Outcrop
6. 20 vertical boreholes, 100 mm diameter
7. Coked coal
8. Concrete dam
9. West passage
10. Burnt coal
11. Steam
12. Boiler
13. Compressor
14. Horizontal boreholes, 50 mm diameter
15. Cuts surveyed after extinction of the site
16. Scale
17. Crosscut

The dams were traversed by concrete pipes 60 cm in diameter for the passage of air or gases, and by an iron pipe 50 mm in diameter for the steam.

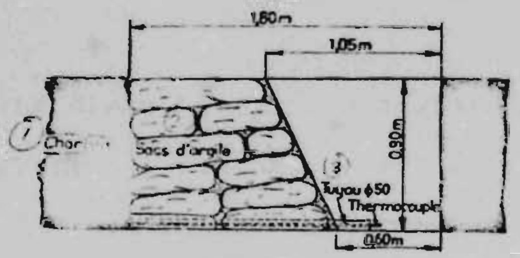


Fig. 24

KEY

1. Coal
2. Bags of clay
3. Pipe, 50 mm diameter

Eight observation passages were excavated from the outcrop of the seam up to nearly 1 m from the outside walls of the site. 5 x 50 cm slits cut through the remaining one meter mass and widening toward the site allowed visual observation through a pane of pyrex.

Explosive discs 60 cm in diameter, placed in the same manner in both observation passages, served as safety valves for the site. Boreholes for gas tapping and temperature measurement were provided at each observation port.

During the tests, these observation passages were not tight enough and were quickly put out of operation. They were replaced by horizontal boreholes drilled into the seam from the outcrops up to the site.

100 mm vertical holes, numbered from 1 to 20, enabled gas to be tapped and temperatures and pressures measured throughout the entire zone affected by the fire. They were drilled from the surface into the panel and into the crosscut at uniform intervals of 3 m.

The symmetrical arrangement of the pipes and valves enabled the site to be supplied with air, oxygen and steam in either direction.

The compressor (volume) powered by a 75 horsepower electric motor was capable of supplying 12,500 m^3/h at a pressure of 1400 mm of water. The suction end of the compressor could be connected to the stack of the boiler in order to send into the mine an inert plug of steam to eliminate any risk of explosion when the gas stream was reversed.

The oxygen was supplied through a 75-mm pipe from a tank of liquid oxygen and a gasifier located some distance away and capable of delivering a maximum flow rate of 350 m^3/h.

The flow rates of air, oxygen and steam were measured by diaphrams.

The gas escaped through two chimneys placed at the outlet of the two passages. Atomizers placed at the base of the chimneys and in the passages cooled the gas and prevented combustion of the coal walls.

1. *Ignition and performance of the test*

A channel 45 cm deep and 20 cm high was cut (by hand) in the crosscut at the base of the south face of the panel (fire face).

The excavated material had been piled up along the opposite wall of the passage and the latter had been filled over 10 m, up to 20 cm from the roof, with alternate layers of coal and wood soaked in fuel oil. Thermite bombs were placed just below borehole no. 1 (southeast corner of the panel).

On 21 January 1947 at 14 hours, four thermite bombs were ignited at the surface and thrown through hole no. 1, while an air flow rate of 4000 m^3/h passed in the east-west direction. A few seconds later, a large amount of thick black smoke emerged through the west chimney.

After a few hours of operation, the site was discovered to be leaky. From 21 to 31 January it was necessary to gradually plug up all the observation passages by means of bags of dust and to gunite them. The inlet and outlet dams also required constant care and repeated reinforcement.

During these operations it was possible to confirm, on 26 January, that the roof and wall of the crosscut were on fire and that a free passage remained at the upper part of the passage above a pile of debris from the roof.

Furthermore, on 1 February the fire was visible a dozen meters from the east entrance of the mine, but invisible from the west side.

Since the combustion did not seem to be very active at the end of the panel, dynamite charges were exploded at the bottom of vertical boreholes no. 8 (28 January) and 5, 6, 7 (30 January) and again no. 5 (3 February) with immediate release of smoke and gas through these respective holes.

On 3 January the fire had encroached upon the external wall of the crosscut over a depth of between 1 and 2 m.

On 5 February, the gas burned at the orifice of holes 5 and 8.

On 14 February, it also burned at holes 6 and 7.

From 18 to 24 February, 12,700 m^3 of this gas were collected through these holes and, after preheating, burned under the boiler with a mean calorific value of 625 cal/m^3.

On 12 March at 8 hours in the morning, the site was halted after a last test of intense blowing.

Throughout this period there was a continuous struggle against the numerous leaks and beginnings of fire appearing in the observation passages, at the outlet dams, along the outcrop of the seam and over the entire surface of the dead ground.

2. *Results*

From 21 January to 12 March the installation had operated for 1039 hours with alternate blowing in both directions and only 160 hours of shutdown (13% of the time).

These tests are summarized in the Table below.

1 Comburant	2 Nombre de périodes	3 Sens est-ouest	4 Sens ouest-est	5 Durée moyenne
Air (du 21-1 au 12-3) 6	63	33	31	13 h 20'
Air suroxygéné (du 29-1 au 2-2) .. 7	20	9	11	3 h
Air, oxygène, vapeur (1 et 2-2) ... 8	2	1	1	8 h 15'-2 h 15'
Oxygène, vapeur (2-2) 9	1	—	—	2 h 50'

KEY

1. Combustion agent
2. Number of periods
3. East-west direction
4. West-east direction
5. Mean duration
6. Air (from 1-21 to 3-12)
7. Superoxygenated air (from 1-29 to 2-2)
8. Air, oxygen, steam (2-1 and 2-2)
9. Oxygen, steam (2-2)

In addition, steam alone was injected into the site on various occasions with the aim of producing water-gas (see the results of four tests further on).

