ARC Information Series 117

Coal Research Contractors' Conference

Proceedings

October 30-31, 1991
Sheraton Cavalier Hotel
Calgary, Alberta

Alberta ENERGY

ALBERTA OFFICE OF COAL RESEARCH & TECHNOLOGY

ALBERTA RESEARCH COUNCIL

Alberta
ENERGY

Coal Research Contractor's Conference

October 30-31, 1991
Sheraton Cavalier Hotel
Calgary, Alberta

Sponsored by

ALBERTA
OFFICE OF
COAL
RESEARCH &
TECHNOLOGY

Contents

PREDICTING GEOTECHNICAL PARAMETERS WITH LOGS FROM SURFACE MINES[1]

by

N.H. Wade, Ph.D., P.Eng.
Monenco Consultants Ltd., 801 - 6th Avenue S.W., Calgary, Alberta, T2P 3W3

Abstract

The paper incorporates part of the results of a four year study conducted to establish correlations between downhole geophysical logs and geotechnical parameters for overburden formations in western Canadian surface mines. One study objective was to identify and refine methods that would increase the quantity of geotechnical data while maintaining quality and reducing collection costs. Databases consisted of geotechnical classification and strength test results and geophysical logs from an operating coal mine and a proposed coal mine site. The suite of logs included density, multi-channel sonic, short-spaced neutron, focused electric resistivity, natural gamma and calliper logs. The study utilized data from approximately 50 drillholes selected from an overall database of some 400 holes. All holes were logged uncased with water as the drillhole fluid. In preparation for correlation analyses, the geotechnical data were edited to remove data gaps and the recorded core sample depths adjusted where necessary to ensure alignment with the digitized geophysical logs. Good correlations obtained include uniaxial compressive strength vs. sonic log, moisture content vs. density log and laboratory bulk density vs. sonic log.

INTRODUCTION

The Downhole Geophysics Research Project was undertaken as a joint venture by coal and oilsands mining operators, consultants, geophysical companies, research organizations and government agencies. From 1986 to 1990, the feasibility of determining geotechnical and other parameters from downhole geophysical logs in overburden formations was assessed and a number of empirical correlations were developed. This paper is about geotechnical parameters, the approach used to ensure data consistency and the procedures adopted to establish correlations with various geophysical logs. Data were obtained from coal overburden at two locations in Alberta - Highvale Mine, an operating mine with a coal production of about 14 mta located 80 km west of Edmonton, and a coal prospect at Big Valley 100 km northeast of Calgary.

OBJECTIVE

The objective was to identify and refine methods that will improve the collection of geotechnical data for coal mines in western Canada. This would be accomplished if geotechnical parameters could be reliably predicted from geophysical logs. Benefits would thus accrue from an increase in quantity of data as well as a reduction in collection costs.

AVAILABLE DATA

Geotechnical and geophysical data from both sites were obtained from drillholes put down through overburden overlying the coal measures. The regional overburden materials are composed of gently dipping sequences of mudstones, siltstones and sandstones of Upper Cretaceous and Tertiary age. Overlying this succession are Pleistocene

[1] This Paper was previously published in Proceedings of the 4th International MGLS/KEGS Symposium on Borehole Geophysics for Minerals, Geotechnical and Groundwater Applications; Toronto, 18-22 August, 1991.

deposits consisting of clay till containing ice thrusted blocks of Tertiary rock. Total overburden thickness varies to about 50 m in the areas studied. Montmorillonite, an expansive clay which can jeopardize pitwall stability, is a common clay mineral at both sites, particularly in the bentonitic units.

All geophysical logs were run in drillholes from which geotechnical samples were recovered.

Geotechnical Database

The geotechnical data used in the study were retrieved both from in-house files and from the computerized database maintained by TransAlta Utilities Corporation, owner of Highvale Mine. The database contains data from over 400 drillholes. The test parameters used were derived from the following geotechnical tests:

- natural moisture content
- liquid and plastic limits
- bulk density
- uniaxial compression
- grain size

Geophysical Database

Geophysical logging of all geotechnical drillholes was conducted by BPB Instruments who produced the following logs:

- Gamma (API)
- Caliper (cm)
- Long Spaced Density (sdu)
- Bed Resolution Density (sbrdu)
- Linear Density (gm/cc)
- Focussed Electric (ohm-m)
- Short Sonic (micro-sec/ft)
- Medium Sonic (micro-sec/ft)
- Long Sonic (micro-sec/ft)
- Short-Spaced Neutron (snu)

All holes were logged uncased with water as the drillhole fluid.

METHODOLOGY

Geotechnical

Cross-plots of the geotechnical data were generated to confirm consistency of the data. An example of the good correlation between plasticity index and liquid limit for the Highvale data is shown on the Plasticity Chart where the bestfit line through the data points has a correlation coefficient essentially equal to unity (Fig. 1). The triangular plot of three classification test parameters (Fig. 2) shows the overburden materials exhibit, relative to the variation in natural water content and clay fraction, a large range in plasticity index which is indicative of materials with variable

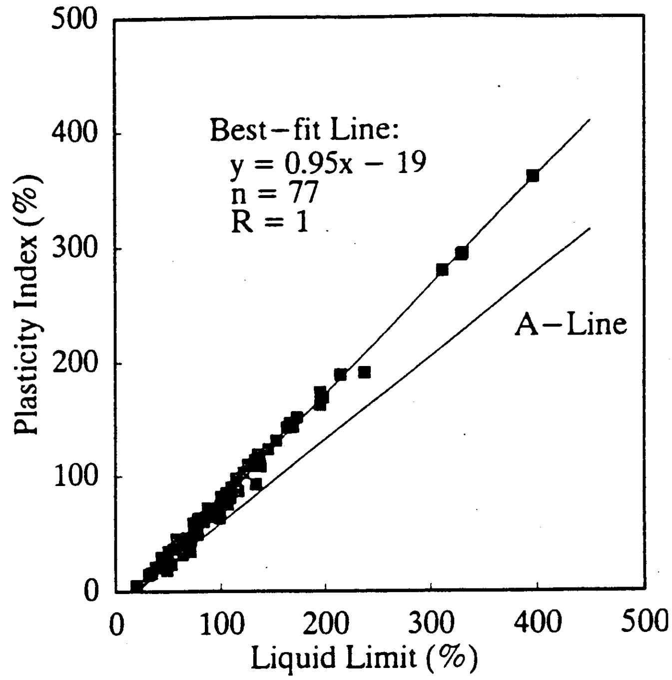


Figure 1 Plasticity Chart - Highvale formations

bentonitic contents.

Adjustment of the core depths was found to be necessary, however, to improve correlations between geotechnical parameters and geophysical logs. The approach adopted took advantage of the fact that both extremely hard sandstone lenses (hardbands) and weak bentonitic seams are present in the overburden formations. These strata not only were extensively sampled geotechnically but also exhibited pronounced sonic log signatures. By selecting samples with uniaxial compressive strengths greater than 10 MPa and plotting the digitized short sonic log readings over a depth interval 0.5m above and below the recorded sample depth,

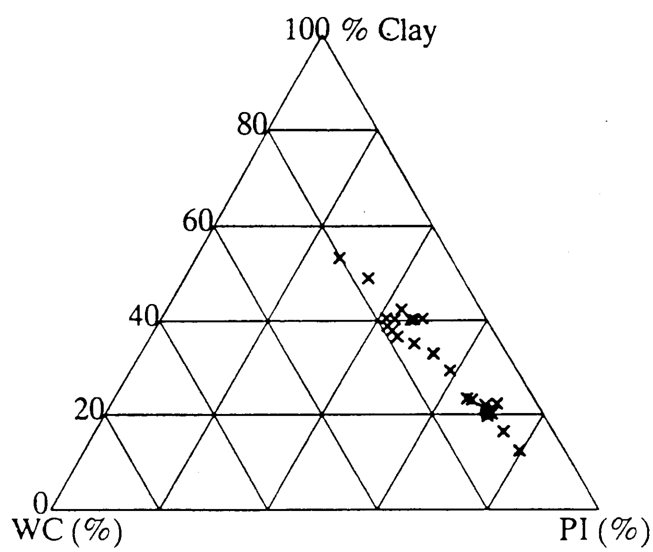


Figure 2. Classification parameters

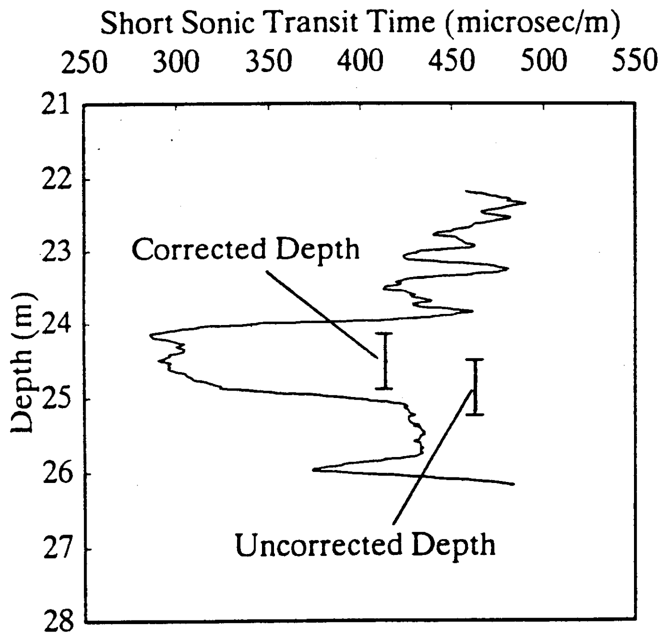


Figure 3. Typical depth correction for uniaxial compression test specimens. The trough in the sonic log corresponds to a hardband with a strength of about 18 MPa

it was possible to realign the sample with the characteristic trough or low point on the sonic signature as indicated on Figure 3. A similar approach was used for very low strength (less than 2 MPa) bentonitic seams except that the sample was aligned with the peaks or high values on the short sonic curve. Depth corrections were generally not possible for specimens having compressive strengths intermediate between 2 and 10 MPa since a characteristic peak or trough on the curve was rarely present. The magnitude of depth corrections applied to uniaxial compression test samples varied from 0 to a maximum of 0.7m as depicted on Figure 4.

A second method of sample depth alignment was used in which the laboratory-determined sample bulk density and linear density log values were compared to obtain the depth of best match. A histogram of depth corrections applied on a density log basis is illustrated on Figure 5. This procedure resulted in enhanced correlations especially between moisture content and density logs. The correlation coefficient, R, increased from about 0.8 for unadjusted data to 0.96 after sample depth alignment (Fig. 6). For correlations with the uniaxial compressive strength, average values of each of the sonic logs (short, medium and long sonic) were used for samples having strengths less than 10 MPa, whereas minimum sonic values were used for samples with strengths greater than 10 MPa.

For all other correlations, the various geophysical logs over depth intervals corresponding to the corrected geotechnical sample depths were downloaded from the database and the average, maximum and minimum values were in turn correlated with each geotechnical parameter.

Geophysical

Based on cross-plots conducted early in the program, it was determined that individual geophysical logs did not require depth adjustment with respect to other logs. In

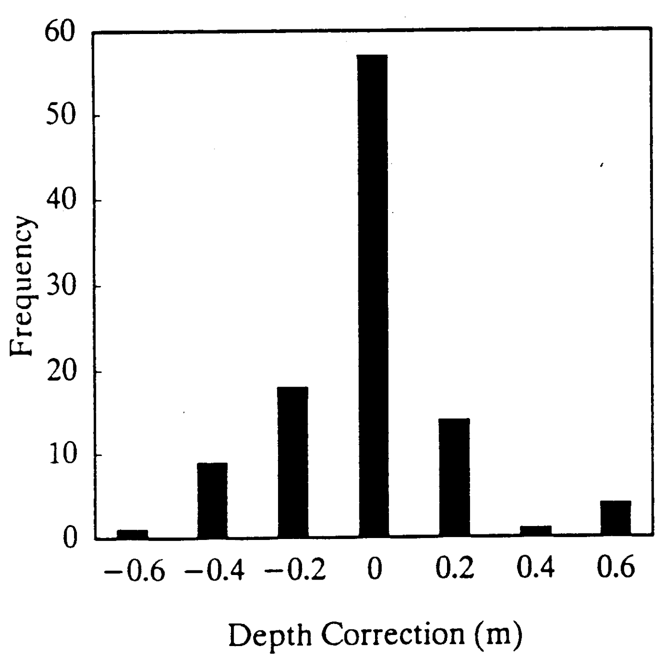


Figure 4. Sample depth corrections - sonic log basis.

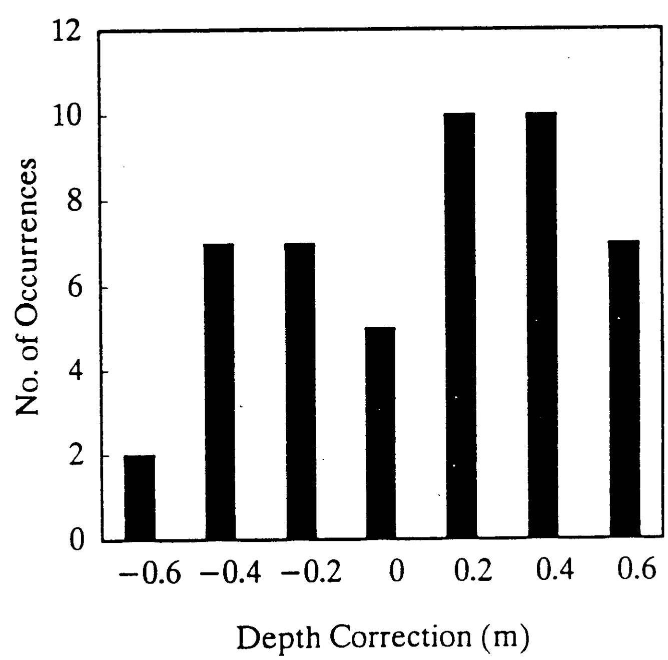


Figure 5. Sample depth corrections - density log basis.

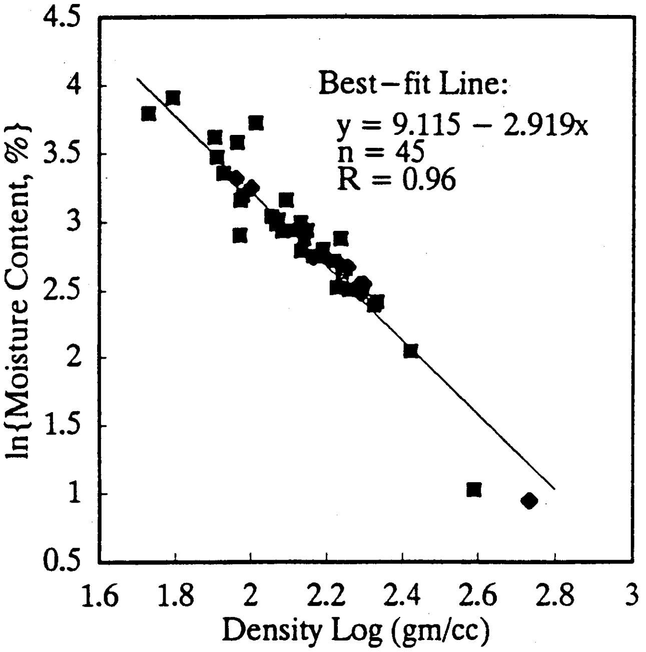


Figure 6. Moisture content vs. density log.

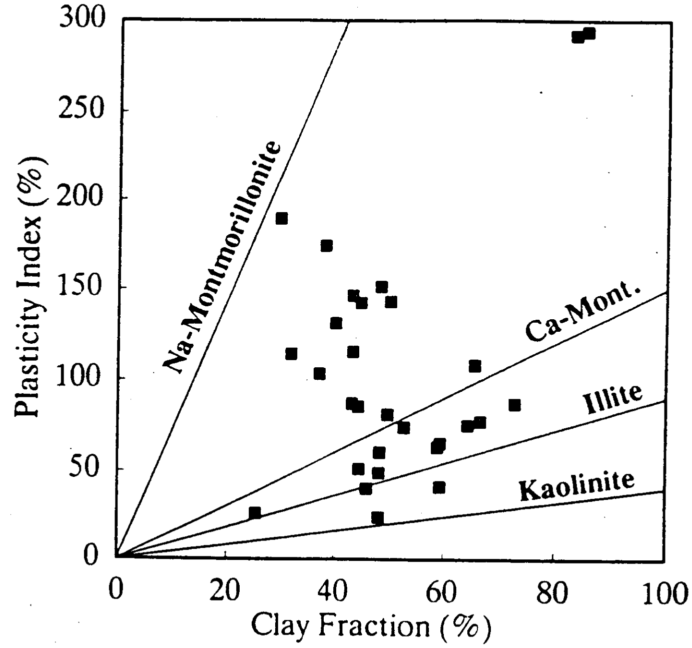


Figure 7. Activity Chart.

addition the standard 9m/min logging rate was found to be satisfactory for correlation studies and did not need to be reduced.

CORRELATIONS

General

After compilation and editing in tabular form, the geotechnical and geophysical parameters were correlated using the linear regression procedures in the Lotus 1-2-3, version 3.0, computer program.

In addition, a few cross-plots of geotechnical parameters were prepared to illustrate the degree of consistency within the data set. These plots included the Plasticity Chart (Fig. 1) and 3-parameter plot (Fig. 2) discussed previously as well as the Activity Chart (Fig. 7), a graph of specimen dry density vs. moisture content (Fig. 8) and finally a plot of uniaxial compressive strength vs. secant modulus (Fig. 9).

The Activity Chart for Highvale formations (Fig. 7) indicates a wide range in clay mineral composition encompassing both sodium and calcium montmorillonite as well as illite and kaolinite. The relatively high concentration of sodium montmorillonite denotes a potentially active material with an affinity for water and a potential for high change in volume with change in moisture content. The effect of variable montmorillonite content in the geologic formations on geophysical log responses is difficult to predict.

The relationship between dry density and moisture content determined in the laboratory from representative core samples shows a good correlation on a log-normal plot (Fig. 8). Another interesting, if not unexpected, relationship is the very good correlation between the uniaxial compressive strength and the secant modulus shown on Figure 9. The secant modulus is defined as the ratio of the compressive

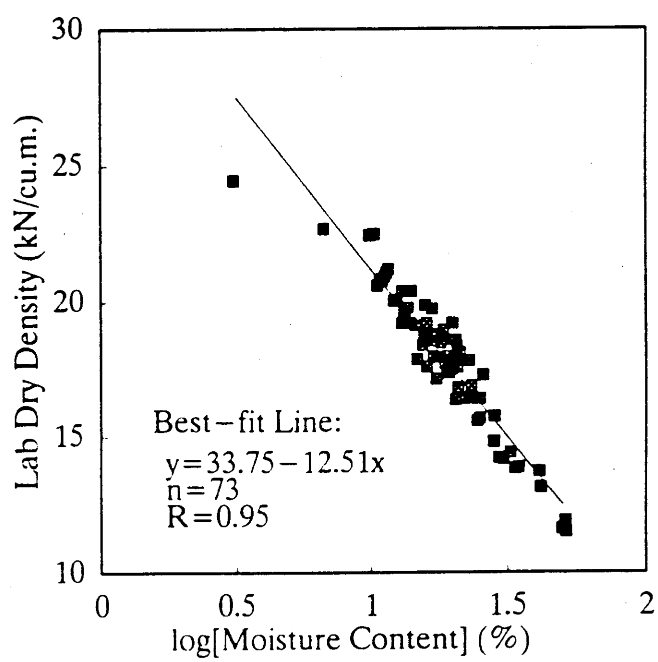


Figure 8. Laboratory dry density vs. moisture content.

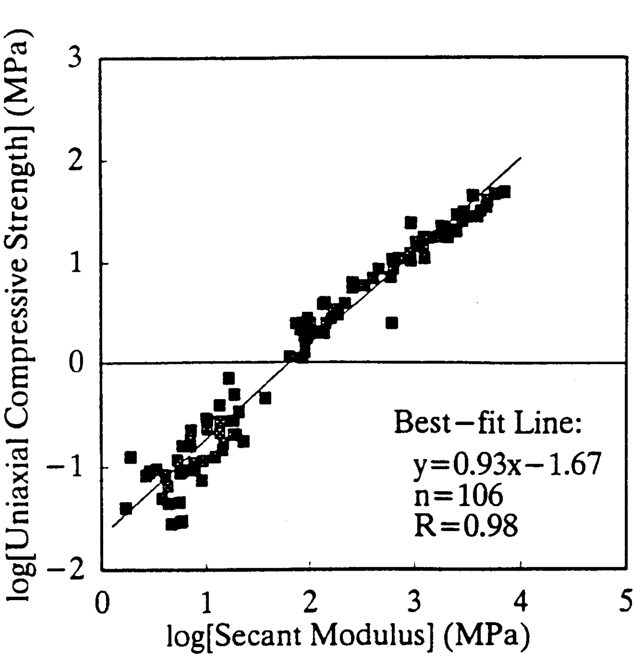


Figure 9. Compressive strength vs. secant modulus.