Moreover, recordings were made of the composition of the gases formed under the action of the steam plug used to facilitate inversions, and that of the gas released from the mine under natural draft during shutdown periods.

a) *Air blowing*

From 21 to 29 January, the working conditions were essentially unstable. The counter pressure from the site increased continuously and several shutdowns were made necessary by mechanical trouble or work to assure the leaktightness of the site.

The gas produced during that period had a composition varying between the limits below:

	CO_2	O_2	CO	H_2	CH_4	C_nH_m	N_2	g.c.v.	n.c.v.
min.	14.0	0.2	0.4	1.8	0.4	0	74.4	180	160
max.	16.6	2.1	2.9	5.0	2.4	0.6	79.8	450	400
mean	14.8	1.5	1.9	3.9	1.5	0.1	76.5	328	294

The air flow rate varied from 1330 to 4000 m^3/h (about 2000 m^3/h on the average) and the gas flow rate was approximately equal to it (0.98 to 1.06 m^3 of gas per m^3 of blown air).

The highest air flow rates gave the best gas, but half the calorific value of the gas produced is due to the methane and other distillation products which are relatively more abundant during this first phase than subsequently, since the coal is still fresh.

The oxygen tests discussed above took place from 29 January to 2 February.

After 2 February, the site became much more stable.

During this last period, a gas varying between the following limits was obtained:

	CO_2	O_2	CO	H_2	CH_4	C_nH_m	N_2	g.c.v	n.c.v	Débit air
min.	12.6	0.2	0.7	2.2	0.6	0	67.4	237	218	1.300 m^3/h
max.	17.8	2.5	5.0	8.7	3.5	0.6	78.8	620	560	2.800 »
le 3 mars	16.1	0.4	3.4	7.8	2.0	0.4	69.9	570	507	2.150 »
3-4 mars	15.0	0.5	3.9	8.7	1.6	0.3	67.1	620	560	1.880 »
moyenne	15.2	1.0	3.1	5.5	1.5	0.2	73.5	424	380	2.000 »

Débit de gaz = 0,98 . 1,17 m^3 par m^3 d'air.

KEY

1. Air flow rate
2. 3 March
3. 3-4 March
4. Mean
5. Gas flow rate = 0.98-1.17 m^3 per m^3 of air

The compositions cited for 3 and 4 March represent the best average results obtained for one entire period of blowing. However, some isolated analyses gave a 680 cal/m^3 gas, and taps made at some points of the gas circuit exhibited up to 1140 cal/m^3.

If it is attempted to set up a balance of materials and to interpret the gas formation quantitatively according to the equation:

$$c \text{ (kg of coal)} + a \text{ (m}^3\text{ of air)} =$$
$$k \text{ (kg of coke)} + h \text{ (m}^3\text{ of steam)} +$$
$$1 \text{ m}^3 \text{ of gas}$$

we obtain the following results for the two periods under consideration (23-31 January, 3 February - 12 March):

Period from 23 to 31 January

Elements of the balance per Nm^3 of gas	Inputs	Fuel $c = 0.261$ kg Combustion agent $a = 0.962$ Nm^3 H_2O reduced $h =$ Nm^3	Distillation 472 cal Air-gas 52 cal Water-gas cal Combustion -230 cal n.c.v. of gas 294 cal 13.9 %	Sensible heat of gas 220 cal 10.4 %	Total heat extracted 514 cal 24.3 %
	Outputs	H_2O formed $-h = 0.090$ Nm^3 Unburnt residues $k = 0.121$ kg	Unburnt residues 8100 x 0.121 = 980 cal 46.5 %	Losses to ground 616 cal 29.2 %	Heat remaning in the mine 1596 cal 75.7 %
Yields		$\eta_c = 45$ % $\eta_{th} = 26$ %	Total potential heat 1274 cal 60.4 %	Total sensible heat 836 cal 39.6 %	n.c.v. of fuel 0.261 x 8084 = 2110 cal 100 %

Period from 3 February to 12 March

Elements of the balance per Nm^3 of gas					
	Inputs	Fuel $c = 0.240$ kg Combustion agent $a = 0.928$ Nm^3 H_2O reduced $h =$ Nm^3	Distillation 445 cal Air-gas 87 cal Water-gas cal Combustion -152 cal n.c.v. of gas 380 cal 19.6 %	Sensible heat of gas 220 cal 11.4 %	Total heat extracted 600 cal 31 %
	Outputs	H_2O formed $-h = 0.059$ Nm^3 Unburnt residues $k = 0.095$ kg	Unburnt residues 8100 x 0.095 = 770 cal 39.8 %	Losses to ground 565 cal 29.2 %	Heat remaning in the mine 1335 cal 69 %
	Yields	$\eta_c = 53$ % $\eta_{th} = 32.6$ %	Total potential heat 1150 cal 59.4 %	Total sensible heat 785 cal 40.6 %	n.c.v. of fuel 0.240 x 8084 = 1935 cal 100 %

According to these calculations, 425 tons of clean coal (corresponding to 517 tons of moist ashy coal) would have burned in 50 days for a total production of 1,800,000 m^3 of gas. 117 tons of coke (assumed to be pure carbon), corresponding to 210 tons of coal, remained in the mine. About 50% of the available carbon was gasified.

This calculation agrees approximately with the findings made when the site was opened. Those findings indicated a total of [illegible]00 tons of coal affected by the experiment and 164 tons of coked coal.

This coke production does not mean that the combustion was incomplete. The coke remaining in the mine would have burned in turn if the experiment had been carried on. The formation and combustion of coke would be approximately in equilibrium after a sufficiently long steady state.