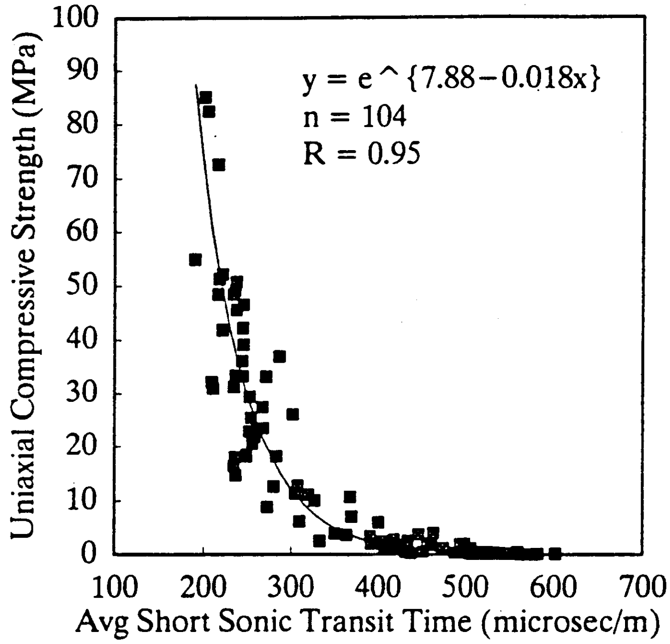


Figure 10. Compressive strength vs. sonic log.

strength to the strain required to cause failure, i.e. the slope of the line extending from the origin to the point of failure on the stress-strain curve.

The geophysical data for the desired geotechnical sample depth intervals were downloaded from the VAX database and manipulated to obtain the maximum, minimum and average values for each log and depth interval. Upon tabulating and aligning with the geotechnical data in a Lotus 1-2-3 spreadsheet, the correlation procedure commenced.

Regression Analyses

A significant number of satisfactory (R = 0.7 to 0.8) to very good (R = 0.9+) correlations were obtained from the regression analyses conducted. Most correlations were derived from linear regression analyses, although two relations were obtained manually from curvilinear analysis. A summary of satisfactory correlations is given in Table 1.

The good correlation between uniaxial compressive strength and short sonic transit time is illustrated on Figure 10 and comparable relations by other investigators are shown on Figure 11. Data on concrete cores tend to support the relationship from the present study (Fig. 12).

The usefulness of the sonic log-compressive strength relationship becomes apparent when preparing maps of the occurrence of hardbands and other difficult-to-dig strata prior to mining. Whether a material is diggable or not depends not only on the digging capability of the stripping equipment but also on the formation strength (O'Regan 1987; Wade et al, 1987; Wade, 1989). Since strength data from core samples rarely provide sufficient areal coverage to permit other than rudimentary mapping, sonic logs, which are generally run in both cored and uncored holes, can be used to provide additional strength data and thus significantly improve areal coverage. The development of a

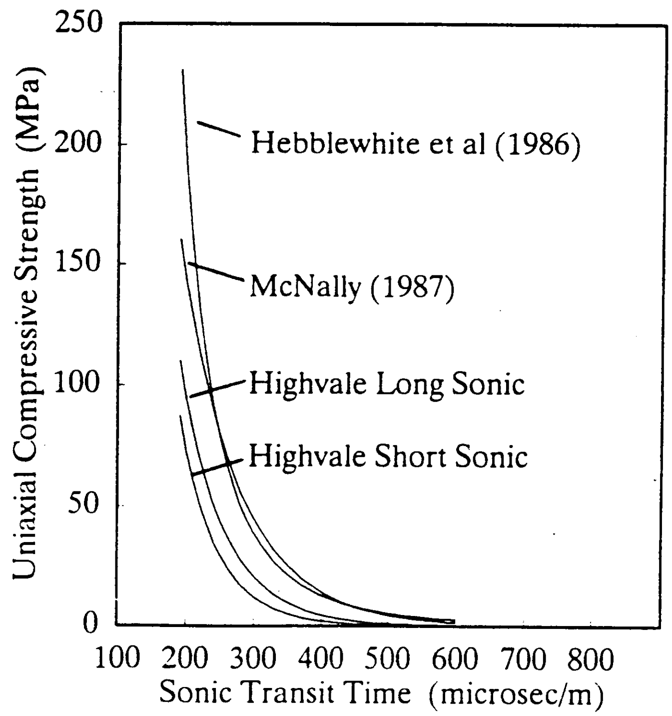


Figure 11. Comparative strength-sonic relations.

Table 1. Summary of satisfactory correlations

Parameters y	x	Equation Obtained	n	R
i) Geotechnical/Geophysical Correlations				
Bulk Density	Nt	y = 1137 + 0.48x	127	0.78
Bulk Density	LSD	y = 2605 - 0.44x	88	0.75
Bulk Density	LD	y = 1.043 - 0.108x	46	0.98
Bulk Density	SS	y = 3180 - 2.1x	120	0.81
Bulk Density	log(SS)	y = 7434 - 1969x	120	0.79
log(Bulk Density)	SS	y = 3.53 - 0.00004x	120	0.80
Moisture Content	SS	y = 0.3x - 27.2	121	0.76
Moisture Content	S-wave	y = 20.67 + 0.01x	14	0.96
ln(Moisture Content)	ln(LD)	y = 9.12 - 2.92x	45	0.96
ln(Moisture Content)	ln(LD)	y = 9.12 - 2.92x	128	0.77*
ln(Moisture Content)	ln(LDxRES/MC)	y = 2.95 - 0.53ln(x)	45	0.91
loglog(MC)	log(RES)	y = 0.68 - 0.61x	123	0.77
log(MC)	ln(Nt)	y = 33.7 - 4.07x	127	0.79
% Clay	LD	y = 203 - 74x	43	0.79
ln(% Clay)	LD	y = 8.06 - 2.03x	43	0.84
Qu	SS	y = 153 - 0.65x + 0.0068x^2	104	0.89
Qu	SS	y = e^(7.88 - 0.018x)	104	0.95
ln(Qu)	ln(SS)	y = 38.42 - 6.35x	104	0.94
1000ln(Qu/SS)	SS	y = 29.2 - 1.063x	104	0.96
1000ln(Qu/SS)	ln(SS)	y = 139.9 - 22.88x	104	0.98
Qu	LS	y = e^(7.31 - 0.015x)	59	0.87
ln(Qu)	ln(LS)	y = 32.04 -5.10x	59	0.85
1000ln(Qu/LS)	LS	y = 30.87 - 0.067x	59	0.93
1000ln(Qu/LS)	ln(LS)	y = 140.39 - 22.82x	59	0.95
Qu	log(RES)	y = 52.8x - 49.2	92	0.77
Qu (sst only)	log(RES)	y = 26.4x - 54.5	44	0.87
Secant Modulus	SS	y = 5707 - 10.98x	85	0.73
Secant Modulus	SS	y = 17394 - 65.2x + 0.61x^2	85	0.84
Secant Modulus	ln(SS)	y = 25294 - 11512x	85	0.78
ln(Secant Modulus)	ln(SS)	y = 23.51 - 10.09x	85	0.85
ln(Secant Modulus)	SS	y = 6.72 - 0.0104x	85	0.86

Table 1. (Continued)

Parameters y	x	Equation Obtained	n	R
ii) Geophysical Cross-Plots				
avg LS	avg SS	y = 63.53 + 0.903x	59	0.97
avg LS	min SS	y = 58.65 + 1.92x	59	0.95
avg SS	avg LS	y = 2.64 + 0.975x	2711**	0.89
min SS	avg SS	y = 0.998x - 8.717	66	1.00
iii) Geotechnical Cross-Plots				
Dry Density	MC	y = 23.10 - 0.255x	73	0.94
Dry Density	log(MC)	y = 33.75 - 12.51x	73	0.95
Plasticity Index	LL	y = 0.95x - 20.5	74	1.00
Qu	Sec Mod	y = 0.0086x + 1.609	106	0.96
log(Qu)	log(Sec Mod)	y = 0.926x - 1.672	106	0.98

* Prior to sample depth correction
** Data from one hole only

Legend:

avg	=Log value averaged over a particular depth
Bulk Density	= lab determined bulk density (gm/cc)
Dry Density	= lab determined dry density (kN/cu.m)
LD	= Linear Density Log (gm/cc)
LL	= liquid limit (%)
ln	= natural logarithm
log	= common logarithm
Log	= geophysical Log
LSD	= Long Spaced Density Log (sdu)
LS	= Long Spaced Sonic Log (microsec/m)
MC	= moisture content (%)
min	= minimum Log value over a particular depth interval
n	= No. of observations
Nt	= Neutron Log (snu)
Qu	= uniaxial compressive strength (MPa)
R	= coefficient of correlation
RES	= Resistivity Log (ohm-m)
Sec Mod	= secant modulus (MPa)
SS	= Short Spaced Sonic Log (microsec/m)
sst	= sandstone
S-wave	= Shear Wave Transit Time (microsec/m)

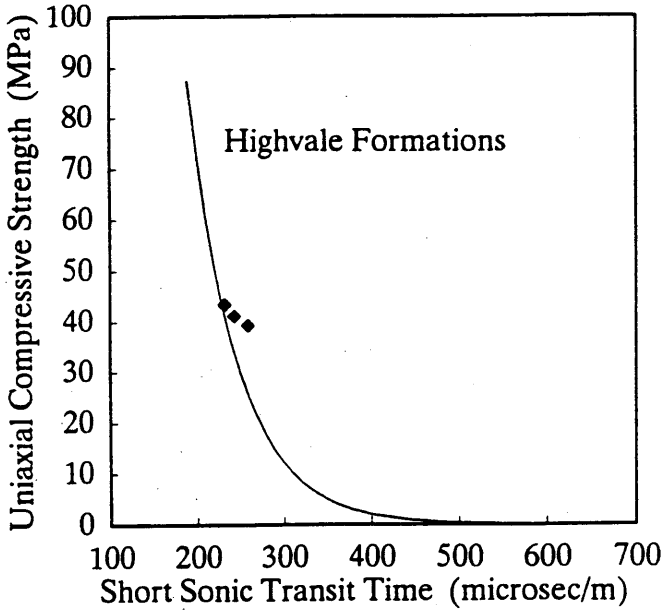


Figure 12. Strength-sonic data from tests on cores of 80-year old concrete (Monenco, 1981).

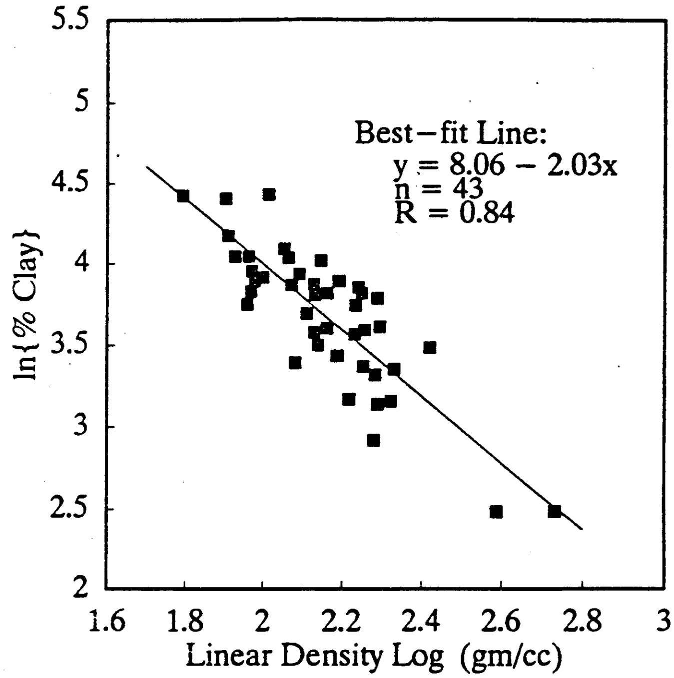


Figure 14. % clay fraction vs. linear density log.

Scanning Routine for scanning digitized sonic logs to ascertain the depth and thickness of undiggable strata further enhances the mapping procedure (Wade and Peterson, 1991).

Geophysical logs which show promise in predicting geotechnical classification parameters for overburden include linear density, neutron and sonic logs. After depth corrections have been applied, for example, the linear density log correlates well with the bulk density determined in the laboratory (Fig. 13) and reasonably well with the moisture content (Fig. 6) and the clay fraction (Fig. 14). With no depth correction applied, the correlation coefficient for the moisture content-linear density relation reduces from 0.96 to 0.77 (Table 1). For bentonitic-rich materials, the relationship between moisture content and shear wave transit time as computed from sonic logs appears to be consistent with published data (Fig. 15).

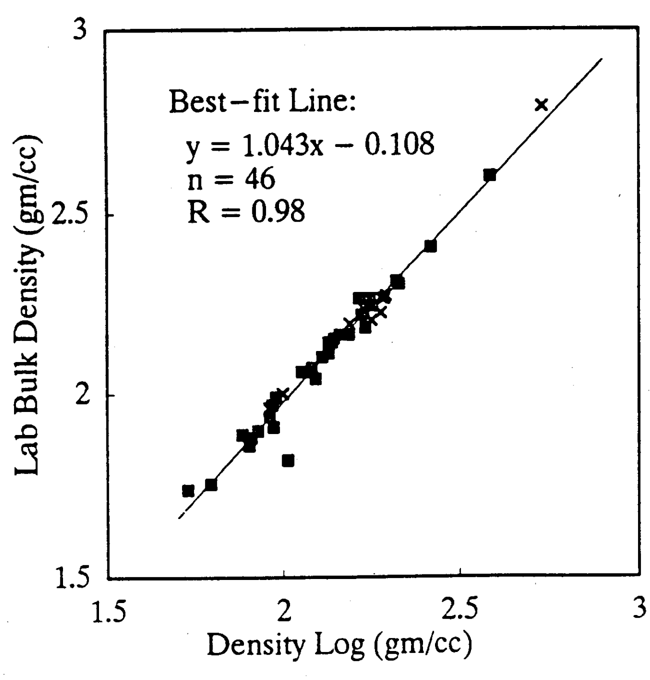


Figure 13. Lab bulk density vs. linear density log.

CONCLUSIONS

From the results of the study, it was concluded that, by depth aligning geotechnical samples with the corresponding geophysical logs, a number of different geotechnical parameters can successfully be correlated with individual logs. Some enhancement of the correlation is often possible when groups of logs are used. All of the satisfactory correlations obtained, however, would be considered more reliable if, by increasing the number of data points, the generated correlation coefficients were not materially reduced.

Although correlation coefficients are sometimes enhanced by using logarithmic functions of the parameters, the procedure may actually de-sensitize the resulting relationship thereby diminishing its effectiveness as a predictive expression.

Additional work needs to be done to determine whether other important geotechnical parameters, such as Modulus of Elasticity, Poisson's Ratio, consolidation

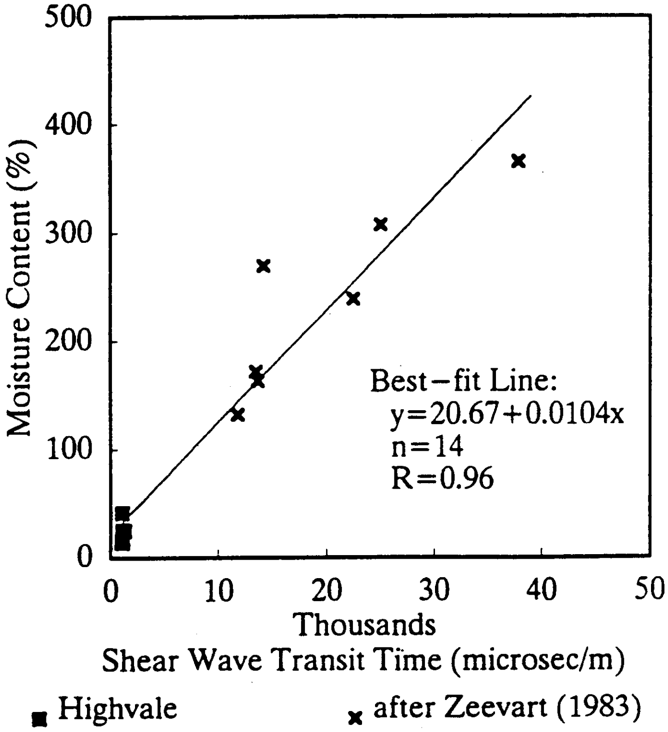


Figure 15. Moisture content vs. shear wave transit time for bentonitic materials.

coefficient and angle of internal friction, to name a few, can be correlated with geophysical logs. The effects of carbonaceous content of overburden strata on geophysical log response, and thus on geotechnical parameter correlations, should also be studied further.

ACKNOWLEDGEMENTS

The author wishes to thank TransAlta Utilities Corporation, owner of Highvale Mine, for their support and permission to publish this paper. Appreciation is also expressed to Mr. T.C.(Mackie) Lam who assisted with the computer work.

REFERENCES

Hebblewhite, B.K., Blackwood, R.L., Holt, G.E., Mokula, P.A., Richmond, A., O'Regan, G., Mallett, C. and Enever, J. (1986) Rock Mechanics and Stability of Excavations, Australasian Coal Mining Practice, Monograph 12, AIMM, Victoria, Australia.

McNally, G.H. (1987) Geotechnical Applications and Interpretations of Downhole Geophysical Logs, ACIRL End-of-Grant Report to the Australian Coal Association; Project A21; July.

Monenco (1981) Investigation of St. Mary's Compensating Works, Great Lakes Power Limited, Sault Ste. Marie, Ontario, Report prepared for Great Lakes Power Ltd. by Monenco Ltd.; August.

O'Regan, G., Davies, A.L. and Ellery, B.I. (1987) Correlation of Bucketwheel Performance with Geotechnical Properties of Overburden at Goonyella Mine, Australia, Proc. Int'l Sym. on Continuous Surface Mining; Ed. by Golosinski & Boehm; Trans Tech Publications, Fed. Rep. of Germany; pp. 381-396.

Wade, N.H., Ogilvie, G. A. and Krzanowski, R.M. (1987) Assessment of BWE Diggability from Geotechnical, Geological and Geophysical Parameters, ibid, pp.375-380.

Wade, N.H. (1989) Overburden Diggability Criteria for Surface Coal Mines, Proc. Western Canada Coal Geoscience Forum, Alberta Research Council, Edmonton; April.

Wade, N.H. and Peterson, T.W. (1991) Interpretation of Geophysical Logs using a Scanning Routine, Accepted for presentation at the Second Canadian Conference on Computer Applications in the Mineral Industry, to be held in Vancouver, B.C.; Sept.

Zeevart, L. (1983) Foundation Engineering for Difficult Subsoil Conditions, Van Nostrand Reinhold, New York; 2nd Ed; p.552.

Applied Surface Geophysical Applications

The Foothills and Mountain Coalfield Experience

Allister Peach[1] and Kathryn Cochrane[2]

Abstract

In 1988 a consortium of coal mining companies, contractors and government agencies recognized the potential for application of surface geophysical techniques in the foothills and mountains of the Canadian Cordillera. Geophysical methods included reflection seismic, direct current (DC) resistivity profiling, electromagnetic induction EM, Max-Min horizontal loop EM, very low frequency (VLF) EM, gravity and ground penetrating radar. With the exception of VLF and ground penetrating radar, each method was successful in delineating coal subcrop or continuity at depth. The applicability and cost effectiveness of various methods in different geological scenarios are discussed.

1.0 Introduction

During the past decade significant advances have been made in the research and development of surface geophysical applications in the coal industry in Canada. Coal mining companies in western Canada in cooperation with the Alberta Office of Coal Research and Technology (AOCRT) have collaborated to evaluate surface geophysics in a broad range of geological and physiographic settings. Previous investigations in the mid 1980's were successful in demonstrating the application and cost effectiveness of seismic and electrical methods in the coalfields in the plains of Alberta.

In 1988 a consortium of coal mining companies including Esso Resources Canada Limited, CrowsNest Resources Limited, Luscar Sterco (1977) Limited, Manalta Coal Limited, Quintette Coal Limited and Smoky River Coal Limited recognized the potential value of extending applications of surface geophysics to the more complex deposits in the foothills and mountain regions of the Canadian Cordillera. A three year agreement was signed between AOCRT, which administers funding provided by the Alberta/Canada Energy Resources Research Fund (A/CERRF), and Esso Resources Canada Limited, representing a joint venture of the aforementioned coal mining companies. This paper will discuss the success of each method and demonstrate the cost effectiveness of integrating surface geophysics with current resource delineation techniques.

---

[1] Esso Resources Canada Limited

[2] Alberta Energy - Alberta Office of Coal Research and Technology

2.0 Technical Objectives

The technical objectives of the research project were :

a) to determine the applicability and limitations of surface geophysical methods in increasingly more complex geological settings from the outer foothills to mountain terrain.

b) to demonstrate the cost effectiveness of surface geophysics as an exploration and resource delineation tool.

c) to determine possible refinements, solutions to problems and enhancements for future applications.

The surface geophysical techniques that were tested included reflection seismic, direct current (DC) resistivity profiling, electromagnetics including EM Induction, Max-Min Horizontal Loop and VLF, gravity and ground penetrating radar.

3.0 Surface Geophysical Methods

3.1 Reflection Seismic

Reflection seismic is a direct method of locating stratified units below the ground surface that, because of the differences in densities display contrasting acoustic impedances. Coal, due to its low density, is considered to be an excellent material for seismic reflection as compared to the higher density lithologies usually associated with coal. When energy is sent into the ground using explosive charges, the large contrast in acoustic impedance at the coal/rock contact can be measured by an array of variably spaced geophones. Recent advances in data acquisition techniques and processing have allowed resolution of strata to within 30 meters of surface, an acceptable depth for surface mining.

3.2 Direct Current Profiling

Direct Current Profiling (DC) measures the resistivity of lithological units in the ground. A current is sent into the ground through current electrodes and the current distribution is profiled by potential electrodes. Coal is generally a highly resistive material while most associated rock types such as shale, mudstone, siltstone and argillaceous sandstones have clay minerals, are less resistive and act as conductors. The resistivity contrast allows coal beds to be located at subcrop and the continuity traced within 30 meters of the surface if field conditions are favourable. An abundance of sandstones and gravel of similar resistivity to coal may mask the resistivity profile.

3.3 Electromagnetic Methods

Electromagnetic methods, including induced EM-31, Max-Min and VLF, map the resistivity of subsurface materials by measuring the time varying magnetic field. A vertical or horizontal magnetic dipole transmitter or a very low energy transmitter creates the current flow which induces a magnetic field which is sensed by a receiver dipole. The amplitude and phase of the magnetic field is related to the resistivity of the materials in the subsurface. As coal is a highly resistive material in contrast to associated rock types, it is able to be mapped at subcrop and at shallow depths.

3.4 Gravity

The differences in density of various lithological units creates the potential to locate gravity anomalies. Gravity anomalies are the result of horizontal variation in density created by structural phenomena in the stratigraphy. As coal exhibits low density compared to associated lithologies any structure that has juxtaposed coal and contrasting higher density materials can be measured by a gravity meter. The method is extremely sensitive to elevation changes and resolution in the varying topography of the foothills and mountains can be difficult if proper control is not maintained.

3.5 Ground Penetrating Radar

The use of ground penetrating radar in subsurface detection of coal and structural resolution is in the relatively early stages of development. The method entails sending a very high frequency electromagnetic wave into the ground and receiving differing energy reflections of contrasting lithologies through detectors, much like seismic. The depth capability of the method is restricted in that the energy is absorbed by the groundwater and is rapidly lost at depth. In addition the radar signal is attenuated through ground materials as a result of the ground conductivity. In the near surface, radar has the potential to produce images with higher resolution than seismic.

4.0 Project Results (location reference FIGURE 1)

4.1 Phase 1 : 1988 - 89 Program Summary

During Phase 1 of the project a reflection seismic program was conducted at Luscar's Coal Valley Mine and at the Smoky River Mine in the outer and inner foothills of west central Alberta.

Seismic profiles produced at Coal Valley were consistent with the interpreted stratigraphy and structure of the deposit based upon existing drilling information. The presence of an additional thrust fault was also inferred. Confirmation of this fault was delayed to Phase 3 due mainly to budgetary constraints.

Seismic profiles at Smoky River were consistent with known stratigraphy and structure although the complex topography created difficulty in resolving statics in processing and subsequently caused the interpretation to be tenuous in some parts of the section. Small scale faulting was difficult to evaluate.

The most important conclusion from Phase 1 was that better data on subsurface velocity control should be obtained from density and sonic logs from downhole geophysics during preliminary drilling.

4.2 Phase 2 : 1989 - 90 Program Summary

During Phase 2 refinements continued on the acquisition and processing parameters for reflection seismic. The project was joined by the British Columbia Geological Survey through funding from the British Columbia/Canada Mineral Development Agreement (BCMDA).

Seismic processing refinements were tested using seismic data from Esso's Tower property in the outer foothills of Alberta. A reflection seismic program at Esso's Springhill property in Nova Scotia demonstrated the differences in resolution of 5 and 10 meter spacing on geophones in the upper 300 meters of subsurface.

This knowledge was then used to complete a reflection program for the CrowsNest Telkwa deposit in northwestern British Columbia. The Telkwa deposit is well known through drilling and the results of the seismic program were in strong agreement with the existing interpretation.

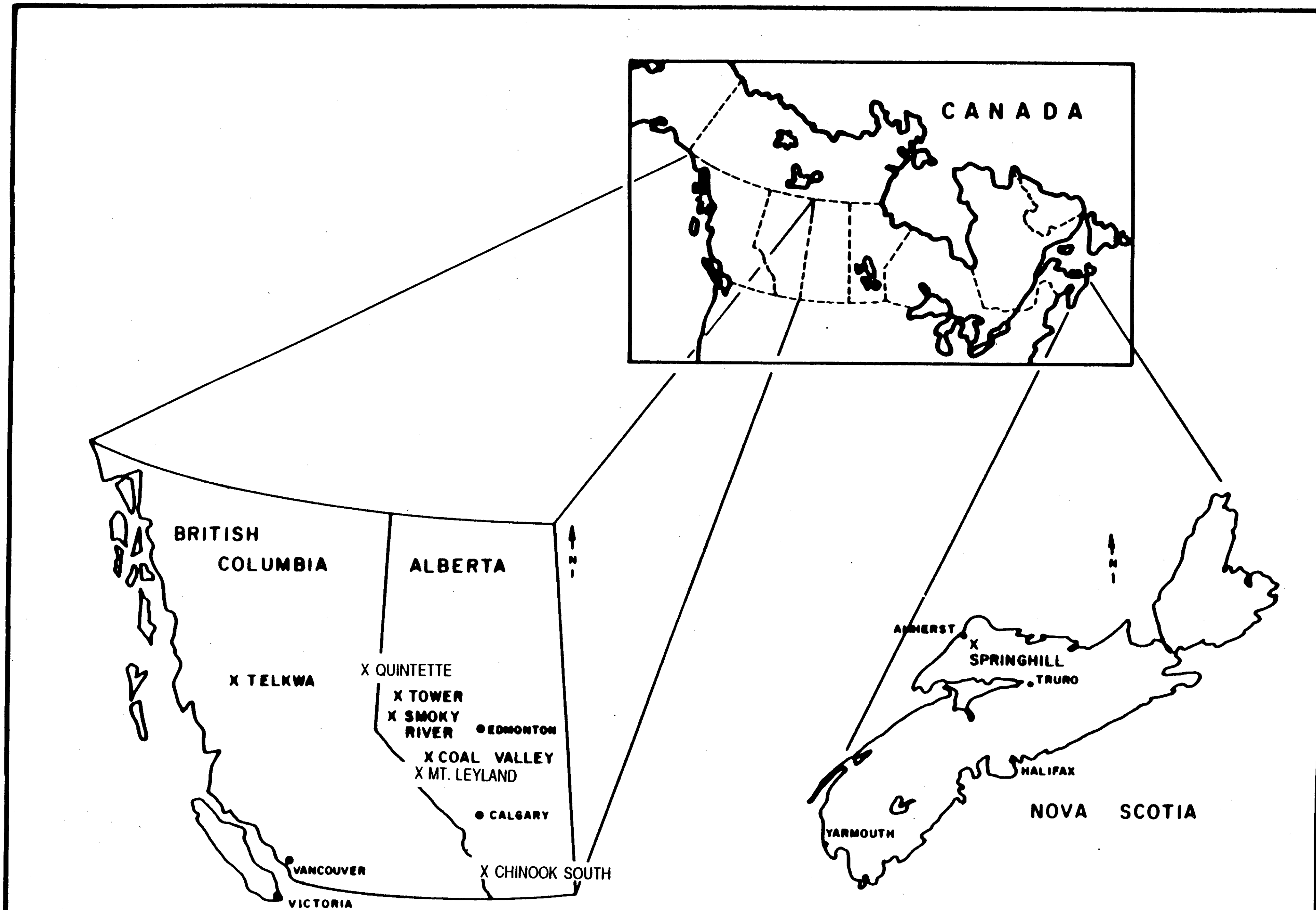


Figure 1 Location of Test Sites

At the conclusion of Phase 2 it had been demonstrated that reflection seismic was useful in obtaining structural and coal continuity profiles in gently dipping strata underlying outer foothills topography as well as in more complex deposits of the inner foothills and intermontane coal basins of British Columbia and Nova Scotia.

4.3 Phase 3 : 1990 - 91 Program Summary

During Phase 3 fieldwork was completed at 4 sites including Coal Valley, Mt. Leyland, Chinook South and Quintette. The consortium was joined by the Alberta Research Council through the Alberta Geological Survey and by Geo-Physi-Con Geophysical Consultants.

At Coal Valley, drilling confirmed the presence of a thrust fault that had been detected in Phase 1 using reflection seismic.

At Manalta's Mt. Leyland property reflection seismic was completed in an exploration property in the inner foothills that is known to be complexly folded. The coal bearing units were visible to seismic with the exception of vertical or near vertical strata. Large scale diffraction patterns affected the ability to resolve small scale faulting.

At the Chinook South coal deposit owned by Manalta a variety of non-seismic geophysical methods were tested. The program successfully used DC profiling, fixed frequency electromagnetic induction (EM-31) and Max-Min horizontal loop EM to detect coal continuity and subcrop location. VLF-EM was attempted but was not successful at this location. Gravity was also used to locate previously known and undiscovered coal occurrences. Chinook South demonstrated the portability and accessibility of such methods in mountainous terrain.

With the exception of gravity and VLF-EM, all the above non-seismic methods were tested at Quintette. Similar detection of coal continuity and subcrop location were completed successfully, even in extremely rugged terrain. Ground penetrating radar was also attempted but proved to be impractical due to equipment malfunction and breakdown.

The Alberta Geological Survey completed structural interpretations of seismic data from Smoky River and Mt. Leyland and integrated the information with existing geological data to further refine the interpretation at both sites.

The results from Phase 3 indicated that reflection seismic can be used in complex geological settings and that some processing techniques can be used to enhance the data.

Phase 3 also demonstrated that electrical and electromagnetic methods were generally successful in coal subcrop location and coal continuity near surface. It was shown that gravity is a viable application for locating substantial pods of coal in difficult terrain. Phase 3 also demonstrated that most of the geophysical methods work in the more rugged conditions and are useful methods of exploration.

5.0 Cost Effectiveness of Surface Geophysics

To demonstrate the cost effectiveness of surface geophysics a comparison of resource/reserve delineation programs using the traditional drilling approach and the integrated surface geophysics/drilling approach is required. Three hypothetical cases are reviewed to illustrate the potential differences in interpretation and cost that may be realized by integration of surface geophysics.

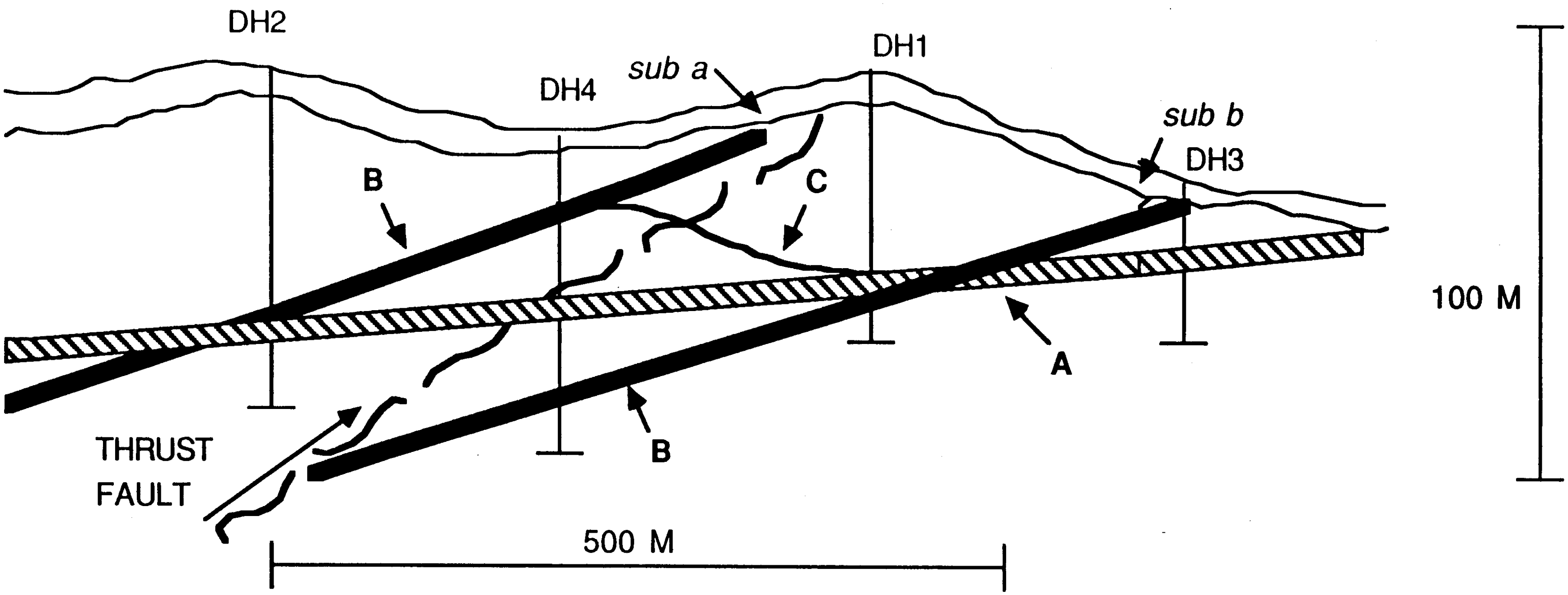
Criteria for resource/reserve delineation are taken from Geological Survey of Canada Paper 88-21: A Standardized Coal Resource/Reserve Reporting System for Canada (Hughes et al, 1988).

5.1 Case 1 : Surface Geophysics - An Interpretative Tool

The first hypothetical case involves a Moderate/Complex Geology Type coal deposit in a foothills scenario. Figure 2 depicts moderately undulating terrain underlain by coal-bearing strata complicated by thrust faulting. Typical data point spacing for determining an indicated resource on this deposit is 500 meters, while 200 meters spacing is required for a measured reserve.

Using a typical drilling approach, the coal seams intersected in drillholes DH1 and DH2 indicate a simple gently dipping monocline (A). DH3 completed for subcrop definition supports the simple interpretation with minor flexure near surface and it is seen that not until drillhole DH4 is completed to confirm continuity at depth, that the structural complexity is revealed (B). It is even possible to interpret an anticline (C) unless DH4 is drilled deeper to intersect the repeated coal section.

Using an integrated geophysics/drilling approach, drillhole DH1 would indicate the presence of a coal-bearing sequence. Geophysical logging from the hole, including density, sonic and



<u>INTERPRETATION</u>

A - SIMPLE EXTRAPOLATION BETWEEN DRILLHOLES 1 & 2- NO STRUCTURE

B - MORE COMPLICATED/THRUST FAULTED SEQUENCE : REPEATED COAL SECTION, STEEPER DIPPING COAL SEAMS, TWO SUBCROP OCCURRENCES. GEOPHYSICS COULD HAVE DETECTED STRUCTURE(REFLECTION SEISMIC) AND COAL SUBCROP (EM OR DIRECT CURRENT METHODS)

C - POSSIBLE GENTLE ANTICLINE IF DRILLHOLE 4 NOT COMPLETED DEEP ENOUGH

FIGURE 2. SCHEMATIC CROSS-SECTION FOOTHILLS - COMPLEX GEOLOGY TYPE

resistivity, allow modelling of the responses for reflection seismic and electrical methods. The geophysical stage would include completion of a reflection seismic program with a geophone spacing of 5 meters to observe near surface strata and completion of an electromagnetic survey to locate the subcrop locations at *sub a* and *sub b*. The second phase of drilling would include a different location for drillholes DH2, DH3 and DH4 and may allow deletion of one drillhole.

5.2 Case 2 : Moderate to Complex Reserve Delineation

The second hypothetical case involves delineation of a measured coal reserve in a Moderate to Complex Geology Type coal deposit in the inner foothills scenario. The deposit consists of a 5 meter thick coal zone at a depth range of 0 to 100 meters that covers an area of 3 by 5 kilometers (15 sq.km.). Delineation of a measured reserve requires spacing of 250 meters for a total of 273 data points (Figure 3).

Traditionally drillholes would be required at all grid nodes resulting in 273 holes totalling 13 650 meters. Total reserve delineation cost at $50.00 per meter of drilling is $682,500.00.

Using 30 kilometers of reflection seismic at 5 meter geophone spacing for coal continuity and structure, a 20 kilometer electromagnetic program (Max-Min or EM-31) for subcrop delineation and 160 drillholes for the remaining data point control would complete the reserve delineation requirements. Total cost of the integrated geophysics and drilling program is $600,000.00. In addition the drilling program can be more intelligently planned with regards to structure and depth.

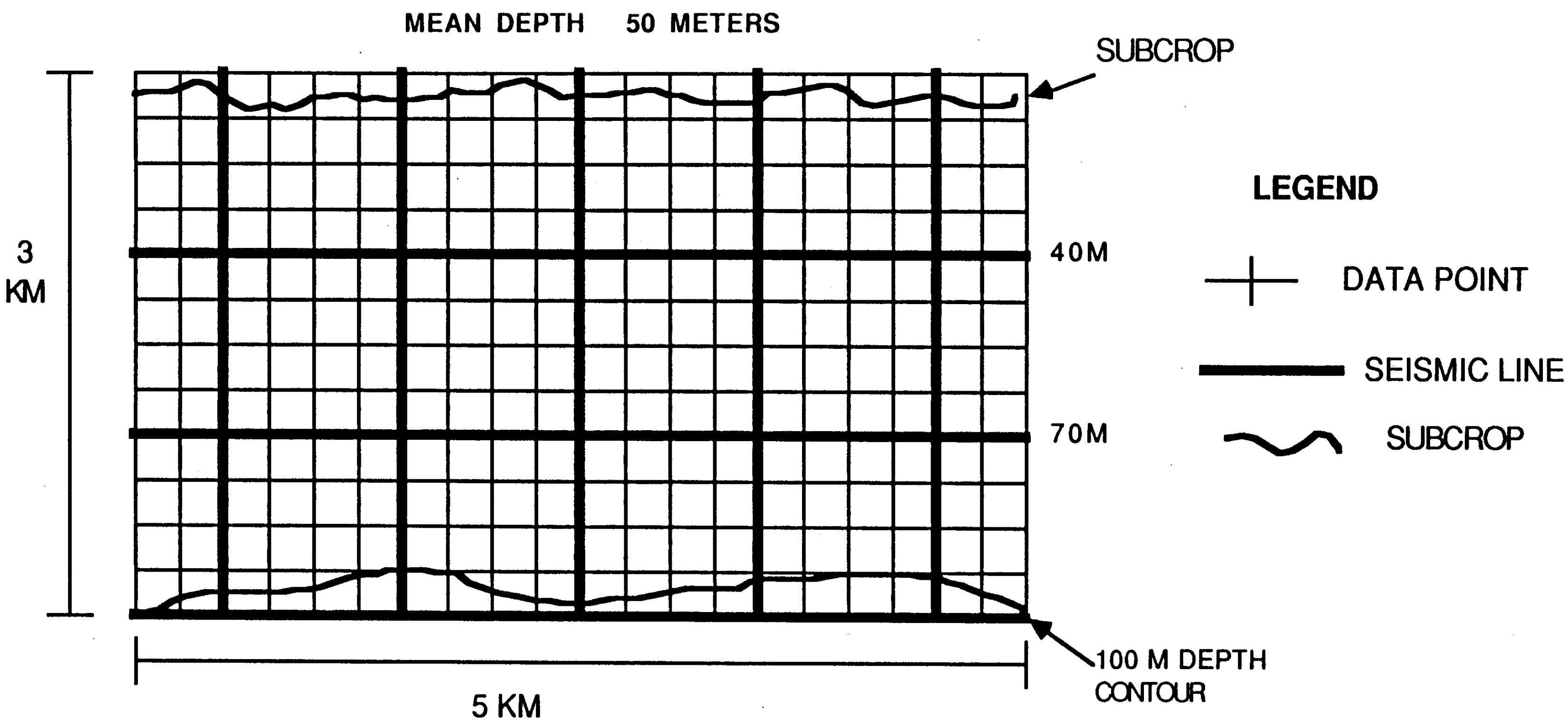
5.3 Case 3 : Underground Deposit Reserve Delineation

The third hypothetical case involves a Low Type B to Moderate Geology Type coal deposit in an underground setting with a reserve 3 by 5 kilometers in size at an average depth of 350 meters. The data point spacing is 500 meters resulting in 77 data points being required to delineate the deposit (Figure 4).

Using drilling as the sole delineation tool, a total of 77 holes would be required for a total of 26 950 meters. Total cost of drilling at $75.00 per meter is $2,021,250.00.

An integrated geophysics and drilling program would include 30 kilometers of reflection seismic and 39 drillholes including grid nodes and crosspoints for geophysical interpretation for a total of 13 650 meters. Total cost of the reserve delineation is

FIGURE 3. FOOTHILLS COAL DEPOSIT - GEOLOGY TYPE MODERATE - COMPLEX

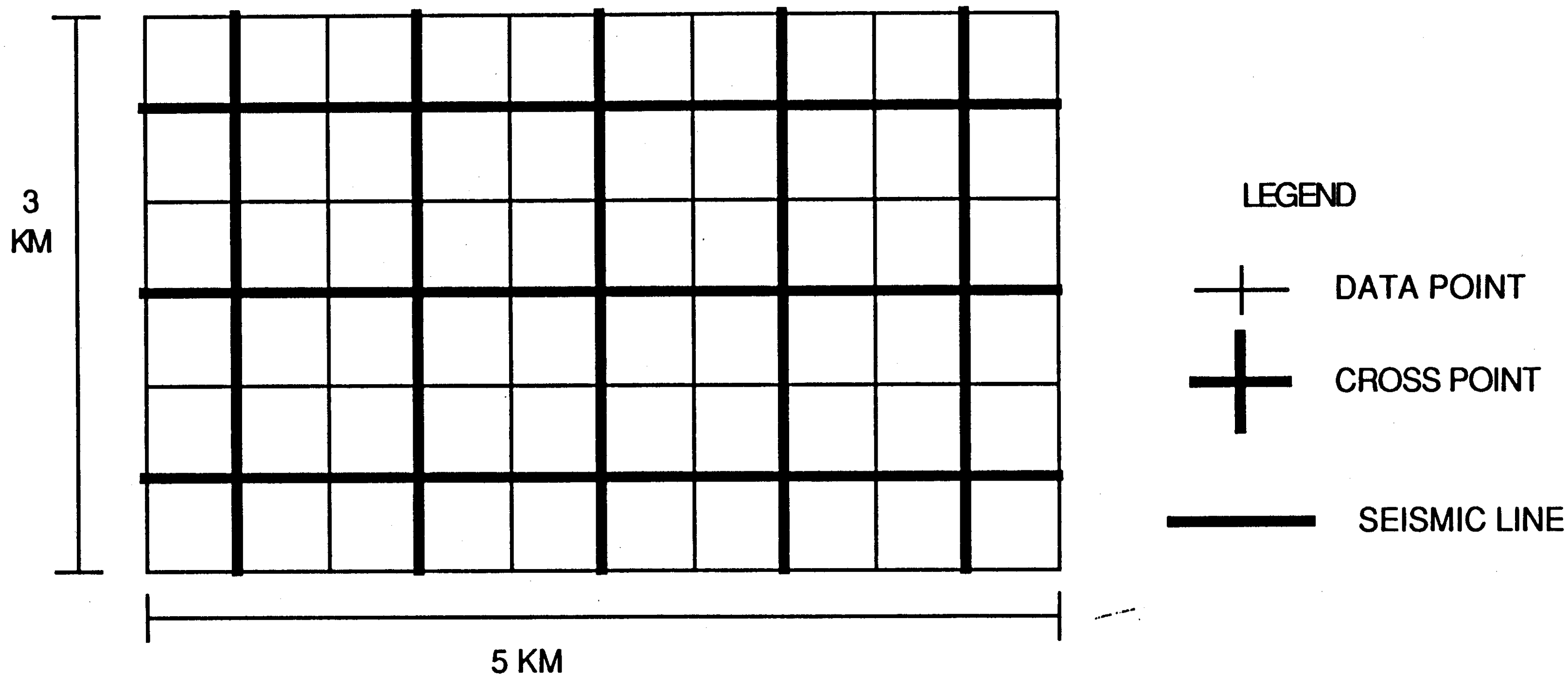


273 DATA POINTS @ SPACING 250 METERS

RESERVE DELINEATION ALTERNATIVES

A	B
273 DHS @ 50 M	30 KM REFLECTION SEISMIC
TOTAL COST : $ 682.5 K	20 KM SUBCROP DELINEATION EM/MAX-MIN
	160 DHS (DATA PTS)
	TOTAL COST : $ 600 K

FIGURE 4. UNDERGROUND COAL DEPOSIT - GEOLOGY TYPE LOW TO MODERATE
350 METERS IN DEPTH



77 DATA POINTS @ SPACING 500 METERS

RESERVE DELINEATION ALTERNATIVES

A	B
77 DHS @ 350 M	30 KM REFLECTION SEISMIC
TOTAL COST : $ 2.02 M	39 DHS (CROSS PTS & DATA PTS)
	TOTAL COST : $ 1.2 M

$1,203,750.00. The geophysics would also identify areas of structural concern and allow better planning of drilling.