However, it should be noted that the combustible portion of the gas corresponds practically to the volatile matter in the consumed coal. The gasification reactions are completely masked by secondary combustions. The gas from the second period can actually be decomposed as follows:

	CO_2	O_2	CO	H_2	CH_4	N_2	n.c.v.
Distillation ...	1.1	—	0.2	11.4	1.7	0.3	443
Air-gas ...	14.1	1.0	2.9	—	—	62.1	87
Combustion ..	—	—	—	— 5.9	—	11.1	— 152
Total	15.2	1.0	3.1	5.5	1.7	73.5	380

b) *Blowing with superoxygenated air*

From 29 January to 1 February, twenty blowing periods were performed with oxygenated air for a total of 59.5 hours of activity. The mean duration of the periods (3 h) was greatly reduced relative to that for the air tests. The cycle time of the inversions was accelerated. In fact, the fire had a tendency to raise the stream of combustion agent and menaced the entrance dams.

The mean composition of the product gas is given below:

Combustion agent			gas								
O_2	N_2	flow rate	CO_2	O_2	CO	H_2	CH_4	C_nH_m	N_2	g.c.v	n.c.v
34.3	65.7	1,700 m^3/h	32.3	0.9	2.1	0.1	2.1	0.1	30.4	470	416

But these results are hardly indicative, since the mine had not yet reached a stable condition and the supply of oxygen available was too low to extend the test sufficiently. Furthermore, it proved difficult to regulate the oxygen flow rate.

c) *Blowing with superoxygenated air and steam*

This test lasted a total of 10.5 hours on 1 and 2 February. The results obtained were much better than those from the preceding tests and there was no need to worry about fire near the dams.

Combustion agent				gas								
O_2	N_2	H_2O	flow rate	CO_2	O_2	CO	H_2	CH_4	C_nH_m	N_2	g.c.v.	n.c.v.
32.3	43.4	23.3	1.700 m^3/h	40.7	1.4	4.1	13.2	4.4	0.4	35.8	1.023	909

d) *Oxygen-steam blowing*

A test was performed for 2 hours 50 minutes with an oxygen-steam mixture. The results are given below, but undoubtedly would have been better if the available oxygen supply had enabled the test to be prolonged or the flow rate of the combustion agent increased:

Combustion agent			gas								
O_2	H_2O	flow rate	CO_2	O_2	CO	H_2	CH_4	C_nH_m	N_2	g.c.v.	n.c.v.
65	35	610 m^3/h	59.6	1.9	8.6	18.8	5.7	0.4	7.0	1.260	1.125

The calorific value, which increased to 950 cal/m^3 from the beginning of the test, exceeded 1400 cal/m^3 at the instant when it had to be interrupted.

It is difficult to evaluate the amount of gas produced and the amount of steam that reacted, since the nitrogen balance makes it impossible to compare the incoming and outgoing products.

However, judging from the methane content it can be estimated that a good portion of the hydrogen (about 12% of the total gas) originates from the distillation of the coal. The calorific value obtained would then come in equal parts from distillation and gasification. About a quarter of the water vapor injected would have been decomposed.

e) *Steam blowing*

On many occasions, the compressor was halted and steam alone was directed into the mine in order to produce water-gas. Furthermore, at each inversion the mine was purged with steam before reversing the air stream (the use of combustion gas as a plug proved superfluous).

The gas obtained during these purges was identical to that from the actual steam tests.

The latter lasted from 40 minutes to 2 hours. They usually yielded gas with more than 1800 cal but containing large quantities of CO_2 and little CO, which indicates low temperatures.

Moreover, the presence of free oxygen in significant quantities in the gas indicates air returns at the time when specimens were tapped, since the site was no longer pressurized. The analyses were corrected by subtracting this oxygen and the corresponding nitrogen from the measured compositions:

Steam flow rate kg/h	Steam flow rate Nm³/h	Duration and date		CO_2	O_2	CO	H_2	CH_4	C_nH_m	N_2	g.c.v.	n.c.v.
900	1 100	135 min	rough	11.4	6.6	7.3	36.6	5.8	0.5	31.8	1.935	1.670
		23 Jan.	corr.	17.0	0	10.6	51.4	9.0	0.8	8.2	2.910	2.550
195	245	90 min	rough	27.7	1.1	16.8	27.1	5.2	0.2	25.9	1.660	1.490
		4 March	corr.	29.5	0	17.8	28.7	3.4	0.2	20.6	1.750	1.580
835	1 040	60 min	rough	31.1	1.6	7.4	23.2	6.6	0.3	30.4	1.530	1.360
		5 March	corr.	33.8	0	8.1	25.2	0.5	0.3	26.1	1.660	1.470
630	785	40 min	rough	32.0	0.8	11.6	13.6	5.4	0.2	36.4	1.300	1.180
		10 March	corr.	33.3	0	12.1	14.2	5.6	0.2	34.6	1.350	1.225
Purges		mean	rough	18.6	4.0	10.6	38.9	5.0	0.5	22.4	2.030	1.790
			corr.	23.2	0	13.3	48.6	6.3	0.6	8.0	2.540	2.240

f) *Without blowing*

During interruptions of the blowing made necessary by repairs on the site, the gas released under natural draft exhibited compositions varying between:

	CO_2	O_2	CO	H_2	CH_4	C_nH_m	N_2	g.c.v.	n.c.v.
min.	13.0	2.6	2.8	22.2	3.5	0	9.7	1.460	1.290
max.	28.0	7.2	14.7	41.6	10.9	0.8	59.0	2.430	2.160
mean	21.5	3.9	7.4	30.4	7.8	0.3	28.7	1.920	1.700
Corr.	20.7	0	9.2	37.7	9.7	0.4	16.3	2.390	2.110

Thus, this gas is very similar to that obtained with steam blowing. Its flow rate was not measured.

3. *Special observations*

a) Most of the tests described above using oxygen or steam as combustion agent or without blowing were of too short duration and performed under too variable conditions for it to be possible to set up a balance.

However, one is struck immediately by the high content of methane and CO_2 in the gases obtained during these tests.

The high methane content is readily explained if it is assumed that the intensity of the distillation hardly depends at all on the flow rate of the gas stream. When oxygen and steam are used, the flow rate is less than when air is used and the probable condensation of a good portion of the steam further decreases the volume of gas in which the distillation products are diluted.