6.0 Conclusion

From the work that has been completed during the three year research project it can be seen that surface geophysical methods can be used successfully in more difficult geological and topographical settings.

It has been realized that key information such as the density, sonic and resistivity logs are critical to create synthetic profiles that will enable the geophysicist to assess the success of a geophysical application in any particular scenario.

It has been realized that there are limitations to each method and that a thorough review of the known geology be conducted between the geologist and geophysicist. A site visit to review field conditions is recommended.

It has been demonstrated that integrating a surface geophysical program with exploration and development drilling can be cost effective and allow more intelligent planning and placement of drillholes as well as be a valuable tool in completion of the proper geological interpretation.

7.0 References

Henderson, J.D. & Nikols, D., 1988. Cost Effectiveness of surface geophysics. Preprint from CIM District 6 Meeting, Fernie, B.C.

Hoffman,G.L., Henderson,J.D. & Sartorelli,A.N., 1989. Foothills/Mountain Surface Geophysics Project: Phase 1 Report.

Hoffman,G.L. & Sartorelli, A.N., 1990. Foothills/Mountain Surface Geophysics Project: Phase 2 Report.

Hoffman,G.L. & Sartorelli, A.N., 1991. Foothills/Mountain Surface Geophysics Project: Phase 3 Report.

Hoffman,G.L. & Sartorelli, A.N., 1991. Foothills/Mountain Surface Geophysics Project: Final Summary Report.

Hughes,J.D., Klatzel-Mudry, L. & Nikols, D.J., 1989. A Standardized Coal Resource/Reserve Reporting System for Canada. G.S.C. Paper 88-21.

Acknowledgements

The authors would like to acknowledge the following government agencies, companies, and their representatives for their contribution to this project.

Alberta/Canada Energy Resources Research Fund
Alberta Energy - Alberta Office of Coal Research & Technology
    Tom Sneddon & Kathryn Cochrane
Alberta Research Council - Alberta Geological Survey
    Denis Nikols, John Pawlowicz, Mark Fenton & Willem Langenberg
British Columbia Geological Survey
    Greg MacKillop & Barry Ryan
Geo-Physi-Con Geophysical Consultants
    Jim Henderson, Tony Sartorelli & Mike Pesowski
Coal Mining Research Company - Project manager 1988-90
    Robert Wilson & Georgia Hoffman
Westwater Mining Limited - Project manager 1990-91
    Georgia Hoffman
Esso Resources Canada Limited
    Allister Peach
CrowsNest Resources Limited
    Brian McKinstry & Ted Hannah
Luscar Sterco (1977) Limited
    Gary Johnston
Manalta Coal Limited
    Randy Karst & Peter Van Katwyk
Quintette Coal Limited
    Dave Johnson, Rob Booker & Kevin Sharman
Smoky River Coal Limited
    Richard Dawson, Bruce Mattson & Brian Klappstein
The University of Calgary - Geophysical Department
    Don Lawton

GEOLOGY AND COAL RESOURCES OF THE CADOMIN MAPSHEET (83F/3), ALBERTA

WILLEM LANGENBERG

Alberta Geological Survey, Alberta Research Council,
Box 8330, Station 'F', Edmonton, Alberta, T6H 5X2 Canada

ABSTRACT

Cretaceous and Tertiary strata are known from outcrop in the northeast part of the area, while older rocks are present in the southwest and in the subsurface as known from oil and gas wells. The major structures are, from north to south: the Pedley Thrust, Coalspur Triangle Zone (formerly called Coalspur Anticline), Entrance Syncline, Mercoal Thrust, Brazeau Flats, Brazeau Thrust, Brazeau Syncline, Grave Flats Thrust, Cadomin Syncline and Nikanassin Thrust. The Pedley Thrust appears to have at least 1 km of southwest directed displacement. This fault defines the Coalspur Triangle Zone. The Mercoal Thrust may have about 2 km of southwest directed displacement and defines a triangle zone that probably formed before the Coalspur Triangle Zone. The Brazeau Thrust shows at least 3 km of northeast directed movements and places Blackstone shales on top of the Brazeau Formation. The Brazeau Syncline has an overturned southwest limb and is a tight fold. The Nikanassin Thrust forms the boundary with the Front Ranges. Three cross sections through the area were obtained by the TRIPOD Structural Geological Information System. The cross sections have been shortened by deformation in the order of 27 to 35 percent.

The economic coal seams of the Tertiary Coalspur Formation are of high volatile C rank and are present in three parallel bands in the Entrance Syncline and Coalspur Triangle Zone, where they have been mined extensively in the past. The Mercoal band is the southernmost and dips about 30 degrees to the northeast. The Coalspur band is in the middle and dips generally to the southwest. The Robb band is the northernmost band, contains northeast dipping strata and is less deformed than the Coalspur band. In the Entrance Syncline, the Coalspur coals are buried at various depths (up to 1 km) and may form exploration targets for coalbed methane. The Lower Cretaceous coals of the Luscar Group are exploited in the Cadomin-Luscar Coal field. The rank of these coals range from high volatile A to low volatile bituminous. Where buried (for example west of the town of Cadomin), they may form exploration targets for coalbed methane.

INTRODUCTION

The Cadomin mapsheet (see figure 1 for location) contains large parts of the Cadomin-Luscar, Coalspur and McLeod River coal fields. The Cadomin-Luscar Coalfield contains two major coal mines and the Coalspur Coalfield

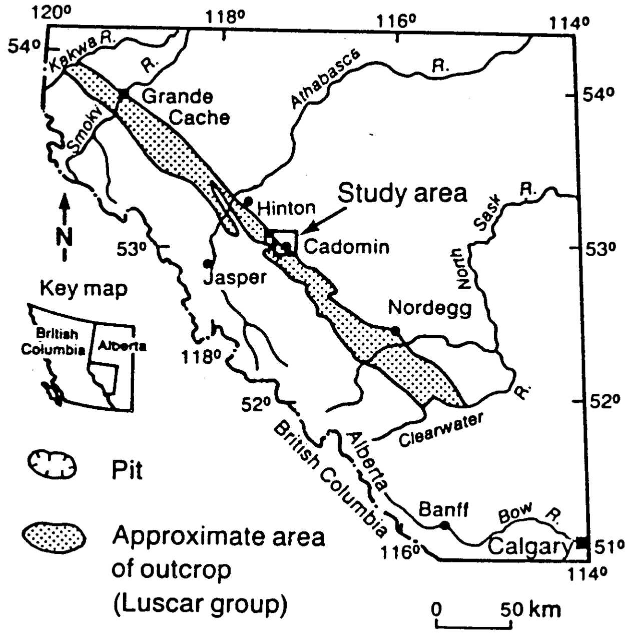


Figure 1. Location of the Cadomin (83F/3) mapsheet.

possibly contains Alberta's next new coal mine. For this reason, the AGS has performed mapping of the coal-bearing strata in this mapsheet. Another reason is that no recent geological maps exist of this area; the only available geological map is from MacKay (1929). Fieldwork was performed during the summer of 1989 and 1990. The area is heavily forested and consequently the exposure of bedrock is limited. The main exposures are along McLeod River, Gregg River and MacKenzie Creek. Other exposures are along roads, rail-road cuts and ridges formed by competent rock units.

Hill (1980) mapped part of the Cadomin mapsheet and the coal quality of the Cadomin-Luscar coal field was presented by Langenberg et al. (1989).

Methodology

Information from outcrops and drill holes were collected with the TRIPOD Structural Geological Information System (Charlesworth et al., 1989). The coordinates of outcrops and drill holes are in a UTM grid (Zone 11). A geological map and cross sections were constructed with this computer system.

Outcrops

Information from outcrops were entered in the database. Additional outcrop information was obtained from Johnston, 1985, and unpublished coal company exploration reports.

Drill holes

Information from coal exploration drill holes were entered in the database. Most of these holes were drilled by Manalta, Luscar, Denison and Crows Nest Resources (Shell).

In addition, stratigraphic picks (done by the ERCB) from oil and gas wells in this area were entered in the database.

STRATIGRAPHY

Some 4500 meters of Cretaceous and Tertiary strata are present in the area. The stratigraphic units are discussed separately from oldest to youngest.

Nikanassin Formation

The largely Jurassic Nikanassin Formation consists of marine and non-marine sandstones and shales. It is uncomformably overlain by the Cadomin Formation of the Luscar Group.

Luscar Group

The largely Albian Luscar Group consists of the Cadomin, Gladstone, Moosebar and Gates formations (Langenberg and McMechan, 1983). This group, which is equivalent to the Mannville Group of central Alberta, shows both marine and non-marine sedimentary environments and is about 600m thick.

Cadomin Formation

The Cadomin Formation was deposited in alluvial fans and on braided river pediment plains with chert pebble conglomerates, and is unconformably overlying the Nikanassin Formation. Much of the Cadomin that developed in the west as alluvial fans was transported and reworked into the northwest trending Spirit River channel that developed parallel to the mountain front during the early Aptian. The Cadomin likely represents a major drop in sea level and a major sequence boundary. The Cadomin Formation is an excellent marker horizon, both at the surface and in the subsurface.

Gladstone Formation

The Lower Cretaceous (Aptian-Albian) Gladstone Formation is equivalent to the coal-bearing Gething Formation in northeastern British Columbia and the oilsands-bearing McMurray Formation in northeastern Alberta. The formation lies conformably on the Cadomin with no apparent major stratigraphic break. The section at Cadomin shows a transition from well drained alluvial plain deposits with thin coals near the base, to coastal plain deposits with thicker coals near the top.

Moosebar Formation

Several marine cycles can be recognized in the Albian Moosebar/lower Gates succession (Macdonald et al., 1988). The fully marine succession includes offshore to lower shoreface (with storm deposits) and shoreface to foreshore (and possibly beach).

The lower Moosebar Formation consists of a series of fine-grained mudstones interbedded with sharp based siltstones and thin sandstones. It is interpreted to have formed in an offshore environment in which storm events periodically deposited thin sand units.

In the upper Moosebar more sandstones (often with hummocky cross stratification) are found. The hummocky cross stratification supports the storm deposited origin and has been found in other locations of the Moosebar to Gates transition. A glauconitic, sharp based pebble conglomerate bed is found in the upper Moosebar and is interpreted as an offshore transgressive deposit. It may in part represent very slow deposition of a condensed section. The boundary between Moosebar and Gates is gradational in a coursening-up succession.

Gates Formation

The Albian Gates Formation consists of the Torrens, Grande Cache and Mountain Park members. Both shallow marine and non-marine sedimentary environments are deduced for this formation.

Torrens Member

The offshore/transgressive deposits of the Moosebar give way upsection to lower shoreface and finally foreshore sediments of the Torrens Member. The trace fossil assemblage is consistent with this interpretation.

The Torrens member consists of massive (though occasionally thinly bedded), fine to very fine grained sandstone. Faint parallel laminations are the predominant structures, with some trough cross bedding and hummocky cross stratification also present. Scour surfaces, with pebble lag deposits, are also common in the unit. Mudstone is a very minor lithology in this cycle and is usually associated with the hummocky cross stratification. Trace fossils may be present. The Torrens is interpreted to be a succession of prograding shorelines sequences, with upper shoreface to foreshore environment transitions being present.

Grande Cache member

The base of the Grande Cache Member in the Cadomin area is the 10 m thick Jewel seam. Additional coal seams are present higher up in the section (such as the R seam, stratigraphically 60 m above the Jewel seam). However, these coal seams are generally uneconomic in the near surface of the Cadomin area. The low-lying Okefenokee Swamp, which is some distance from active shoreline processes, might be a modern day model for these Cretaceous coals.

The sandstones of the lower part of the Grande Cache Member are often brackish until the first major fluvial sandstones are encountered. A brackish water interpretation is based on the presence of trace fossils and the presence of lenticular bedding throughout this interval. Specimens of the siliceous foraminifera *Hippocrepina (?) sp., Miliammina (?) sp.* and *Saccammina sp.* have been recovered from three locations above the Jewel seam in the Cadomin area and are indicative of a shallow brackish (not normal marine) marine environment (John Wall, pers. comm.). The upper part of the Grande Cache member is largely non-marine and contains fining-upward sequences.

Mountain Park Member

The base of the Mountain Park Member is generally defined at the base the first major greenish colored sandstone encountered going upsection from the major coal seams. These sandstones are interpreted to be large scale fluvial deposits and may occur at various stratigraphic levels, making this field mapping criteria somewhat arbitrary if only limited exposure is available.

The contact between the Mountain Park Member and the overlying dark grey shales of the Blackstone Formation is generally sharp.

Blackstone Formation

The formation consists largely of dark marine shale and siltstone, with minor beds of sandstone, bentonite and some ironstone concretions. Some 500 m of Blackstone sediments may be present in the hanging wall of the Brazeau Thrust (figure 3), although there may be structural repeats present. Its age is late Albian to late Turonian. Stott (1963) distinguishes 4 members in this formation, but they are not easily mappable in this poorly exposed area.

Cardium Formation

The Cardium Formation consists of marine sandstone, siltstones and shale. It forms a useful marker horizon for mapping purposes, because it is relatively thin (about 80 m) and the sandstones forms ridges, that are easy recognizable on the aerial photographs. The marine sandstones often contain hummocky cross beds and trace fossils. The Blackstone and Cardium formations form a coarsening upwards succession, indicative of a fall in relative sea-level.

Four members can be recognized in outcrop (Stott, 1963), but are too thin to be mappable on a 1:50 000 scale. The age of the formation is late Turonian to early Coniacian.

Wapiabi Formation

The Wapiabi Formation includes all the beds between the Cardium Formation and the greenish sandstones of the Brazeau Formation and is about 600 m thick. The age of the formation is Turonian to Campanian. Stott(1963) distinguishes 7 members in the Wapiabi Formation. However, the only easily mappable unit on a 1:50 000 scale is formed by the marine sandstones of the Chungo Member. For this reason the Wapiabi Formation is divided into the Upper and Lower Wapiabi members, whereby the base of the Chungo Member is used as marker horizon.

Lower Wapiabi members

The lower members are the Muskiki, Marshybank, Dowling, Thistle and Hanson members, which are dark grey, marine shales and siltstones. The Marshybank Member contains a larger percentage of siltstones and the Thistle Member consists of calcareous shales (Stott, 1963), but these members could not be mapped separately.

Upper Wapiabi members

The upper members are the Chungo and Nomad members. The Chungo consists of about 70 metres of fine grained, often reddish-brown weathering sandstones and minor siltstone. Hummocky cross-stratification and trace fossils, such as *Planolites* sp. and *Skolithos* sp., indicate that these sandstones are largely marine and clearly distinctive from the younger alluvial sandstones of the Brazeau, Coalspur and Paskapoo formations. Marine bivalves can be found.

The Nomad Member consists of dark grey marine shales in between the Chungo sandstones and the greenish sandstones of the Brazeau Formation. This member is about 30 metres thick.

Brazeau Formation

The Brazeau Formation, together with the Coalspur and Paskapoo formations forms part of the Saunders Group (Jerzykiewicz, 1985). The Brazeau Formation consists of about 1200 metres of sandstones, shales and some coal seams above the marine shales of the Wabiabi Formation and below the basal Entrance Conglomerate of the Coalspur Formation. Some exploration for coal in the Brazeau Formation has been performed by Luscar and Crows Nest Resources. Up to 4 metres of coal has been reported in their drilling programs.

Coalspur Formation

The Coalspur Formation (Jerzykiewicz, 1985) contains a 600 metres thick continental succession of interbedded sandstones, mudstones and thick economic coal seams. The base of the Coalspur Formation is the so-called Entrance Conglomerate. Thick coal seams interbedded with coaly shales and numerous bentonites occur in the upper part of the formation. This interval is known as the Coalspur coal zone. The Val d'Or coal seam is at the top of the interval and the Mynheer coal seam is at the bottom. These seams (plus other coal seams) are recognizable in the whole area between Hinton and Coal Valley. The Cretaceous-Tertiary boundary is at the base of the Mynheer coal seam (Jerzykiewicz and Sweet, 1986). The Coalspur Formation represents a nonmarine, fluvially dominated environment of deposition.

Paskapoo Formation

The Paskapoo Formation consists of at least 1000 metres thick alluvial sandstones and mudstones above the uppermost coal seam of the Coalspur Formation, which is the Val d'Or Seam in the Cadomin area.

STRUCTURAL GEOLOGY

The major structures of this foothills area are, from north to south the Pedley Thrust and Coalspur Triangle Zone (formerly called Coalspur Anticline), Entrance Syncline, Mercoal Thrust, Brazeau Flats, Brazeau Thrust, Brazeau Syncline, Grave Flats Thrust, Cadomin Syncline and Nikanassin Thrust. These structures are shown in the maps of figures 2 and 3. The down-plunge cross section BB" is shown in figure 4 and 5.

The Pedley Thrust appears to have at least 1 km of southwest directed displacement. This fault defines the Coalspur Triangle Zone. The Mercoal Thrust may have about 2 km of southwest directed displacement and defines a triangle zone that probably formed before the Coalspur Triangle Zone. The Brazeau Thrust shows at least 3 km of northeast directed movements and places Blackstone shales on top of the Brazeau Formation. The Brazeau Syncline has an overturned southwest limb and is a tight fold. The Nikanassin Thrust forms the boundary with the Front Ranges. The cross sections have been shortened by deformation in the order of 27 to 35 percent.

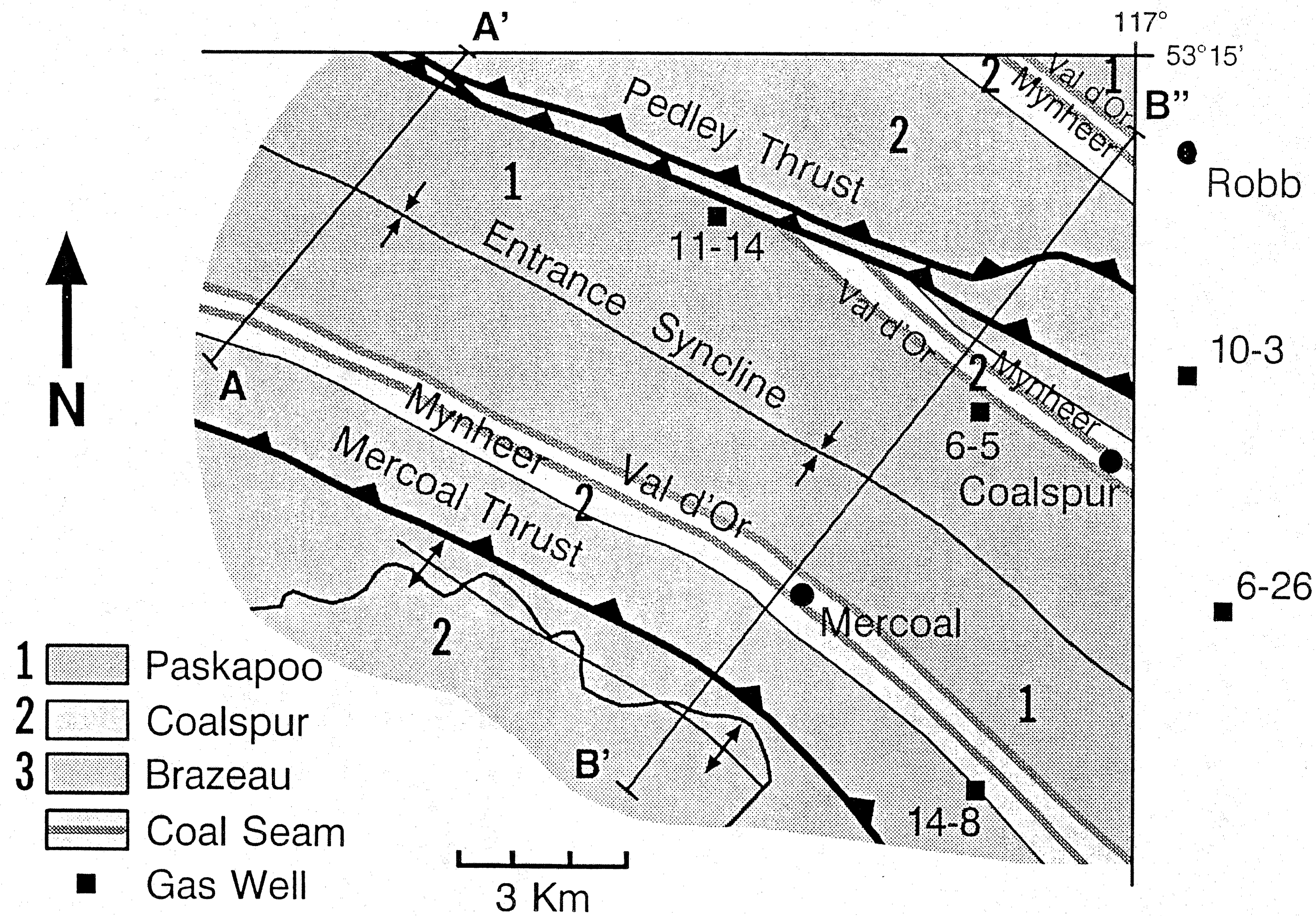


Figure 2. Geological map of part of the northern half of the Cadomin mapsheet.

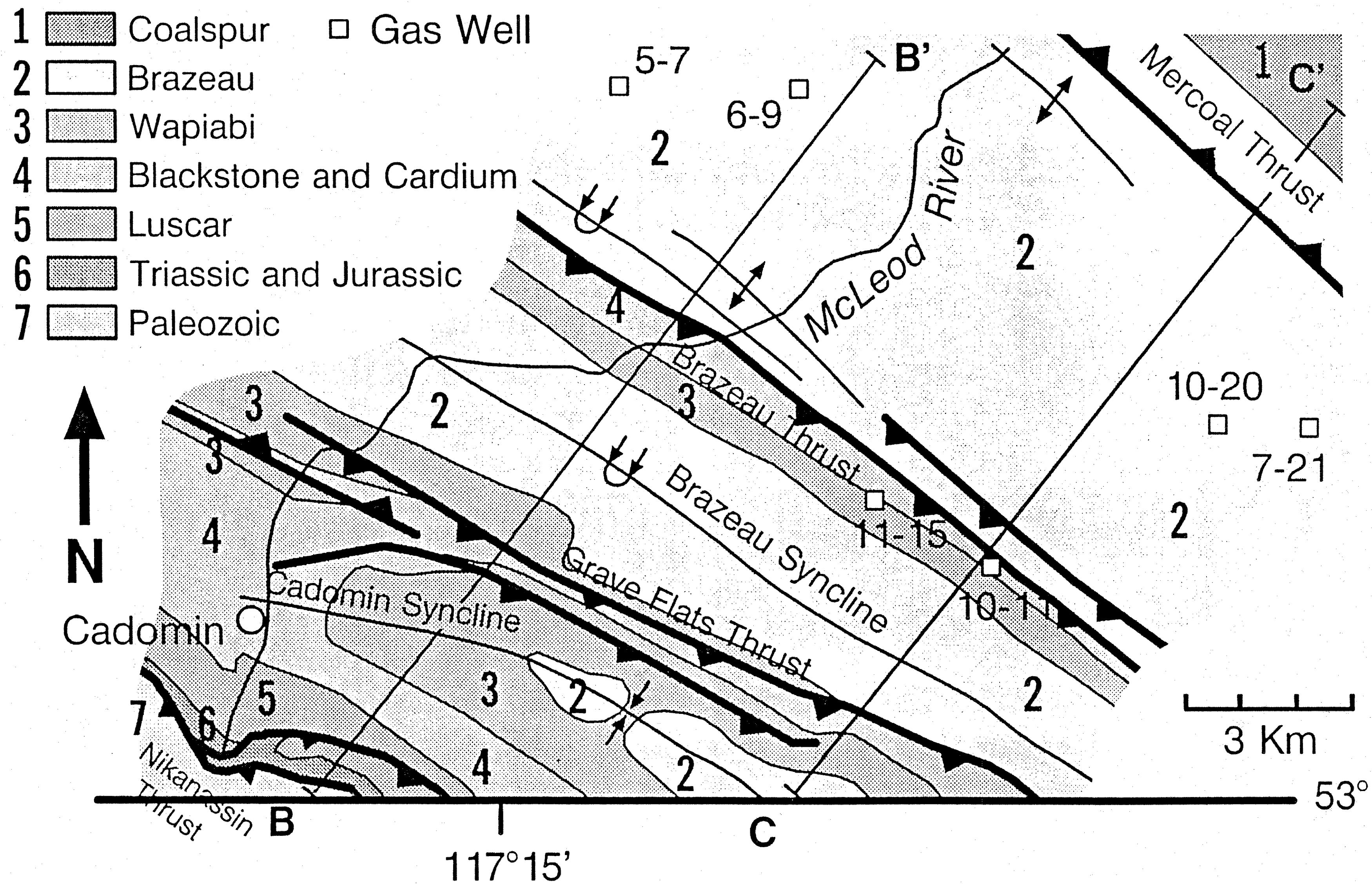


Figure 3. Geological map of part of the southern half of the Cadomin mapsheet.

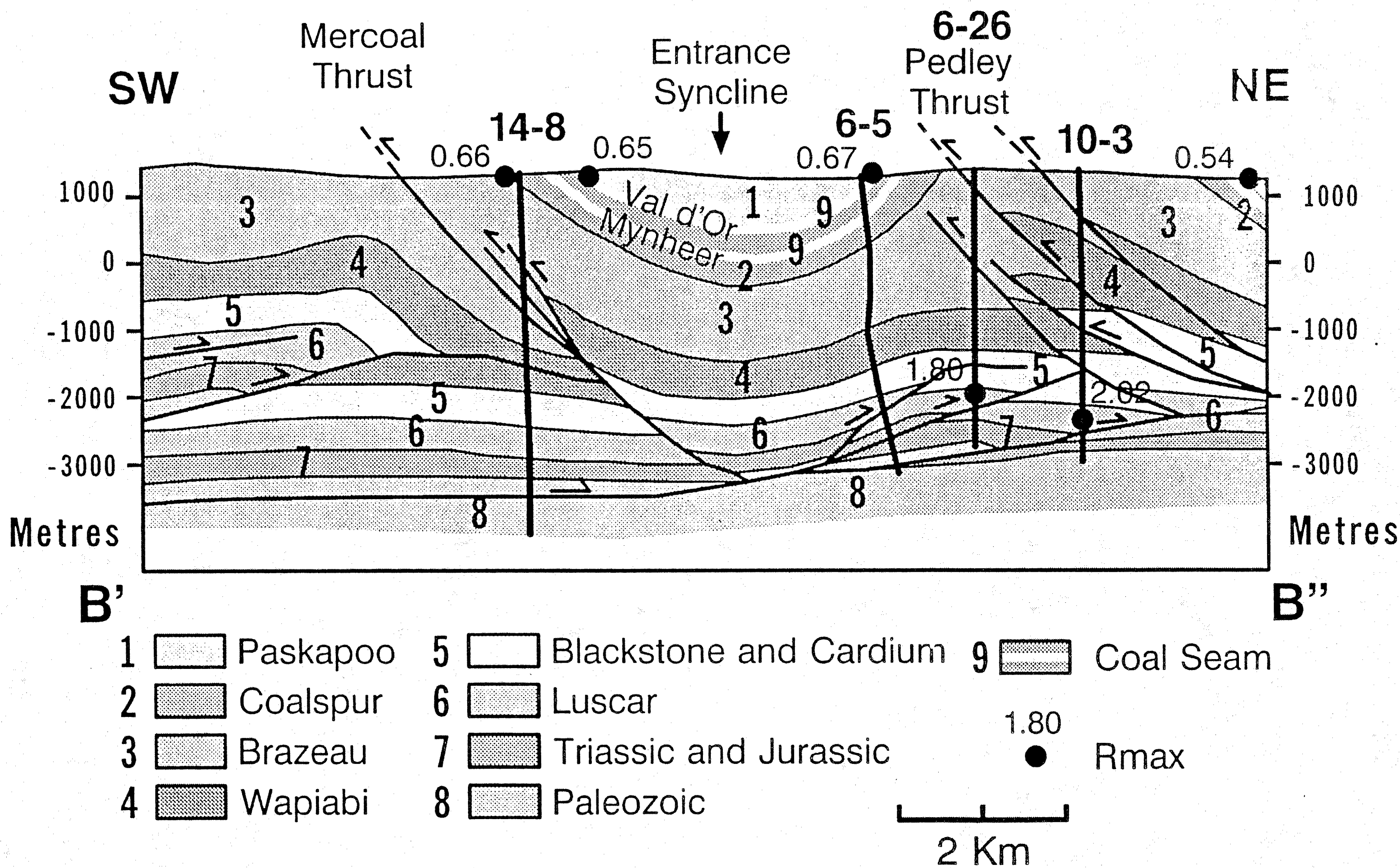


Figure 4. Cross section B'B" through the northern half of the Cadomin mapsheet.

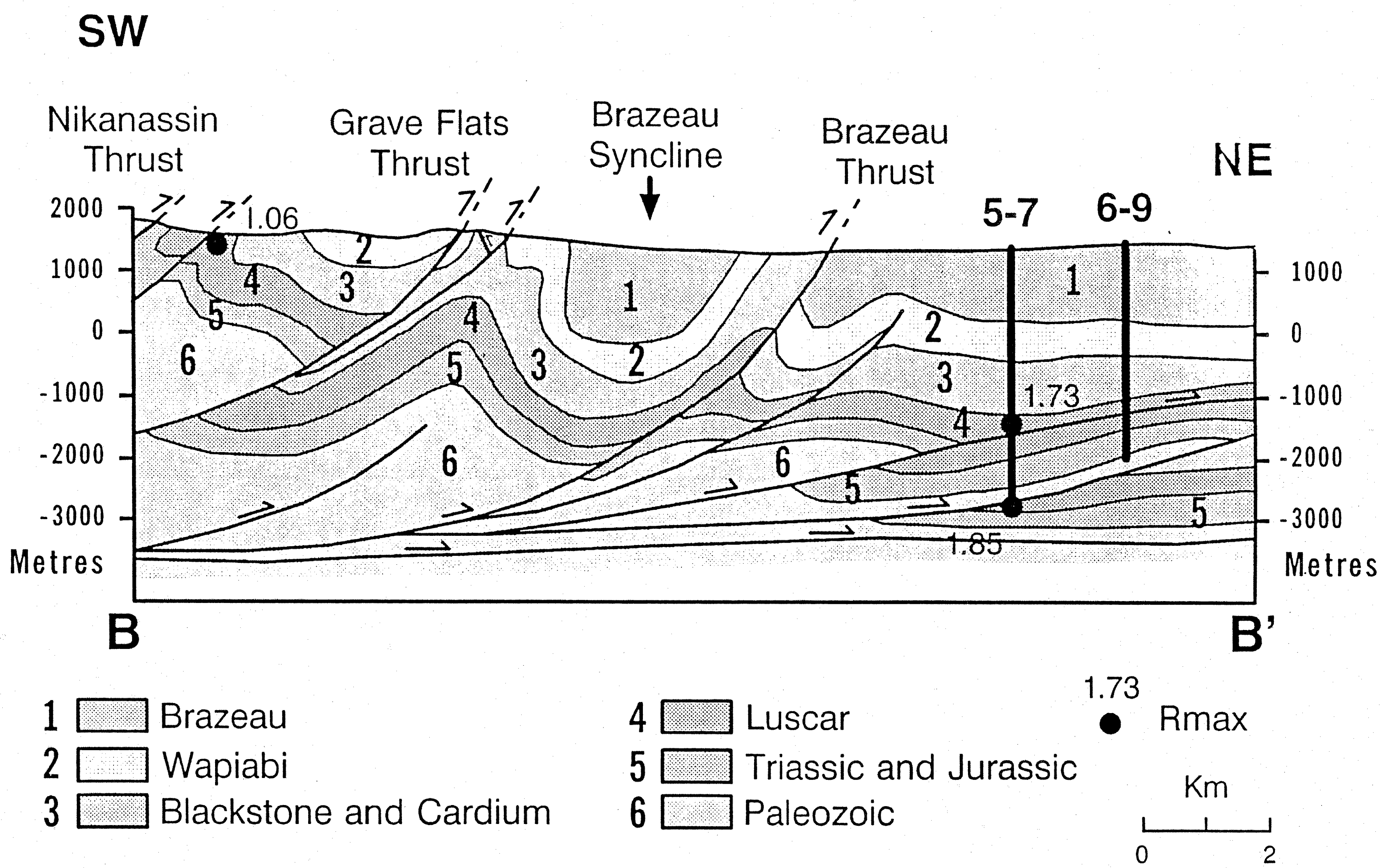


Figure 5. Cross section BB' through the southern half of the Cadomin mapsheet.

COAL RESOURCES

The economic coal seams of the Coalspur Formation are present in the area in three parallel bands in the Entrance Syncline and Coalspur Triangle Zone. The Mercoal band is the southernmost, contains the Mercoal Project of Manalta and dips about 30 degrees to the northeast. The Coalspur band is in the middle and contains the structurally thickened Coal Valley pod east of the map area and dips generally to the southwest. The Robb band is the northernmost band and contains northeast dipping strata and is less deformed than the Coalspur band.

The Lower Cretaceous coals of the Luscar Group are exploited in the Cadomin-Luscar Coal field in the Cardinal River Mine (annual production in 1989 of 3 million tonnes of raw coal) and Gregg River Mine (annual production in 1989 of 2.6 million tonnes of raw coal).

COAL-BED METHANE POTENTIAL

In the Entrance Syncline, the Coalspur coals are buried at various depths (up to 1 km), are of high-volatile bituminous rank and may form exploration targets for coal-bed methane.

The Jewel seam is the major economic coal seam of the Cadomin-Luscar Coalfield and is about 10 m thick. The rank of the Jewel seam ranges from high volatile A to low volatile bituminous (Langenberg et al., 1989). The cross section of figure 5 shows the Jewel seam at varying depth along the Cadomin Syncline, but generally in the range of 100 to 1000 meters deep in the area west of Cadomin. The general plunge of the fold axes is 10 degrees to the southeast, and the Jewel seam is probably at larger depths than 1000 meters in the eastern part of the Cadomin Syncline along the McLeod River. It can be concluded that, in the general area west of the town of Cadomin, there is an area of at least 20 km^2 underlain by the 10 meter thick Jewel seam of high- to medium-volatile bituminous rank. This includes areas that contain structurally thickened coal according to surface exposure to the west of this area. Consequently, this area contains a coal reservoir with a volume of 2×10^8 m^3 and (based on an average density of 1.2 for this type of coal) a mass of 2.4×10^8 tonnes (2.64×10^8 tons).

One can assume an average gas content for coals of this rank of 13 cm^3/g (Levine, 1990). Therefore, the small area west of Cadomin could contain 3.1×10^9 m^3 (about 100 BCF) of methane. This resource is comparable in size to well-known gas fields such as Findley, Basing, and Mountain (ERCB, 1989). The area further west contains additional coalbed methane resources, but a number of thrust faults disrupt the continuity of the reservoir.

AKNOWLEDGEMENTS

This research was funded partly by the Alberta Office of Coal Research and Technology. Coal mining companies active in the area are thanked for providing data and giving access to their properties.

REFERENCES

Charlesworth, H.A.K., Guidos, J., Gold, C and Wynne, D. (1989): TRIPOD 4.0, a microcomputer program for storing, retrieving,displaying and analysing orientation, stratigraphic and positional data from drill holes, outcrops and seismic lines; Computer Manual, University of Alberta, 111 pages.

ERCB (1989): Alberta's Reserves of crude oil, gas, natural gas liquids, and sulphur; Energy Resources Conservation Board, Report 90-18.

Hill, K.C. (1980): Structure and stratigraphy of coal-bearing strata near Cadomin, Alberta; Unpublished M.Sc. thesis, University of Alberta, 191 pages.

Jerzykiewicz, T. (1985): Stratigraphy of the Saunders Group in the central Alberta Foothills - a progress report; Geological Survey of Canada, Paper 85-1B, pp.247-258.

Jerzykiewicz, T. and Sweet, A.R. (1986): The Cretaceous-Tertiary boundary, central Alberta Foothills. I: Stratigraphy; Canadian Journal of Earth Sciences, v.23, pp.1356-1374.

Johnston, S.T. (1985): Structure of the Triangle Zone in the Rocky Mountain foothills near Coalspur, Alberta; Unpublished M.Sc. thesis, University of Alberta, 70 pages.

Langenberg, C.W. and McMechan, M.E. (1985): Lower Cretaceous Luscar Group (revised) of the northern and north central Foothills of Alberta; Bulletin of Canadian Petroleum Geology, v. 33, pp. 1-11.

Langenberg, C.W., Macdonald, D., Kalkreuth, W., and Strobl, R. (1989): Coal quality variation in the Cadomin-Luscar coalfield; Alberta Research Council, Earth Sciences Report 89-1, 65 pages.

Levine, J.R. (1990): Generation, storage and migration of natural gas in coal bed reservoirs; *in*: Coal geology and its application to coal-bed reservoirs - Lecture notes for short course; Alberta Research Council Information Series No. 109, p.84-130, (1990).

Macdonald, D.E., Langenberg, C.W. and Strobl, R.S. (1988): Cyclic marine sedimentation in the Lower Cretaceous Luscar Group and Spirit River Formation of the Alberta Foothills and Deep Basin. *In*: Sequences, Stratigraphy, Sedimentology: Surface and subsurface,D. P. James and D. A. Leckie (eds), Canadian Society of Petroleum Geologists, Memoir 15, pp. 143-154.

MacKay, B.R. 1929. Cadomin mapsheet; Geological Survey of Canada, Map 209A, Scale 1:63360.

Stott, D.F. (1963): The Cretaceous Alberta Group and equivalent rocks, Rocky Mountains Foothills, Alberta; Geological Survey of Canada, Memoir 317, 306 pages.

TITLE: Coal Compilation Project: Geoscience Information System (GSIS) Supported Coal Resource Mapping in Alberta

AUTHORS: R.J.H. Richardson, D.K. Chao and C.W. Langenberg[1]

D. Fietz[2]

AFFILIATION: [1]Alberta Geological Survey
Alberta Research Council
Box 8330, Station "F"
Edmonton, Alberta
T6H 5X2

[2]Elad Enterprises
Calgary, Alberta
T2K 3G7

ABSTRACT: The objective of the Coal Compilation Project (CPP) is to provide coal resource maps to stimulate and support industry exploration programs, and assist government in matters of resource management. An essential feature of the project is the use of cost-effective Geoscience Information System (GSIS) technology that allows the data base and various thematic maps to be analysed, updated and displayed with complete flexibility at any scale.

The CCP encompasses 16 1:50,000-scale mapsheets to be completed over a three-year period. Each map set is designed to be a stand-alone, unique product contributing to an overall synthesis of information. Maps are at a regional or reconnaissance level. Collection of new data is limited. Compiled and evaluated data are based on Alberta and federal government sources, unpublished corporate reports and data from the coal industry.

In 1989/90, the CPP focused on the Hinton-Grande Cache Corridor, located in west-central Alberta, and included four contiguous NTS

mapsheets. From southeast to northwest, they are: 83F/5, 83E/9, 83E/14 and 83E/15. For each mapsheet, a product has been generated that includes a coal resource map (scale 1:50,000), thematic "snapshot" maps (scale 1:250,000) and accompanying text. During this first year of the CPP, substantial time was spent in developing the hard copy product template and in designing the GIS template. Some minor field checking, sampling and analysis were done during each study to provide quality control and additional support for coal quality research.

During 1990/91, the second year of the project, the area north of Grande Cache (83L/2-3, 83L/7 and 83L/10) was the focus of investigation. An additional map (83E/16), east of Grande Cache in the Berland IRP, was completed. Six maps are receiving attention during 1991/92. Four maps (83F/4, 83C/9, 83C/15 and 83C/16) continue the coverage south from Hinton, and two maps are in the Crowsnest Pass area (83G/8 and 83G/9). The sixteenth map area will be selected if time and budget permit. The project will be reviewed in late-1991, and its future will be decided at that time.

Coal Quality Modelling Using Fuzzy Techniques

Shauna Treasure and Dennis Nikols
Alberta Geological Survey, Alberta Research Council

ABSTRACT

The heterogeneous nature of coal and the extensive coal resources of Alberta have made it desirable to develop a method for predicting coal quality and determining how representative is a single sample, when the coal zone or mine is known. The Coal Section of the Alberta Geological Survey has been examining coal quality distribution to derive a predictive model.

There is a large degree of uncertainty and imprecision in coal quality data, caused by the high spatial variability of coal quality parameters, the non-symmetrical distribution and an inadequate number of data points. This has led in this project to the development of a model using "fuzzy" mathematical techniques. Fuzziness is a type of imprecision that arises from arranging elements into classes that do not have sharply defined boundaries. The theory of fuzzy sets provides a technique for solving a problem that is too complex or too vague for analysis by standard techniques. Each coal quality parameter, such as sulphur, ash and calorific value, is represented as a fuzzy set with a degree of membership indicating the extent to which an individual sample relates to the entire population. The quality is determined by the extent to which sample parameters surpass specified minimum or maximum levels in each of the coal quality attributes. Development of the model involved the generation of semi-variograms from drill hole data and the formulation of equations

membership grades for the individual drill holes. Variograms were generated for seam 1 and were used to determine any trends along the drill hole section. These trends were used to confirm that the drill holes roughly followed the trend of the coal. Box plots of ash yield, sulphur value and calorific value for each drill hole were used to generate the model. A linear equation was established between adjacent drill holes to predict the coal quality in between with the solutions being given as ranges rather than discrete values. Providing a range of values as opposed to a single value made the predictive model more flexible.