The very high CH_4 contents (up to 10%) in the gas obtained during interruptions of blowing confirm this conjecture.

In contrast, the CO_2 contents cannot be explained solely by the higher concentration of the gas resulting from the absence or decreased percentage of nitrogen.

In fact, it is found that neither the amount of oxygen introduced by the combustion agent or by the returns of air in the gas (this quantity is measured by the percentage of nitrogen) nor the formation of water-gas (whose hydrogen should be found in the gas) can justify quantitatively the high percentages of CO_2 indicated by the analyses.

In other words, there is an excess of oxygen and a deficiency of hydrogen in the balance of materials.

For example, in the average of the tests with superoxygenated air we find (the ratio A = oxygen/nitrogen of the combustion agent being known):

56.4 % N_2, corresponding to $56.4A = 56.4\ \frac{34.3}{65.7} = 29.4$ % O_2 or CO_2

6.1 % H_2, corresponding (assuming that all the hydrogen comes from the water-gas) to = 3.05 % O_2 or CO_2

32.45 % total

and also:

32.3 % CO_2	= 32.3 %
0.9 % O_2	= 0.9 %
2.1 % CO corresponding to $\frac{2.1}{2}$	= 1.05 % O_2
	34.25 %

The differences are still greater for:

	CO_2+O_2 +1/2 CO	AN_2+1/2 H_2
The air-oxygen-steam test	44.15	33.3
Most of the steam tests	38.2	19.8
The gas collected by natural draft	30.95	25.2

It seems difficult to find an explanation for this phenomenon. Either the cited analyses are not representative of all the gas withdrawn from the mine or else an obscure phenomenon is taking place.

We might imagine that the excess CO_2 comes from the distillation of limestone rocks in the walls. But they are composed solely of more or less sandy schists. Thus, we must assume that part of the CO_2 produced during the preceding periods remains occluded in the coal or stagnates in some corners of the site and is released abruptly as a result of the change in conditions, or else an appreciable fraction of the hydrogen, which is more mobile than the other gases, diffuses into the ground when that gas appears in significant concentrations in the gas.

However, the first steam test on 23 January lasted for 2 hours 15 minutes and yielded an acceptable balance. With the usual notations we have per m^3 of gas:

Elements of the balance per Nm^3 of gas	Inputs	**Fuel** c= 0.648 kg Combustion agent a=0.093 Nm^3 H_2O reduced h=0.342 Nm^3	**Distillation** 1403 cal **Air-gas** 28 cal **Water-gas** 1126 cal **Combustion** cal **n.c.v. of gas** 2557 cal 48.8 %	**Sensible heat of gas** 220 cal 4.2 %	**Total heat extracted** 2777 cal 53 %
	Outputs	H_2O formed -h= Nm^3 Unburnt residues k=0.344 kg	**Unburnt residues** 8100 x 0.344 = 2785 cal 53.2 %	**Losses to ground** -332 cal -6.2 %	**Heat remaining in the mine** 2453 cal 47%
Yields		η_c = 36.8 % η_{th} = 104 %	**Total potential heat** 5342 cal 102 %	**Total sensible heat** -112 cal -2 %	**n.c.v. of fuel** 0.648 x 8084 = 5230 cal 100 %

The losses of sensible heat are negative here. In fact, the formation of the water-gas is endothermic and the heat restored by the ground is greater here than the sensible heat of the gas, resulting in a yield greater than 100%.

34.2% hydrogen comes from the water-gas.

However, we note that the figures calculated below have a different value than that of a purely qualitative indication.

b) *Variation of the gas along the circuit*

The composition of the gas varies continuously along the gasification circuit. During numerous periods of blowing, samples were taken at various points of the site through 2" boreholes (50 mm) drilled from the outcrop of the seam and replacing those of the observation passages which had been plugged up. The averages of the compositions recorded are given below as a function of the distance from the entrance.

They show that the almost complete absorption of the oxygen occurs within the first third of the path. It is nearly at that point that the maximum CO content was recorded. In contrast to what was thought and to what was predicted from the laboratory tests, the CO_2 content had no tendency to decrease at the expense of the CO during the last part of the path (reduction zone). But this is the opposite of what occurs, probably due to secondary combustion phenomena.

Length covered		Stream direction	Composition of gas		
m	%		CO_2	O_2	CO
26	24	east-west	8.5	10.1	0.4
36	33	»	10.2	0.2	13.6
64	59	»	12.9	0.1	8.9
72	67	»	14.8	0.1	5.6
36	33	west-east	4.3	13.0	0.3
44	41	»	10.3	6.5	14.1
72	67	»	13.1	0.1	9.6
82	76	»	12.1	0.4	10.3
Total length of circuit: 108 m					

The best gas composition of all the tests performed with air was recorded at a point located in the first third of the circuit:

CO_2	O_2	CO	H_2	CH_4	C_nH_m	N_2	g.c.v	n.c.v.
6.2	0.2	18.8	12.2	2.0	0.4	60.2	1.207	1.121

c) *Resistance of the face*

The pressures required to pass various flow rates into the site were successively:

	Flow rate m^3/h	Pressure mm H_2O	Eq. or. dm^2
21 January	4.000	200	2.8
25 January	2.000	200	1.4
7 February	2.000	1.000	0.65
12 March	3.700	2.000	0.825

After two weeks, the resistance was stabilized. It passed approximately 2 m^3/h per mm of H_2O of pressure difference. This increase of the resistance of the face is due to the plugging of the passages by fused or softened materials from the roof and ash from the coal, leaving only narrow passages at its surface for the gas.

Pressure measurements at intermediary points enabled incipient obstructions to be localized quickly during the test.

d) *Pyrometry*

The thermocouples placed in the coal at the bottom of the vertical boreholes mostly indicated a plateau at 100°C shortly after they began to be affected. As the fire approached each borehole, each of them in turn indicated a rapid rise and stopped giving indications above a temperature of 550° to 950° which was reached in a few days.