BACKGROUND

Knowledge of the geology of the study area and background on the theory of fuzzy mathematics is required before a reasonable analysis of the area can occur. A very brief description of the stratigraphy and geology of Pit 3 is given. The theory of fuzzy mathematics is described in a great amount of literature, but only a brief overview is provided.

The Highvale mine is located in the Ardley Coal Zone which was deposited in large, sheltered swamps during the lower Tertiary in the Alberta Basin. The Ardley Coal Zone is situated in the upper part of the Scollard Member of the Paskapoo Formation. The rank of the coal at the Highvale mine is subbituminous "B". The depositional environment of the Paskapoo Formation in the area around the Highvale mine is thought to be an alluvial plain system with fluvial and lacustrine deposits. Seam 1 is the uppermost seam at the Highvale mine and is one of the primary coal seams for mining purposes and exhibits good coal quality and relatively consistent thickness through out Pit 3 (Nikols, 1990).

the representativeness of a coal sample with respect to a larger population and predict the values of a coal sample at a specified location within the population. The study was simplified to a manageable study area, namely seam 1 within Pit 3 at the Highvale Mine. To chose a larger area would require lengthy derivations of the equations because of the heterogeneity found in coal. By simplifying the equations, it was possible to test the possibility of using fuzzy math to represent and model coal quality. Work can expand once it has been determined that the results are useful and meaningful.

Determination of the membership function is an important step in using fuzzy set theory. Although this process is subjective in nature, there must be a clear basis for the choices. In constructing a set that defines a certain collection of objects, a feature of these objects, in this case ash yield, sulphur value and calorific value, should be used (Civanlar, 1986). The features are characterized by a histogram (figure 2). The data was separated into classes using the histograms and statistical results such as the median value and ranges. The skewness of the data in all cases required different widths for classes on each side of the histograms. Once the membership grades are defined, it is an easy task to determine the grade of a new sample. This method is a simple way of determining the representativeness of a sample of coal taken from an area where the coal quality is well understood. In areas where less is known about the population's coal quality, membership grades can be established, but may be less meaningful.

The development of a predictive model was not as simple. At first, the variability of the coal was examined. The variograms generated for seam

If a sample is obtained from seam 1 in Pit 3 with an ash yield of 11.27% , sulphur value of 0.26%, and a calorific value of 11086 btu/lb, the corresponding membership grades for that sample is 0.6 for ash, 0.9 for sulphur, and 0.5 for calorific value. The interpretation of these membership grades would be that the sample is very representative of the population with respect to its sulphur value, but only somewhat representative of the population with respect to its ash yield and calorific value, but still well within the range of the population. A sample from seam 1 in Pit 3 with an ash yield of 30.78% , sulphur value of 0.30%, and a calorific value of 8585 btu/lb, the corresponding membership grades for that sample is 0.3 for ash, 0.7 for sulphur, and 0.4 for calorific value. The interpretation of these membership grades would be that the sample is somewhat representative of the population with respect to its sulphur value, but hardly representative of the population with respect to its ash yield and calorific value.

The predictive model uses selected distances between adjacent drill holes to predict a likely range of coal quality values at that location. The range of values with a membership equal to or greater than 7 for each drill hole was calculated. Table 2 shows the results for ash yield. These ranges were used to develop the equations in table 3. By choosing a distance between two drill holes and plugging those values into the equation, a range of values is given that is equal or greater than a membership grade at that location. A prediction of coal quality 23 metres from drill hole 904 towards drill hole 913 would be given as a range between 10.6% to 22.4%. Predicting a smaller range would decrease the possibility that the predicted value would occur at that position. If a sample was then taken at that location and analysized, one could get an idea of how

903 to 907	values equal	y=18.75+0.0375x
907 to 905	y=10+0.009x	y=22.5+0.047x

CONCLUSIONS

Fuzzy math appears to have the capability of determining the representativeness of a coal sample and predicting coal quality. A simplified example was used to test the potential of this technique. The results of the analysis of seam 1 in Pit 3 at the Highvale mine have demonstrated that membership grades provide the flexibility to represent a coal sample with respect to the population. Knowing the distribution of coal quality within an area, allows for the determination of the coal quality of a sample relative to the population If the sample values are closer to the median of the population, the sample is more representative. The sample is more representative of the population, the closer it is to the median value of the population. The further it is from the median value, the more anomalous the sample is. Traditional methods do not allow for sufficient flexibility in the model for the heterogeneous coal. As new samples are available, the mathematical model can be revised and improved. Further prediction outside of the narrow corridor used in this study would require more complex math to generate equations. The next step will be to test the fuzzy technique described here with a larger coal quality population. Other seams, mining pits, and coal zones should be able to be treated in a similar manner. With fuzzy math, it has been determined that it is possible to assign a grade of membership to a coal sample, whether it is very, somewhat or hardly representative of a coal seam or deposit, but in a more quantitative manner.

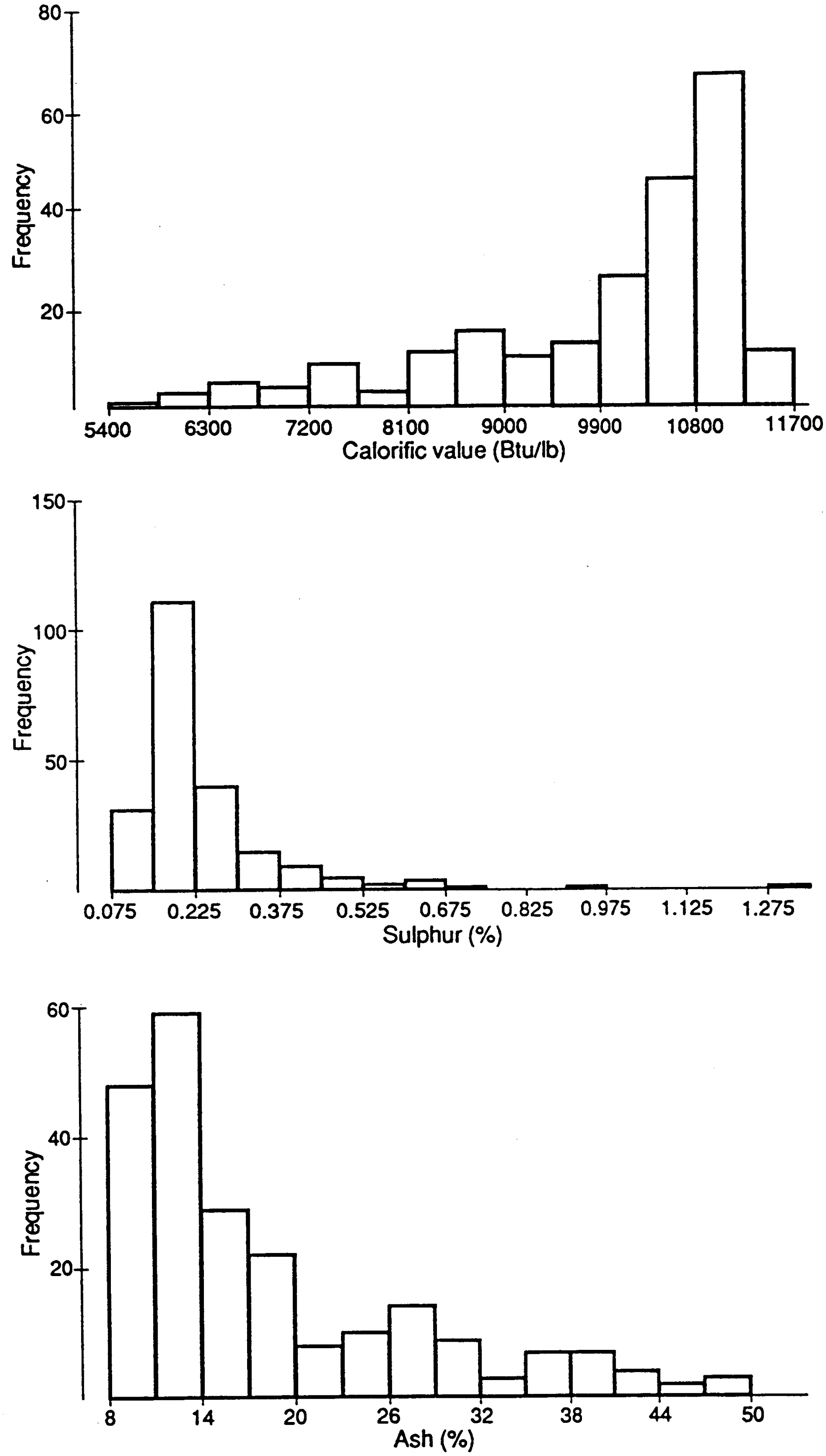


Figure 2. Frequency histograms of calorific value, sulphur and ash for seam 1, pit 3, at the Highvale Mine, Alberta.

Table 1. Membership grades for Seam 1, Pit 3, Highvale Mine.

Grade	Ash (%)	Sulphur (%)	Calorific Value (Btu/lb)
0.0	< 8.75, > 48.9	< 0.06, > 1.32	< 5565, > 11548
0.1	8.75-9.19,41.5-48.9	0.06-0.073,0.91-1.32	5565-6800,11450-11548
0.2	9.19-9.67,36.0-41.5	0.073-0.086,0.72-0.91	6800-7600,11350-11450
0.3	9.67-10.13,30.4-36.0	0.086-0.099,0.60-0.72	7600-8200,11250-11350
0.4	10.13-10.6,26.5-30.4	0.099-0.112,0.51-0.60	8200-8700,11150-11250
0.5	10.6-11.07,23.8-26.5	0.112-0.125,0.43-0.51	8700-9150,11050-11150
0.6	11.07-11.54,21.4-23.8	0.125-0.138,0.36-0.43	9150-9450,10950-11050
0.7	11.54-12.0,19.4-21.4	0.138-0.151,0.30-0.36	9450-9850,10850-10950
0.8	12.0-12.48,17.7-19.4	0.151-0.164,0.26-0.30	9850-10100,10750-10850
0.9	12.48-12.94,16.0-17.7	0.164-0.176,0.22-0.26	10100-10300,10650-10750
1.0	12.94-16.0	0.176-0.22	10300-10650

THICK SEAM MINING DEMONSTRATION PROJECT RESEARCH ASPECTS

1991 Alberta Coal Research Contractors Conference
Calgary, Alberta
October 1991

Authors:

JOHN KULACH, M.Sc., P.Eng.
Senior Mining Engineer

HORST H. SANDKUHLER
Manager, Underground Mines

SMOKY RIVER COAL LIMITED
P.O. Box 2000
Grande Cache, Alberta
T0E 0Y0

ABSTRACT

Smoky River Coal Limited and her Majesty the Queen, represented by the Federal Government of Canada and the Provincial Government of Alberta and its Minister of Energy, represented by the Alberta Office of Coal Research and Technology, have agreed upon a joint research and development program.

The program is to demonstrate the technical feasibility and commercial potential of an integrated coal mining system employing innovative technological developments in the fields of strata supporting techniques, coal extraction and testing the applicability of continuous coal transport for thick (up to 6 m or 20 ft) coal seams.

The program includes the demonstration of roof and side support machinery, mining machinery capable of extracting coal to a height of 6 m (20 ft) in a single vertical pass and a trial of continuous coal haulage by a flexible conveyor system, to enhance the efficiency and safety of coal extraction.

The technological aspects of the project and the seam deposition have raised a number of research questions related to strata behaviour which have to be answered. The program includes an extensive strata control research project. In more than two years, the strata monitoring project has produced a large information database related to roof, sides and pillar behaviour in normal as well as increased excavation heights at various stages of mining activity. The obtained results are extremely valuable for future mine development, design, planning and the safe extraction of thick seams by underground mining techniques.

A substantial part of the program has been successfully concluded. However, the continuous haulage system has experienced considerable operational problems during its testing period. Further manufacturer research and design work was necessary and is ongoing.

In spite of the fact that the program has not been fully successful, the range of results, information and experience already gained is of exceptional value.

This paper describes the present Smoky River Coal Limited Demonstration Project. The project is to demonstrate the technical feasibility and commercial potential of an integrated coal mining system employing innovative technological developments in the fields of continuous coal transport, extracting and strata supporting machinery incorporating new safety features for thick (up to 20 ft or 6 m) coal seams.

The project includes demonstration of continuous coal haulage by a mobile conveyor system, roof and side support machinery and mining machinery capable of extracting coal to a height of 20 ft (6 m) in a single vertical pass.

The technological aspects and the seam deposition have raised a number of research questions related to strata behaviour which have to be answered. The project is combined with an extensive strata control research program. In over two years, the strata monitoring program has produced much information related to roof, sides and pillar behaviour in normal (10 ft or 3 m) as well as increased excavation heights at various stages of mining activity. The results are encouraging and are of exceptional value for future underground mine developments.

A substantial part of the project has been successfully concluded. However, the testing of the continuous coal haulage system has experienced considerable difficulties. Further manufacturer's research and design work is required in order to continue with the haulage system concept.

In spite of this, the range of results, experience and information already obtained is of exceptional value. Some of the initial observations and results are already utilized in the regular Smoky River Coal mining practice. Once the monitoring program is concluded, a complex analysis of results will be done. These results may have an application in any underground mining operation.

The presentation will describe various engineering and operating activities that are encompassed by the project.

The project has been approved by the Federal and Provincial governments for realization within the Western Diversification Program.

1.0 THE THICK SEAM UNDERGROUND MINING PROJECT - DEMONSTRATION ASPECTS

To improve operational safety, resource recovery, productivity and cost effectiveness in underground mining conditions, particularly in the thicker, tectonically disturbed coal seam prevalent in the Foothills regions of Western Canada, alternative mining methods to those conventionally employed must be sought.

Considerable reserves of high quality coal are present in seams over 12 ft (3.7 m) thick in Western Canada. Room and pillar mechanized mining systems currently employed in underground mining are limited in their ability to mine these thick seams safely and efficiently. This equipment has an effective reach of 11.5 ft (3.5 m); extraction beyond this height requires non-routine methods and improvisation. These limitations reduce the proportion of the resource base that is economically recoverable.

Recent technological advances in roof and side support, coal transportation and coal cutting machinery have indicated that higher roadway development and extraction of thicker seams is feasible. Successful introduction of this technology could improve safety, resource recovery and the competitive position of Western Canada in export coal markets.

Equipment design, support methods and physical limitations have dictated that present development of the mine be in the upper 10 ft (3.0 m) of the 20 ft (6 m) thick coal seam. Floor coal is recovered during the depillar phase; the amount of recovery is determined by prevailing conditions. No consistently successful method for the systematic recovery of all of this floor coal can be devised using present methods. This results in large quantities of resource being lost.

Developments in the fields of machinery design and mining systems indicate that it is technically feasible to drive roadways up to 15+ ft (4.6+ m) high and to subsequently extract the pillars of coal. **The important factor is that the pillars could be extracted faster than at present**. Much higher resource recovery and substantial improvements in productivity are indicated. The proposed project, if successfully concluded and the technology applied to mines with similar seams and conditions to those at Smoky River Coal Limited (SRCL), indicates that resource recovery would increase by approximately 15% and production costs would be similar to low ratio truck/shovel or some dragline surface mining operations.

Successful demonstration can result in a stronger position for Western Canadian coal in export markets and can also help to make the low sulphur coals competitive with the high sulphur coals of the United States.

The project is considered to be a research and development experiment.

1.1 Roof and Side Support Machine

A roof and side support machine has been designed for use in both roof and side drilling in roadways up to 20 ft (6 m) high. This machine is electro-hydraulic, track mounted with dual booms and automated temporary roof support.

The essential features which make the machine capable of drilling roofs up to 20 ft (6 m) high include:

- A roof orientated drilling position. The drill mast of the existing machinery needed to be on the floor to enable the thrust of the roof drill to penetrate the roof rocks. This also limits the comfortable reach of the operators to erect mesh and insert bolts into the drill holes. The new roof and side drilling machine incorporates hydraulic locks to provide a thrust anchor so that the drill mast is now at a fixed distance from the roof. Elevating operators' platforms are also incorporated which allow the operators to remain at a comfortable working position in relation to the drill masts.

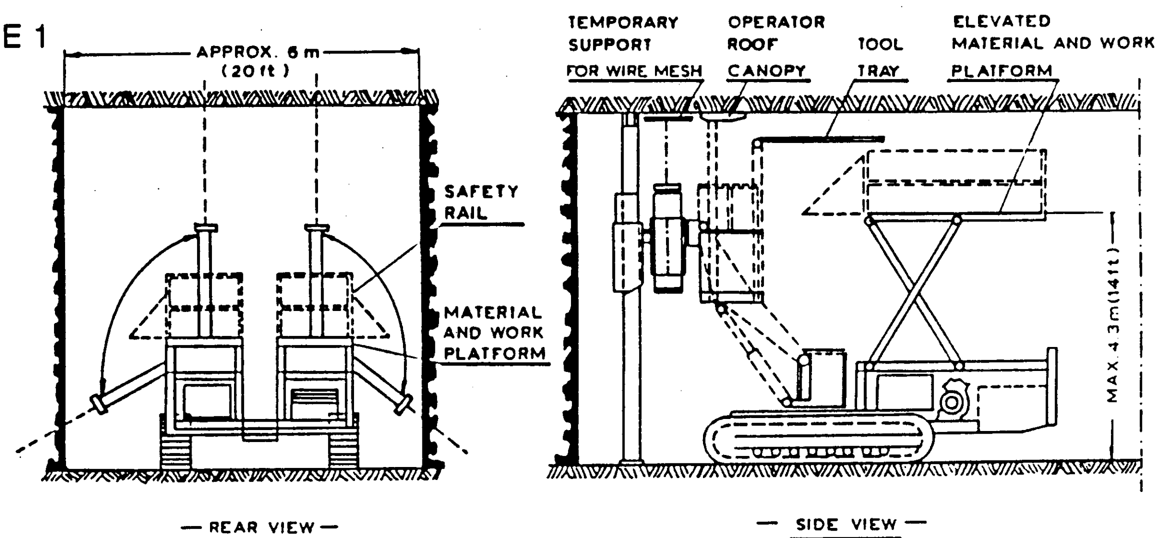
- Automated temporary roof support beams have been designed to include mesh handling facilities and will eliminate the need for the roof bolter operators to work under unsupported roof.

- Elevating materials and tool platforms are included to allow easy operator access for thick seam working.

- All platforms are designed with safety fencing and non-slip floors.

- The machine is designed on a "walk-through" principle such that operators can pass from the back to the front of the machine without being exposed to unsupported ribs. (Figure 1 illustrates these factors.)

FIGURE 1



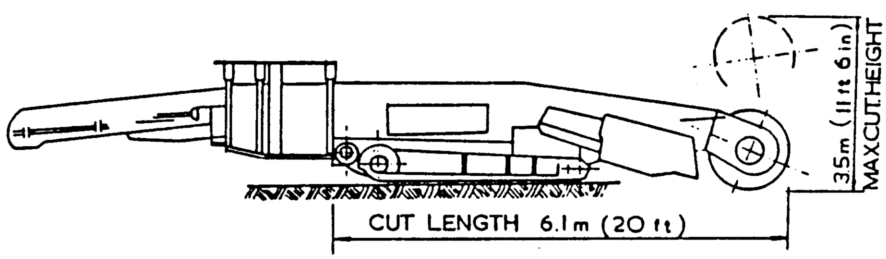
FLETCHER MODEL HDDR DUAL HEAD ROOF AND SIDE BOLTER

1.2 Thick Seam Continuous Miner Machine

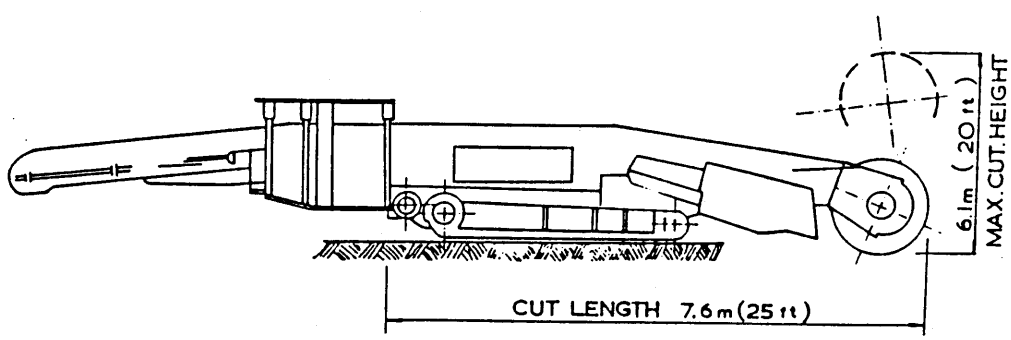
Continuous miner cutting and loading machines presently being used in SRCL's underground operations are capable of operating to a maximum height of 11.5 ft (3.5 m). The thick seam machine would have the same basic functions as those being presently used. However, heavier duty structures, bearings, power trains, tramming mechanisms and cutting head booms would be part of the design. These modifications would result in the operator's cab being 25 ft (7.6 m) from the cutting heads, and the maximum operating height would be 20.0 ft (6 m).

Figure 2 illustrates current equipment and the high reach continuous miner with significant measurements highlighted.

FIGURE 2



STANDARD CONTINUOUS MINER

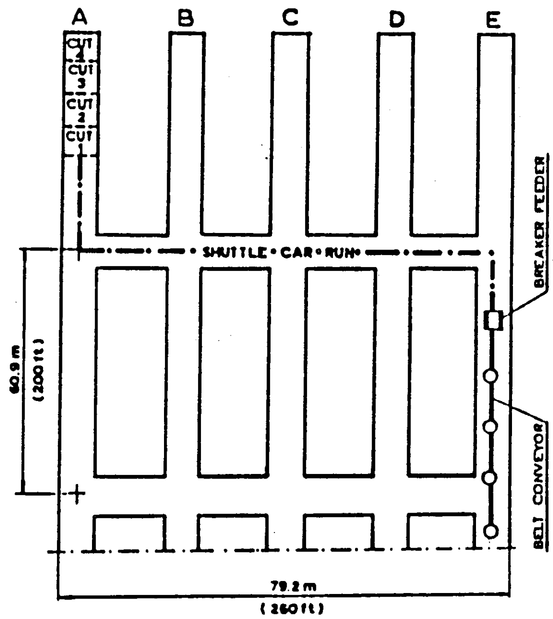


HIGH REACH CONTINUOUS MINER

1.3 Mobile Conveyor Coal Transportation System

In the current operation, coal is transported from the continuous miner working at the coal face by electrically-propelled shuttle cars to the breaker/feeder. The breaker/feeder loads the coal onto the conveyor belt system which transports the coal to the surface raw coal stockpile area. The capacity of the existing shuttle cars is approximately 8 tons. The general arrangement is illustrated in Figure 3.

FIGURE 3



MINING DISTRICT — GENERAL ARRANGEMENT

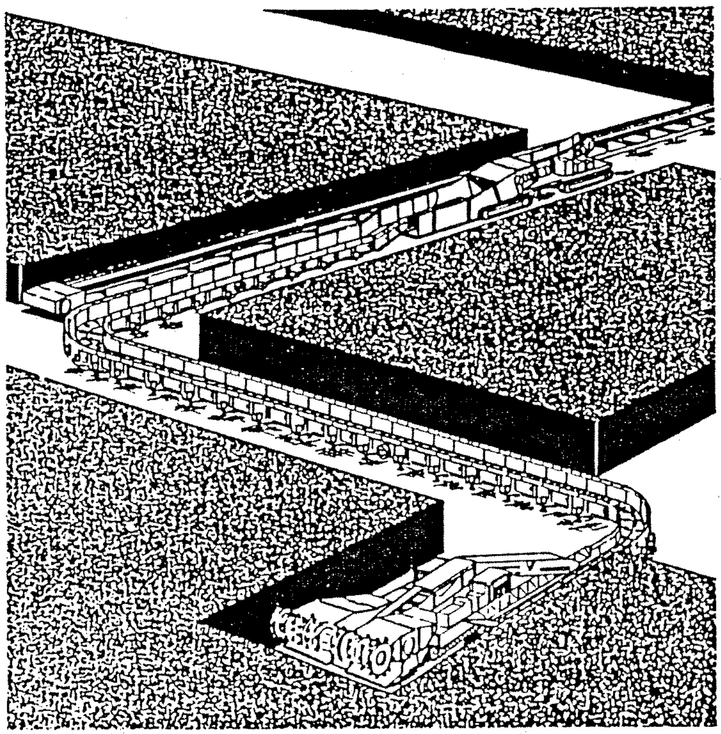
During development, as the continuous miner systematically cuts 20 ft (6 m) forward in each roadway, the distance from the coal face to the breaker/feeder gets progressively longer and hence, the time taken by the shuttle car to travel to the breaker/feeder and back takes longer with each cut. While taking this 20 ft (6 m) cut, the continuous miner has frequent stops waiting for the shuttle car to return. From studies taken of our present operations, the average time spent by the continuous miner waiting for the shuttle car during development is 88 minutes of each production shift, or 21% of the available time.

It has been calculated that if, during development, the higher seam (15+ ft or 4.6+ m) is extracted, then increased production would mean that the shuttle car would make more trips to the breaker/feeder and the continuous miner average waiting time would increase to 123 minutes of each production shift, or 29% of the available time.

A study taken from our present operation shows the average time spent by the continuous miner waiting on the shuttle car during depillaring is 125 minutes, or 29.75% of each production shift. If during depillaring, the full height of the coal seam (20 ft) was extracted, it is calculated that the continuous miner waiting time would increase to 135 minutes or 32.15% of each production shift.

The Demonstration Project would replace shuttle cars and breaker/feeders with a continuous flexible conveyor system. See Figure 4.

FIGURE 4



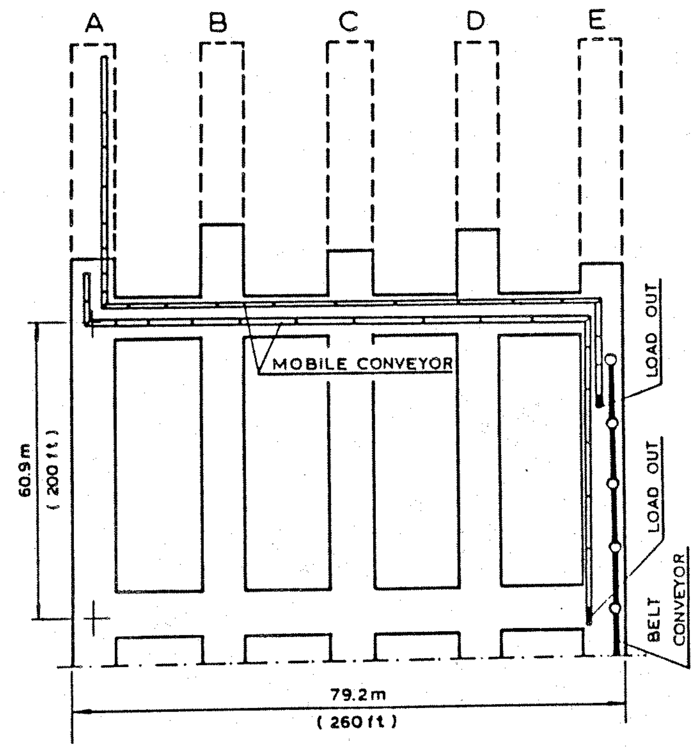
MOBILE CONVEYOR
CONTINUOUS HAULAGE

This system, a "Kloeckner-Becorit" Mobile Conveyor (MC), has the potential to provide continual coal clearance from the continuous miner at the coal face to the outbye conveyors. It is anticipated that a substantial reduction in the non-productive waiting time of the continuous miner would result.

The MC is 600 ft long which is compatible with the mine design (Figure 5). The machinery can negotiate several 90° turns in any single operating position. It can convey up to 12 tons per minute.

It is estimated that production could increase by approximately 30% compared to shuttle car haulage.

FIGURE 5

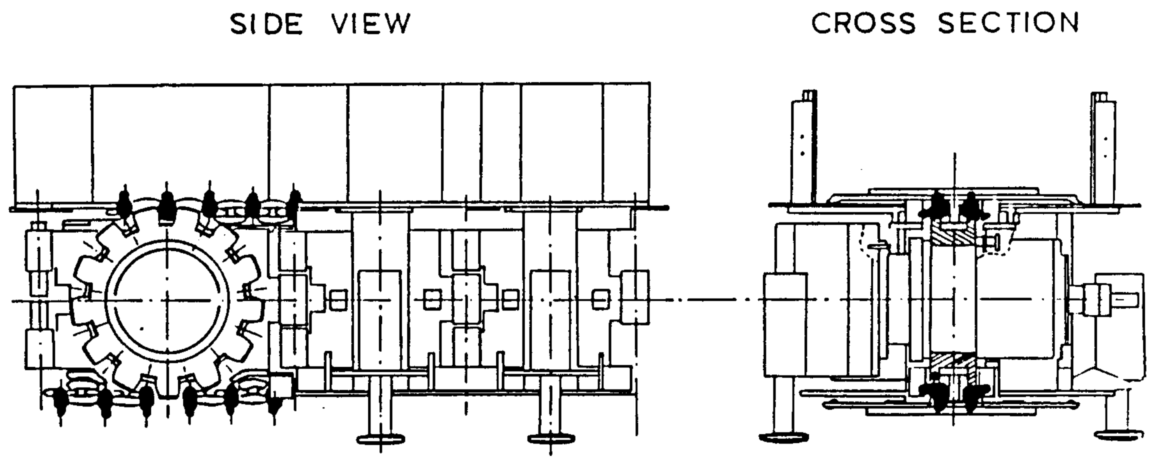


MOBILE CONVEYOR ARRANGEMENT

The MC, based on the application of intermediate drives for armoured face conveyors, has been developed by Kloeckner-Becorit, and basically differs in the following four characteristics from any previous systems.

- The conveyor is mobile. For tramming, the conveyor drives are used, the bottom strand of the chain being employed as a track. (See Figure 6.)
- Tramming on a required course even with tight curves can be achieved by controlling the direction at the loading end of the conveyor. The following pans will follow this once-laid track exactly.
- When the conveyor is in its conveying position with the bottom chain raised, conveying takes place along the course built up during tramming.
- With the assistance of the elevating and steering cylinders, the ends of the conveyor can be raised, lowered and moved sideways.

FIGURE 6



SELF PROPELLED CONVEYOR WITH INTERMEDIATE DRIVES

The basic design of the driving system consisting of a chain strand, chain guide and drive units corresponds to that of a standard belt conveyor with intermediate drives. An important part is the specially designed chain, where the flat links are equipped with a spigot at each side, which engages into the drive sprockets and two guide sections. The chain is covered by flexible panel pans, which are mounted on the flat links and within the frame of the whole system. It fulfils the following tasks:

- protective haulage of the material with relatively well-defined conditions of friction;
- sealing of the chain guards against intrusion of coal fines.

The conveyor itself consists of three main elements:

- Intermediate drives which are distributed over the length of the conveyor. Each drive unit is driven by a 25 kW flame-proof electric motor and pneumatically operated two ratio planetary gear boxes which provide a conveying speed of 165.3 ft/min (0.84 m/sec) and a tramming speed of 40.5 ft/m (0.206 m/sec).

- Standard pans forming an inner curve radius of 13.8 ft (4.2 m) horizontally and 49 ft (15 m) vertically.
- End pans which are short return ends directly connected to standard pans.

2.0 RESEARCH ASPECTS - TECHNICAL EVALUATION

2.1 Introduction

The No. 9G-4 Mine deposit is located deeper than any other seam mined to date in SRCL's history. As mining operations go gradually deeper, higher rock-mass pressures will be expected. For future extraction design, theoretical and to some degree, practical, reliable prediction methods need to be developed.

In SRCL's past, the accepted strata control practice has been the method elaborated by Canadian Mining Research Centre (CMRC). Guidelines recommended by CMRC had fully satisfied our mining requirements up to approximately 500 to 600 ft of depth. Practical observations coincided with theoretical considerations.

Going gradually deeper, some divergences between calculations and practical results became apparent. Pillars seemed to be actually stronger than calculations suggested. Theoretical considerations indicated the pillar width should increase with depth but operational factors, as well as the minimization of coal sterilization, necessitated pillars no more than 40 ft wide (split pillars). Hence, investigation of some of the strata control approaches became necessary.

Systematic data collection on rock-mass, pillars, roadways, and gob behaviour during consecutive years and stages of mining activity, together with geological-tectonical analysis of extracted areas within No. 9B and No. 9G Mines became an integral part of efficient day-to-day production planning.

Future considerations called for the identification of more appropriate guidelines, useful for satisfactory systematic production design of further mining areas deeper than 700 ft. Consequently, SRCL developed its own strata control criteria at the beginning of 1990. This approach is, in fact, a combination of various pillar design methods and SRCL's past underground mining experience. Since then, the developed method is used in mine design, production planning and for operational decisions.

However, it is commonly known that strata control, particularly the determination of phenomena taking place inside the stressed rock-mass in a mathematical relationship form, is an extremely delicate and difficult task. Too many unknown interdependences have to be considered. Therefore, any rock-mass behaviour quantification must contain a certain degree of confidence. The better the deposit's characteristics are known, always leads to a higher degree of confidence. Therefore, this is a very crucial area of mining knowledge. An adequate pillar designing approach is essential for the economy of the operation as well as for safety purposes. Therefore, one of the research aspects of the project is to determine a degree of confidence for the existing pillar design criteria, which in fact, has never been tested before. The approach is still relatively new.

The essential requirement for successful application of the project is compatibility between mining equipment, methods and systems and the prevailing conditions and environment of the working places. Accordingly, it is necessary to acquire, analyze and interpret much site-specific data.

Data derived from the Demonstration Project, as well as from the regular mining districts can be analyzed and interpreted to provide the basis for mine planning. Before undertaking such a substantially different method of mining, a number of possible effects must be addressed before the system can be implemented.

A brief review of the most commonly applied strata control approaches used for North American deposits is presented in this section. This provides a view of the scope which had to be considered before starting the Strata Control Monitoring Program. This program is an integral part of the project.

2.2 Pillar Design Methods - Brief Evaluation

Recent developments in equipment (roof bolters, cutting/loading machines) create an opportunity to drive roadways with an increased height. This presents a challenge for the development of district panel roadways. If this can be achieved, it would be a step towards increased productivity. Various problems must be resolved to put into practice the proposed technology. Pillar behaviour for 20 ft (6 m) extraction (although roadway heights would be closer to 15 ft or 4.5 m) is one of the major considerations.

Theoretical considerations, combined with extensive practical experience acquired in the past 18 years while mining in No. 4 Seam allow a substantial degree of confidence in the design developed.

A number of research approaches are available for estimating pillar strength and "pillar load", or the average pillar stress.

Some commonly used methods of estimating pillar strength in North American practice include:

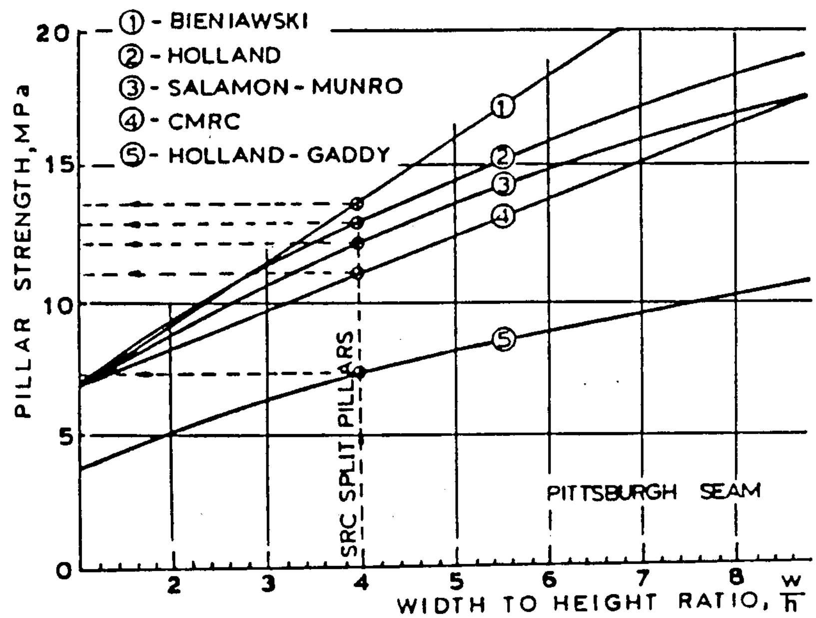
-Canadian Mining Research Centre (CMRC) Method
-Holland-Gaddy (1964) and modified Holland (1973) Method
-Salomon-Munro Method (1967)
-In-situ Methods (Bieniawski and Penn State University).

Each method gives different calculation results. Practical experience indicates some of them are more appropriate for low pillar width/height ratios and some are more adequate for high ratios. It is essential from the economic and safety points of view to select a practical method for No. 4 Seam, SRCL deposit, as well as other Western Canadian coal seams.

2.3 Comparison of Pillar Strength Formulae and Selection of Method for SRCL's Purposes

From all of the available pillar strength formulae, five expressions are commonly used. In Figure 8, the selected formulae are plotted (Rock Mechanics Seminar, Penn State University, USA, 1986).

FIGURE 8



STRENGTH VERSUS WIDTH TO HEIGHT RATIO
(PENN STATE UNIVERSITY)

It is apparent from this figure that for higher width to height ratios, the Holland-Gaddy formula predicts the lowest strength, while Bieniawski's formula predicts the highest strength. At the same time, the gap between these predictions becomes greater for high ratios. Even for the smallest pillars (split pillars with ratio 4) practised at SRCL, the Holland-Gaddy formula value is almost half the calculation using Bieniawski's method. The economic impact on an operation derived from these is substantial and cannot be neglected. This means that the Holland formula is such that it becomes very conservative at large width to height ratios. There is also a noticeable difference between CMRC and Bieniawski's predictions.

Numerous tests seem to prove the fact that **for high width to height ratios, there is a very rapid strength increase. The results are rather more consistent with Bieniawski's formula than with the other approaches. In fact, for width to height ratios greater than 10, pillars are indestructible and no theory indicates this.**

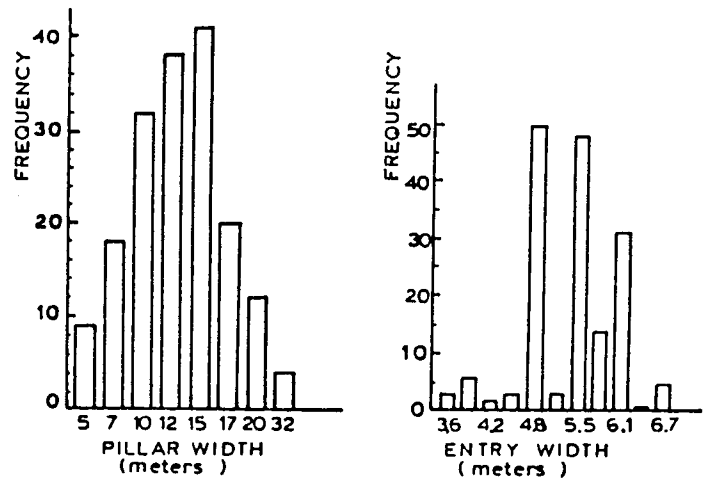
A detailed study of this aspect conducted at Penn State University has revealed that **the strength of coal pillars is considerably higher than even that predicted by Bieniawski's equation**. Thus, at some stage of the research, it was proposed that an exponent could be added to the equation, thus incorporating a higher rate of strength increase with increasing width to height ratios. However, for the sake of safety and due to lack of experimental data, it was decided not to pursue this concept.

For improved design of coal pillars in the United States, Bieniawski's formula is recommended.

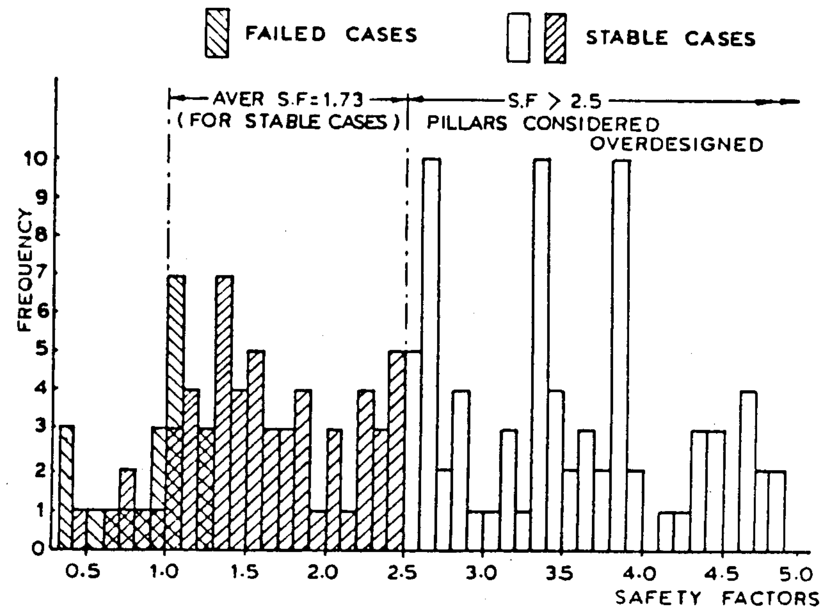
A national survey of coal pillars and roof dimensions in the United States was conducted to identify factors of safety for room and pillar design applicable to the United States coal fields. This safety factor provides a margin of pillar strength to account for uncertainties in the design process and the variability of geologic conditions and structural dimensions encountered in the mining area. The aim was to collect enough field data to rationalize pillar design, incorporating improved procedures and appropriate safety factors for economical, yet safe mining.

In total, 174 cases were available for stability analyses of coal pillars, plus 58 cases of roof falls for analyzing roof spans. The survey included a comprehensive study of such parameters as the depth below surface, seam thickness, roof spans, pillar height, pillar width, pillar length, width to height ratios, percentage of extraction and the design method. Some results are depicted in Figure 9.

FIGURE 9



SMOKY RIVER COAL LTD-SPLIT PILLARS 12.2m WIDE
-ROADWAYS 6.1m WIDE



PENN STATE UNIVERSITY SURVEY OF ROOM AND PILLAR DATA
IN US COAL MINING

It must be emphasized, however, that any calculations should be used as a guide only and local mining conditions and past experience must be taken into consideration for long-term and short-term planning.

Some observations derived from this study indicate:

- The majority of surveyed pillars are 30 to 45 ft (10 to 15 m) wide which is very similar to SRCL's and Western Canadian practices.

- Most surveyed entries are 14.5 to 19 ft (4.8 to 6.1 m) wide; the commonly applied entry width at SRCL is 20 ft.

- No pillar failure case has been recorded for safety factors above 1.3. In fact, some pillars designed with safety factors even below 1.0 have not failed.

- Pillars designed with safety factors above 2.5 are considered to be overdesigned.

It must be emphasized that at least some US coal fields (western fields) could be considered to be very similar to the Western Canadian deposits. Therefore, such a wide survey gives us an important indication and seems to be justified for application in the Western Canadian pillar design approach.

2.4 Brief Design Conclusions

The size of this paper does not permit a full explanation of studies done with respect to strata control at SRCL. Nevertheless, a brief review of some important pre-Demonstration considerations are useful before final design criteria recommendations.