The table below indicates, for each of the thermocouples placed in the coal, the day (counting from ignition) when it was affected by the fire. The numbers are those of the vertical boreholes (Fig. 23).

No. 1 to 4 Crosscut above 1000° beginning at ignition			
no. 8 11 1/2 days	no. 7 23 days	no. 6 23 days	no. 5 9 days
no. 9 18 days	no. 10 at 100° after 35 days	no. 11 at 100° after 48 days	no. 12 18 days
no. 16 24 days	no. 15 less than 100°	no. 14 less than 100°	no. 13 29 days

The gas was cooled by spraying water at the outlet from the site in order to protect the conduits and dams. Their temperature was maintained at around 375°, but at 6 meters from the dams, beyond the sprayers, it was still 600° (which makes it possible to evaluate the heat carried by the gas as 220 cal/m^3). Measurements made by the optical pyrometer indicated 860° for the ignited materials inside the site, but this figure is probably too low due to the smoke filling the passages and falsifying the readings of the apparatus.

Judging from the low CO content of the gases, the temperature of the site would not have been very high. This may be due to the small thickness of the dead ground, frequent reversals and injections of steam or too low flow rates of the combustion agent.

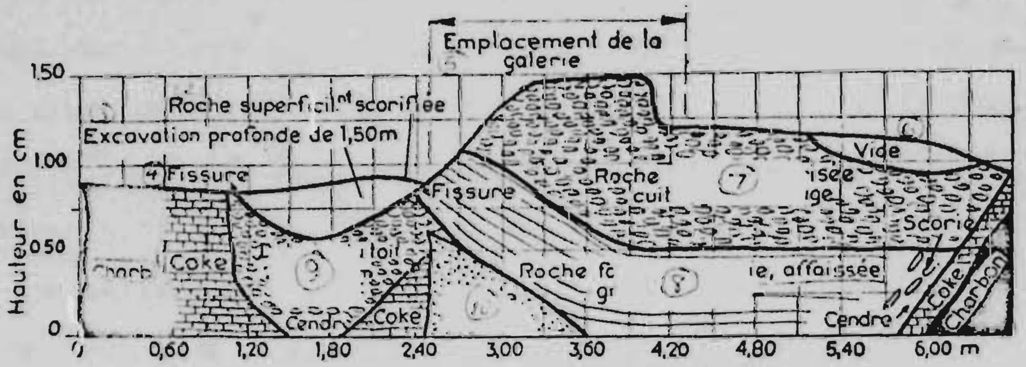
Inspection of the site after extinction and the condition of the ground and ash enable us to assert, however, that the temperature reached and probably exceeded 1300°C.

4. *Cooling and examination of the site*

After 50 days of operation, the site was shut down on 12 March and steam was injected into it for 5 days. The temperatures recorded during this period at the bottom of the boreholes fell from 550° to about 400°. Starting on 17 March, extinction was achieved by flooding the site. Part of the water came out through the fissures at a temperature 85°. On 23 March, the temperatures at the boreholes were 100° to 300°C. There was 0.30 m to 0.60 m of water in the passages.

After extinction of the site, it was examined carefully by means of small cuts excavated across the panel from the outcrops.

The following discoveries were made (Fig. 25):



Charbon = coal

Cendre = ash

Fig. 25

KEY

1. Height in cm
2. Volcanic scorified rock
3. Excavation 1.50 m deep
4. Fissure
5. Location of the passage
6. Void
7. Chipped and broken rock, baked, reddish yellow
8. Fused, softened, subsided rock, dark gray
9. Debris from the roof 20 to 40 m/m ash
10. Baked clay

a) The passages delimiting the panel were nearly completely clogged by rocks from the roof which were softened over a thickness of 0.50 to 1 m and had swelled and folded, coming to rest on the floor. The upper banks were fractured and baked to about 3 m from the seam.

b) The coal in the panel was completely burnt over a depth of 2 m at the end of the panel, reducing gradually to 0 toward the entrances. All the space occupied previously by the burnt coal was filled up by the swelling and subsiding of the roof. There was a thin bed of volcanic slag on the floor under the materials from the roof.

c) A coked zone from a few cm to 1.50 m thick bordered the burnt zone. The surface of contact between the coke and the fused rocks or volcanic slaq was not vertical. In every direction the upper part of the seam had burned more deeply than the lower part, giving the surface of separation the shape of either a ledge of coke 0.50 to 1 m wide or a plane inclined at 45° terminating the coke mass in wedge form under the filling. Between the coke and rock there was also a thin layer of ash and volcanic slag.

This surface of separation and the underlying coke exhibited numurous fissures (on the order of 1 centimeter) through which the air stream undoubtedly passed in direct contact with the incandescent coke.

The coke mass was not compact. The coke in the upper part of the vein was in massive blocks, while in the lower part it formed elongated pieces placed obliquely with respect to the vertical.

d) The coked zone was followed by a zone of partially carbonized coal passing gradually to the intact coal as indicated by the analyses of specimens taken at increasing distances from the fire face:

Distances, m			0.15 m	0.90 m	1.70 m	2.45 m	4.50 m
1 Analyse immédiate	3 cendres (sur sec)		31.6	21.0	17.4	13.7	16.4
	4 Charbon net (sec et sans cendres)	5 MV	3.5	3.9	21.5	39.0	39.5
		6 carbone fixe	96.5	96.1	78.5	61.0	60.5
2 Analyse élémentaire		C	96.4	95.0	85.7	83.9	83.6
		H	0.4	0.6	4.2	5.7	5.8
		O	0.7	1.2	6.8	7.1	7.5
		N	0.9	1.7	2.0	1.7	1.8
		S	1.6	1.5	1.3	1.6	1.3
7 Pouvoir calorifique sup. (charbon net)			7.580	7.850	8.170	8.340	8.325
8 Poids spécifique (réel)			2.060	1.999	1.517	1.394	1.419

KEY

1. Proximate analysis
2. Ultimate analysis
3. Ash (relative to dry)
4. Clean coal (dry and without ash)
5. Volatile matter
6. Fixed carbon
7. Gross calorific value (clean coal)
8. Specific weight (actual)

On the side opposite the panel itself there is found, completely baked, the lining of clay designed to protect the walls of the passages. Its action was not effective: behind this lining a large amount of coal was burnt. A zone of coke was formed, reaching a thickness of 2 m at places.

In the less hot zones of the site (up to 25 m from the entrances), the roof did not creep, but baked and was split, partially filling the passages with debris.

Determinations of apparent density were made (by immersion in a finely divided powder) on specimens of roof more or less affected by the heat.

	Apparent density	Ratio
Schist from the roof, unaltered	2.54	1.00
Completely baked, but without visible swelling	2.07	0.82
Exhibiting swelling and beginning of fusion	1.35	0.54
Same, with bending and deformation of the stratification beds	1.49	0.50
Completely fused, scorified and swollen	1.58	0.62

These figures show how the roof materials were able to block up the passages, the space occupied by the burnt coal and the voids created by the subsidence of the roof.

From the records of the burnt and coked zones, the amount of coal burnt completely was evaluated as 221 tons (to which must be added the 15 tons piled up in the "fire face" for ignition) and the amount of coked coal as 164 tons, making a total of 400 tons of coal affected by the combustion.

It is interesting to note that no islands of unburnt coal or coke were found behind the fire face.

5. *Conclusions*

This test demonstrated:

1) The progressive closing of the gas passage with establishment of a steady state.

2) Plugging by fusion of the roof in the burnt zone.

3) Asymmetric attack of the horizontal fold (the upper part burns first).

4) Possibility of complete combustion of the vein, even the horizontal fold (to be able to draw definitive conclusions concerning this point, it would be necessary to have carried on the test until complete depletion of the panel) and formation of a coked zone along the fire face.

5) Importance of leaktightness of the gasification site and, consequently, necessity of working at a certain depth far enough from outcrops and other workings.

6) Importance of large flow rates and high temperatures in the reaction zone in order to obtain a good gas.

7) Possible deterioration of the gas in the downstream portion of its path.

8) Possibility of creating inversions by using a steam plug.

9) Formation of a relatively rich gas when the blowing is interrupted, with or without injection of steam.

It was not possible to prolong the use of oxygen sufficiently to be able to study all its effects.

B. Second test

A second test on the Pratt layer was organized, making use of the information from the preceding test.

This new site was also laid out in a portion of the vein almost completely isolated from the rest of the deposit by erosion and covering 40 ha [1 ha = 1 hectare = 10^4 m^2]. It is thus larger than the one used in the first test. The site is at least 150 m from outcrops in all directions and the dead ground is nearly 50 m thick.

The Pratt layer here has a thickness of 1.05 to 1.17 m and a dip of 4% (2°20') toward the southeast.

The shape of the site was simplified to a large degree: It consists of a passage 470 meters long excavated along the dip of the seam starting from the outcrop and closed by a dam 43 m from the entrance. This dam is fit in solidly and penetrates 0.60 m into the floor of the seam, 1.80 m into the roof and 7.50 m into the coal.

The cross-section of the passage is 3 m x 1.05 m (height of the seam). In reality, in order to guarantee aeration during the excavation, the first 380 meters comprise two identical parallel passages separated by 3 m of coal and connected by crosscuts every 90 meters. Only the last part has a simple section (90 m) (Fig. 26).

Five boreholes drilled from the surface connect the passage to the open air every 90 m to the right of the crosscuts. The dead ground was solidified in advance by injecting cement at the location of each hole.

Two of them (no. I and V) were made without tubing. They were 45 cm in diameter, equipped with a catch head and sealed with cement.

The others (no. II, III, IV) were drilled with a diameter of 70 cm and provided with a steel tubing 50 cm in diameter, and

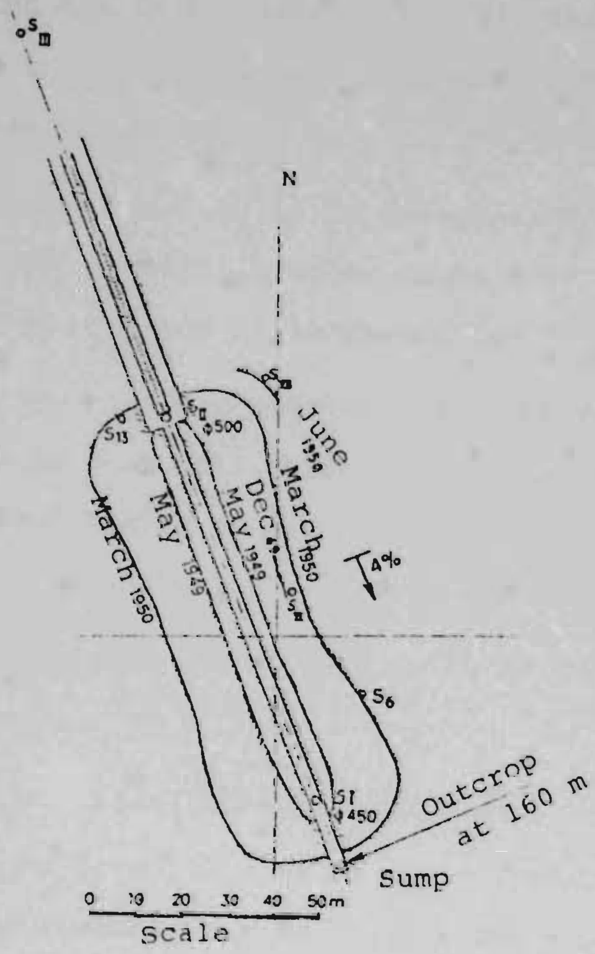


Fig. 26

sealed from the ground by means of refractory cement.

The upper portion of the boreholes was equipped with a water-jacket cooling system and sprayers. The lower end of the tubing rested on masonry at the bottom.

These five main boreholes were able to be used at will as gas entrances or exits, making it possible to reverse the direction of the stream or to modify the length of the site.

Beyond the last borehole (no. I), the passage was prolonged further for 15 m to a sump designed to collect the very abundant seepage water sent to the surface by a centrifugal pump.

Other boreholes 15 cm in diameter were dug at variable distances from the passage in order to be able to trace the advance of the fire by means of thermocouples.

The blowing means consist essentially of a piston compressor (double action) activated by a 800 hp synchronous motor and supplying 12,000 m^3/h of air at 2 kg/cm^2. This compressor is connected to the boreholes by a conduit 500 mm in diameter.

Auxiliary compressors and a 125 hp boiler are also available.

1. *Operations* (see Table III)

Ignition took place on 18 March 1949 by means of a thermite bomb thrown through hole no. I (the one at the end of the passage) onto a pile of 15 tons of coal and wood soaked with fuel oil. The walls of the passage had been undercut in advance over a depth of 0.40 m.

From that date until September 1950, four periods of operation were distinguishable.

a) *Period from 18 March to 22 June 1949*

During the first ten days following ignition, the air stream passed from borehole I to borehole II with a flow rate increasing steadily from 2300 to 10,000 m^3/h. Under these conditions it was impossible to achieve complete combustion. The percentage of CO_2, which reached 13.6% after the fourth day, fell back during the following days to 4% (eleventh day). The combustible components remained in traces.

Furthermore, the fire had been driven toward the base of borehole II whose temperature rose continually.

On 28 March it was decided to reverse the air stream and that maneuver was repeated subsequently at intervals decreasing steadily from 100 hours to 7 hours.

At the end of May, the temperature of the site exceeded 1250°C and the gases emerged at 370° (the outlet temperature was higher the slower the rate of reversal and it was possible to regulate it by modifying the cycle time). At that time, 10 tons of coal were being burned per day and the passage had widened by 2.5 m on each side.

The pressure loss of the site for a flow rate of 12,000 m^3/h reached 3000 mm of water (for an initial value of 30 mm). This pressure loss was distributed uniformly over the entire length of the site. The oxygen always present in the emerging gases demonstrated, however, that the combustion was incomplete. The air stream, following the shortest path between the boreholes, probably remained on the axis of the passage without touching its walls and the natural subsidence of the roof, instead of forcing the gas to enter into contact with the fuel, seemed instead to provide it with parasitic passages through the cracks in the roof.

b) *Period from 22 June to 22 December 1949*

It was then decided to carry out an artificial packing by means of sand "fluidized" in an air stream and injected into the site through supplementary boreholes drilled along the axis of the passage.

This measure was applied beginning on 22 June 1949 and had a very marked effect on the piezometry of the site. From June to September, 156 tons of fluidized sand were injected in this manner, while the monthly consumption increased from 312 tons of clean coal in May to 583 tons in October.

1949 Month	Pressure loss in mm H_2O Flow rate of 12,000 m^3/h	Burnt coal in tons/day (clean coal)
May	2900	10.1
June	3500	11.2
July	4850	14.6
August	5750	14.3
September	6600	16.2
October	9200	18.8
December	11,300	22.7

Beginning in October, the intervals between reversals were increased gradually. From 1 to 22 December, cycles of 100 to 140 hours were used.

These long periods of blowing caused a concentration of the reaction zone at the base of the outlet borehole, leading to an increased consumption of coal at that point and an elevation of the emerging gas temperature. The latter reached 900-1300° at the orifice of the boreholes and caught fire there despite its low calorific value (less than 360 cal/m^3) due to the sensible heat carried along.

Following are the phenomena observed during one cycle of 60 hours which was quite representative of normal operation.

Starting from the beginning of the blowing period (11,000 m^3/h), the percentage of O_2 in the emerging gas decreased progressively while the percentage of CO_2 increased as well as the temperature of the gas.

During the first 42 hours of the cycle, the gas emerged at a mean temperature of 250°C with a mean composition:

	CO_2	O_2	CO	H_2	CH_4	C_nH_m	N_2	g.c.v.	n.c.v.
Crude	8.8	10.0	2.0	2.6	0.8	0.3	73.5	273	245
Calculated without air	16.8	0	3.8	5.0	1.5	0.6	72.3	520	468

This crude gas thus acted as a mixture of 52.6% 520 cal/m^3 gas and 47.4% air. It contained about 100 g of H_2O per m^3 (standard conditions) of dry gas.

When the temperature of the emerging gas reached 430°C after 42 hours of operation, the combustible components burn in the air of the mixture and there is obtained only a flue gas containing 16 to 18% CO_2, 1 to 2.6% O_2 and nitrogen, with 240 g H_2O per Nm^3 dry. The outlet temperature settles at around 900-1100°.

This proves that the gas is produced at the bottom and is reburned by the bypassing of air into the circuit. Through boreholes located in the immediate vicinity of the fire face it was possible to take specimens having the following compositions:

	CO_2	O_2	CO	H_2	CH_4	C_nH_m	N_2	g.c.v.	n.c.v.
Hole no. 6	10.4	0.7	10.0	10.9	2.3	0.3	65.4	912	830
Hole no. 13	8.5	0	15.3	11.4	2.0	0.2	62.4	1.032	966

c) *Period from 22 December 1949 to 5 June 1950*

During the month of December, a new borehole 250 mm in diameter was dug between no. I and no. II, 12 m east of the axis of the original passage. This borehole no. VI was used in combination with no. III for four cycles of 8 hours. It was hoped that a better contact between combustion agent and coal would be achieved in this way.

Blowing of 12,000 m^3/h during the first and third periods from VI to III, i.e. from a hot zone to a cold zone, resulted in complete combustion but hardly any combustible components. Blowing from III toward VI during the second period resulted in an 850 cal/m^3 gas without any residual oxygen. Unfortunately, during the fourth cycle the bypassing occurred again. The gas produced burned at the bottom of the hole so that the rock was scorified under the influence of high temperatures and the bottom 30 meters of the hole were blocked up completely.

Then there was a return to circuit I-II for six months, with alternations of 8 hours maintained very regularly, so as to concentrate the reaction zone at a distance midway between the two boreholes. This objective was achieved approximately as could be ascertained through the inspection boreholes.

d) *June-July 1950*

A new borehole, no. VII, was then drilled 23 m to the east of no. II at a small distance from the fire face. It was reached by the latter at the beginning of June and was put into service on 5 June in combination with no. III. A sector of fresh coal was thus attacked. To assure better contact between combustion agent and fuel, the portion of the passage connecting boreholes II and III was packed by injecting fluidized sand.

The gas produced in June had a mean g.c.v. of 415 cal/m^3 and contained little or no O_2. Unfortunately, beginning at the end of June and throughout the month of July, residual oxygen reappeared in increasing amounts in the analyses, while the calorific value fell to about 250 cal/m^3.

Table III provides a summary of the operations during these four periods.

Table III

Period	Total duration	Boreholes utilized	Flow rate	Cycle duration	Burnt Coal			Grand total
	days	no.	m^3/h	hours	t/day	mm/day	kg/m^2-h	tons
from 3-18-49 to 6-30-49	104	I-II	from 2500 to 12,000	from 240 to 7	from 6.7 to 11.2	from 29 to 41	from 1.63 to 2.3	970
from 7-1-49 to 12-22-49	172	I-II	12,000	from 7 to 140	from 14.6 to 22.7	from 52 to 82	from 2.93 to 4.62	3793
December '49	32 hours	III-VI	12,000	8				
from 12-22-49 to 6-5-50	165	I-II	12,000	8	from 15.7 to 10.9	from 57 to 40	from 3.2 to 2.25	5895
from 6-5-50 to 7-31-50	56	III-VII	12,000		from 30.1 to 18.8			7243

2. *Provisional results*

a) The great difficulty encountered in this test was to guarantee sufficient contact between combustion agent and fuel and to prevent bypassing of oxygen. The shape adopted for the site thus appears unfavorable and it was attempted to remedy it by means of boreholes drilled outside the access of the passage. The results obtained were favorable during the first few hours of operation, but the problems soon reappeared. Packing with sand had some effect but was not a complete remedy.

b) The natural subsidence of the roof did not have the anticipated effect. The boreholes for injection of sand showed that the rocks of the passage's roof not directly exposed to the fire had collapsed without fusing, leaving numerous fissures between the debris to short-circuit the fire face. The rock located immediately above the consumed fuel, in contrast, was heated sufficiently to flow and plug the space liberated by the combustion of the coal.

c) The air stream drove the fire zone downstream. That zone was thus concentrated at the bottom of the outlet borehole where the temperature rose steadily. The collected gas is therefore hotter the longer the periods of operation in one direction or the other. The bypassed air undoubtedly has something to do with this rapid displacement of the fire. By burning the gases formed along the walls, it spreads the combustion downstream.

It often happens that this combustion occurs just at the bottom or in the outlet borehole itself. Very high temperatures, on the order of 1100°C, are then obtained. The fact that it is easier to produce very hot gases than gases having a satisfactory calorific value gave rise to the idea of using these gases to heat a boiler or, by pressurizing the site, to supply a gas turbine.

d) Despite the precautions taken to guarantee leak-tightness of the site, leaks were discovered at the outcrops of the seam 150 m from the burning site and at the entrance dam more than 300 m away. Observation boreholes drilled at variable distances from the site made it possible to get an idea of the gas permeability of the seam in situ.

e) The amounts of clean coal gasified varied from 1.6 to 4.6 kg/m^2-h, corresponding to an advance of the fire of 30 to 80 mm per 24 hours at each wall of the passage. These figures are low compared to those cited by the Russians for the tests at Gorlovka (9 kg/m^2-h for the average of the month March 1938 on panel 6).

During the first eighteen months of operation, nearly 7000 tons of coal were burned. Furthermore, there is no indication that this figure represents a maximum and that the zone of action of the system formed by two boreholes could not have been extended still further.

f) The thermal balances show that the calorific value of the gas (a few hundred calories) represents 25 to 33% of that of the fuel consumed and its sensible heat (including that of the water vapor carried along), nearly 20% for rapid cycles and 30 to 45% for prolonged cycles. The losses to the mass represent 30 to 60%.

Using the ordinary method of calculation for the gas collected during the 60-hour cycle described above (second period), we obtain the Table below.

Elements of the balance per Nm^3 of gas					
	Inputs	Fuel c=0.1035 kg Combustion agent a= 0.94 Nm^3 H_2O reduced h= Nm^3	Distillation 227 cal Air-gas 57 cal Water-gas cal Combustion -39 cal n.c.v. of gas 245 cal 29.3 %	Sensible heat of gas 141 cal 16.8 %	Total heat extracted 386 cal 46.1 %
	Outputs	H_2O formed -h= 0.016 Nm^3 Unburnt residues k= 0.022 kg	Unburnt residues 8100 x 0.022 = 177 cal 21.1 %	Losses to ground 274 cal 32.8 %	Heat remaning in the mine 451 cal 53.9 %
	Yields	η_c = 75 % η_{th} = 37 %	Total potential heat 422 cal 50.4 %	Total sensible heat 415 cal 49.6 %	n.c.v. of fuel 0.1035 x 8084 = 837 cal 100 %

The air flow rate of 11,000 m^3/h thus corresponds to a gas flow rate of 11,700 m^3/h and a coal consumption of 1210 kg/h, of which 75% or 900 kg/h are actually gasified, i.e. 22 tons/day. With a wall length of 190 m and a 1.08 m net thickness of the seam, this amounts to the gasification of 4.4 kg/h per m^2 of equivalent surface area or an advance of 78 mm per day.

(to be continued)

END OF
PAPER