- Calculated pillar strength values (square pillars) are up to 40% lower than actual pillar strength.

- Numerous tests indicate calculated pillar stress (the value corresponding to a pillar load) values are overestimating the actual pillar stresses by approximately 40%.

- SRCL's coal deposit (No. 4 Seam) seems to carry very similar geotechnical characteristics (cube specimen compressive strength is very similar) to US coal (Pennsylvania deposit). SRCL's No. 4 Seam indicates approximately 1,000 psi compressive strength of the cube specimen while Pennsylvania's coal indicates the strength of approximately 930 psi (Rock Mechanics Design in Mining and Tunnelling by Z.T. Bieniawski, A.A. Balkema, Rotterdam, 1984).

Also, extensive practical experience data gained in SRCL's operations have been compared with conclusions resulting from theoretical considerations.

2.5 Tentative Design Criteria

Based on preceding observations, calculations and analysis, the following design criteria have been developed for SRCL's future mining practices. The criteria are formulated in a general form, however, from a practical point of view, they pertain to mining activity under more than 700 ft (210 m) of overburden.

Figure 10-A illustrates terminology used for particular types of pillars.

Figure 10 shows a Pillar Design Criteria graph for pillars in No. 4 Seam, SRCL, for 20 ft wide entries and 15 ft high pillars. Similar graphs for 10 ft and 12 ft high roadways have been elaborated in SRCL. These graphs allow a quick pillar design criteria estimation in a described condition. For example, a 100 ft wide and 15 ft high pillar with a 1,000 ft thick cover meets requirements for split, panel and between district pillars, and is not expected to perform duties for entry or barrier pillars.

The strength pillar criteria practised at SRCL are as follows:

Barrier Pillars - It is tentatively suggested that their tolerable safety factor should be 4.0 or better. Recommended safety factor is 5.0.

Entry Pillars - It is tentatively suggested that their tolerable safety factor should be 3.0 or better.

Panel Pillars - It is tentatively suggested that their tolerable safety factor should be 1.3 or better. It is preferably recommended to stay within a safety factor zone between 1.5 and 2.0.

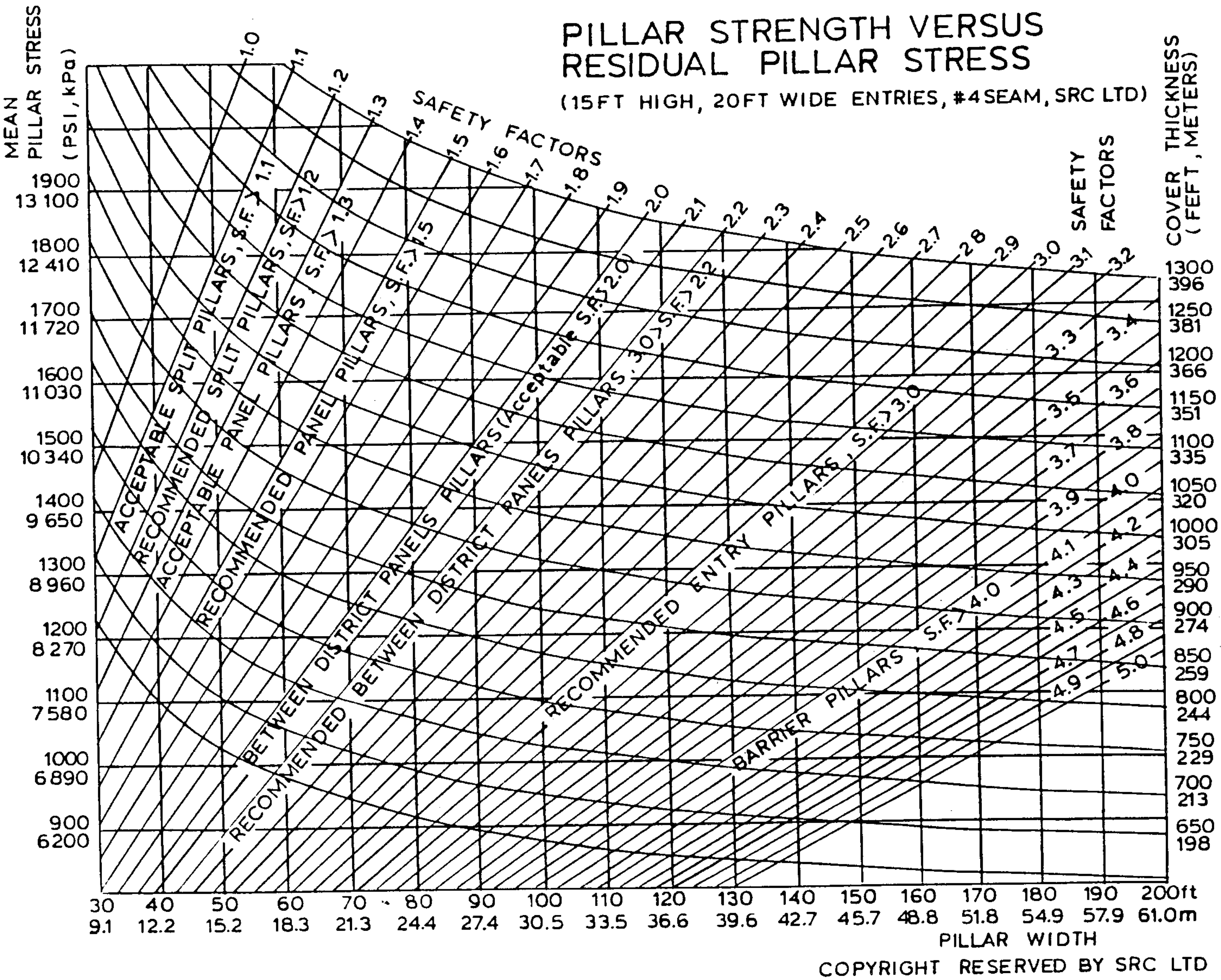
Between Panel Pillars - It is tentatively suggested that their tolerable safety factor should be 2.2 or better (recommended no greater than 3.0). A safety factor less than 2.2 will not promote eventual pillar failure, nevertheless, it may cause maintenance problems (rib stability) adjacent to the extracted district road; for example, belt road maintenance.

The operation schedule has to ensure that successive district development is completed before the preceding district is extracted. In other words, the lifetime of the district should not be unnecessarily extended.

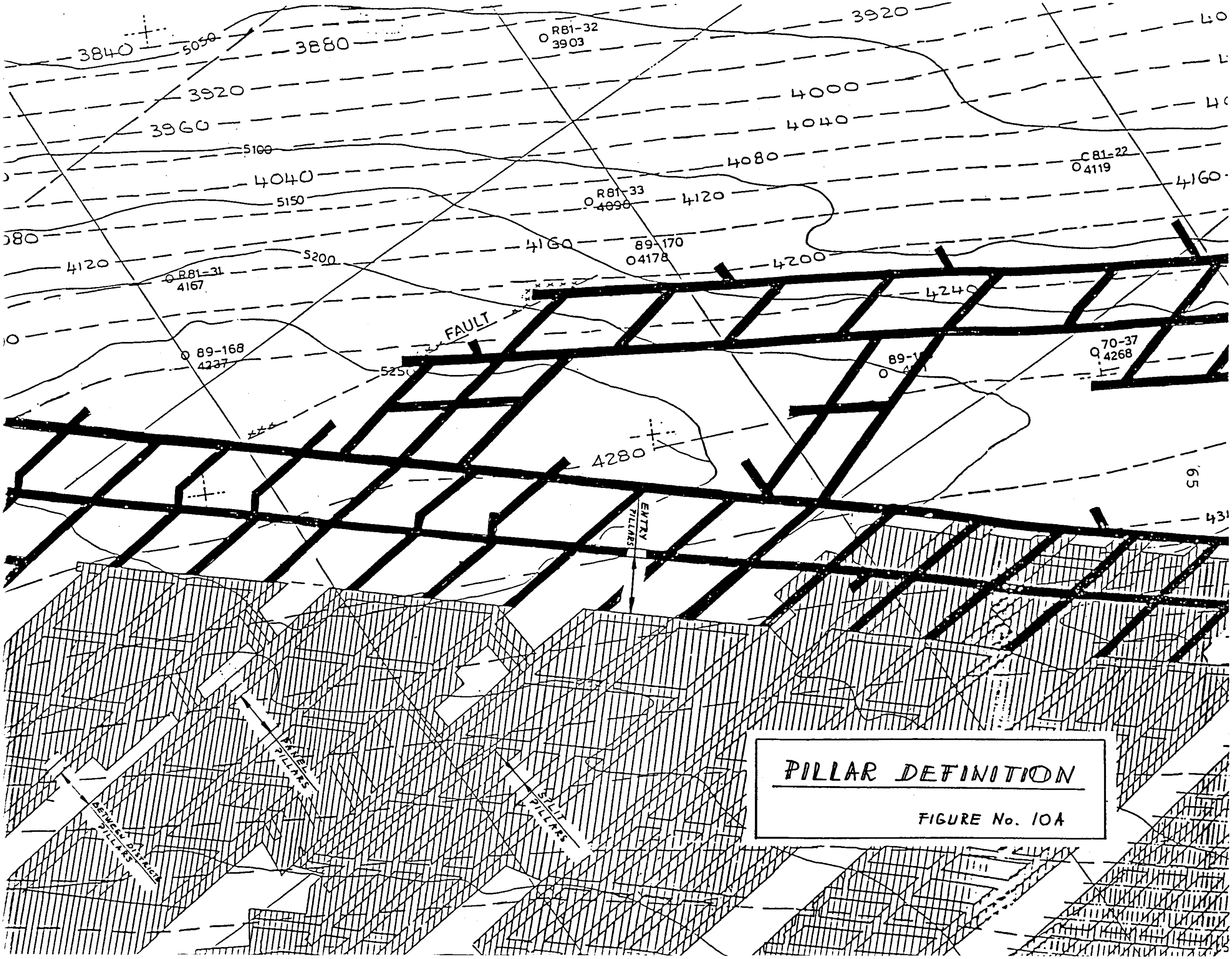
Also, safety factors may be closer to their lower limits for panels with a relatively short lifetime.

Split Pillars - It is tentatively suggested that their tolerable safety factor should also be 1.2 or better. (In SRCL's mining practice, a split pillar's lifetime may sometimes be close to one month.) However, for short-term (one week or so) operation, a 1.1 safety factor may also be acceptable.

FIGURE 10



PILLAR DESIGN CRITERIA



PILLAR DEFINITION

FIGURE No. 10A

It is emphasized that these are tentative suggestions aimed to form a further basis of comparison with actual mine experience in particular roof and coal seam conditions.

Estimation of design criteria reliability is one of the reasons why the Demonstration Project has been initiated.

The Test Area Layout is shown on Figure 11.

Gob Edge Pillars - The gob caving characteristics, condition of pillar edges, stress in residual pillars and conditions of roof and sides outbye of the depillaring zone are continually monitored and reported during the Demonstration. Daily foremen's shift reports are collected to allow full documentation of the mining activity.

When coal is extracted from the pillars during the depillar sequence, the support offered to the roof of the coal seam by the coal pillars is systematically removed. Subsequently, when the weight of the unsupported roof becomes greater than the inherent strength of the roof rock, this rock collapses (caves).

The remaining pillar edges form a new support line for the intact roof beds. The roof and caved rock beyond the last pillar edges is known as the "gob" and the support line formed by the remaining pillar edges is known as the "gob edge".

Full seam extraction would enable the floor of the workings to be kept relatively uniform without the need for "ramping down" to recover floor coal. This would improve the depillaring activity by extracting the coal in a single vertical cutting pass. This would shorten the extraction time in a particular place (speed up extraction) which would improve gob edge rock-mass stability as well as economical effectiveness of the operation.

Behaviour of gob, pillars and roadways is well understood in the current mining practice at SRCL where up to 20 ft (6 m) thick coal seams are extracted by multi-lift methods during depillaring (floor coal recovery). However, recovery of floor coal cannot be considered to be systematic; the rate of recovery being dependent on condition and operational opinion. Therefore, the next purpose of this Demonstration is to more systematically extract all of the coal thickness by using a more engineered system. It is essential that any differences in strata behaviour be identified as quickly as possible to build an understanding of observed phenomena. To build this database, numerous tests, observations and instrumentation monitoring are being carried out.

Conclusion

The whole strata control monitoring program should give a substantial comparative database for better understanding of mountainous coal deposits, more accurate pillar design, higher confidence in already developed designing criteria by SRCL and therefore, more effective planning for future underground mining activity.

3.0 MINE DESIGN

In determining the layout and sequence of mining in the proposed test area of the mine, due regard must be paid to the expected geological features, drainage requirements, methane emission rates and anticipated strata behaviour.

3.1 Test Area Geology

Within the test area (Figure 11), No. 4 Seam dips to the north at between 9° and 18° (average of 15°) and strikes in a general east-west direction. No. 4 Seam maintains a true thickness of 20 to 21 ft (6 to 6.4 m) throughout the area.

Roof rock consists of 3 to 5 ft (0.9 to 1.5 m) of shale overlain by the hard Super 4 Sandstone unit. Three to 10 ft (0.9 to 3.5 m) of hard sandy siltstone forms the floor material.

Mineable reserves within the test area (Figure 11) were calculated at 1.5 million long tons raw. Allowance has been made for 85% coal extraction factor.

3.2 Test Area Layout

Consideration of the geological evaluation of the proposed test area indicates that the depth of overburden thickness increases steadily in a northerly direction from 550 ft (167 m) adjacent to the main entry pillars of the mine to a maximum of 880 ft (268 m) adjacent to the Taylor syncline at the north of the proposed area.

In order to allow any strata water to gravitate to the lowest working area from where it can be pumped out of the mine, access entry roads were driven into the area close to the Taylor syncline with the mining sections being developed up gradient to the south of these main entries.

Substantial barrier pillars will be left to protect the western main entries which are required to remain stable for the life of the mine.

Consideration of pillar size data (Figure 11) supports the following planning parameters:

- Pillars of 130 ft (39 m) wide are being applied to the 1 East corridor and the depillaring limit in the test area. This gives a calculated acceptable safety factor of 3.15.

- The barrier pillar between 1 West main entries and the test area is 200 ft (61 m). It represents a safety factor of more than 5.0. This is definitely satisfactory protection for this roadway.

- It was planned to drive SE 1 district development roadways at the height of 10 to 12 ft (3 to 3.7 m). If assuming 12 ft (3.7 m) high roadways, the safety factors for pillars 40 ft (12 m) wide fluctuate from 1.52 to 1.79. This means that the 40 ft (12 m) wide pillars are acceptable as the panel pillars and can be developed right from the very beginning of the development phase. However, since it was the first district in this area, it was justified to consider splitting the first two blocks right before depillaring.

It was intended to increase the district development roadway heights (up to 15 ft or 4.6 m) in SE 2 district. The decision was expected to be made after SE 1 depillaring. Currently, SE 2 district has been depillared, however, the district was developed and depillared in a normal (12 ft or 3.6 m) height. It was also assumed to implement a high reach continuous miner in SE 2 district. Taking into account a 25 ft (7.6 m) reach continuous miner, the split pillar size was increased to 50 ft (15 m). The applied safety factors for these pillars (15 ft or 4.6 m) fluctuate from 1.68 to 2.0. Unfortunately, the difficulties with the MC have not permitted us to implement the increased height roadways test.

Following the completion of major modifications, the MC tests will be carried out in the 1 East road extension and eventually in the SE 4 district.

4.0 INSTRUMENTATION AND DATA COLLECTION

A comprehensive database related to the behaviour of roof, sides and pillar stability has to be compiled and interpreted for all aspects of mining activity. Numerous strata behaviour instruments distributed through the Demonstration Area as well as in regular mining districts have been installed and monitored.

The monitoring program was started at the end of June 1989, and continued as mining activity has progressed throughout 1990 and 1991.

The program was initially planned to be carried out in two regular mining districts (SW 7 and SW 8) and in two Demonstration Area districts (SE 1 and SE 2) and has been extended to two subsequent districts in the regular mining areas (SW 9 and SW 10) and also to a third district in the Demonstration Area (SE 3).

The collected data is regularly stored in a computer database. The monitoring data is also regularly analyzed to serve the current mining operation, however, the full analysis has to be done after the whole program is complete. A comprehensive analysis should produce a set of information of exceptional value for any underground coal operation particularly in mountainous seams.

SRCL is already utilizing strata control techniques and information derived from the monitoring program in its regular mining operation. At the current stage, since a full analysis has not been done, the monitoring program observation are used for roadway junction roof collapse predictions. This basically pertains to the depillaring activity, however, it may be utilized in any other roadway behaviour. SRCL has already started to install and monitor roof behaviour in main access roadways.

4.1 List of Subjects to be Monitored

Considering the whole area of the Demonstration Project, the following issues are being monitored:

- Road Behaviour

  a) Roof Depression (vertical beam translation)
  b) Rib Movement
     -Each rib horizontal translation
     -Total rib convergence
  c) Roof Bed Separation (movement within the strata between particular layers)

- Pillar Behaviour

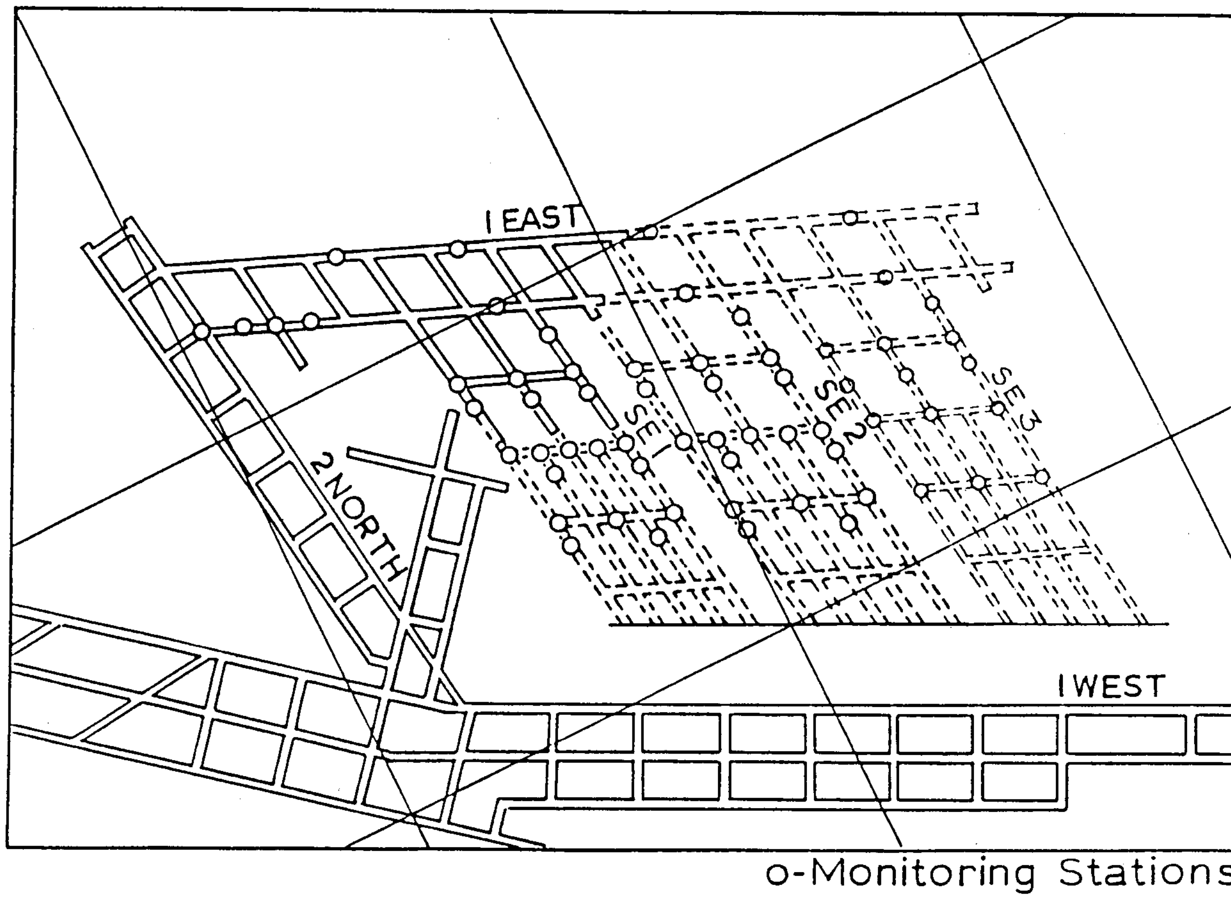
  Pillar behaviour, which is equally as important as road behaviour, tested during the program, contains factors such as inside pillar coal-mass movement, roof to floor pressure near the coal ribs (at sides of roadways) pillar squeezing, corresponding to residual pressure roof failure predictions, pillar pressure generated within the pillar during the development stage as well as during retreat mining. All this data, when related to the cover thickness, pillar size, activity stage, time factor, coal cleat and geological and tectonical conditions, should give us substantial information regarding pillar behaviour for future mine designs.

When extracting higher than normal (10 ft or 3 m) pillars, it is expected that their behaviour will differ from current pillar behaviour.

The obtained results will be compared to theoretical expectations so that we can obtain a significantly higher degree of confidence in pillar stress and strength calculation methods and their applicability to the SRCL area.

Figure 11 illustrates the location of the monitoring stations within the Demonstration Area. As mentioned before, monitoring stations were installed in regular mining areas (SW 7, SW 8 and extended to SW 9, SW 10 and 2 West main road).

FIGURE 11



DEMONSTRATION AREA - MONITORING STATIONS

4.2 Initial Conclusions

Stations located at roadway junctions are called "Junction Stations". Stations located between junctions are called "Full Stations".

Junctions, as areas being the most vulnerable to roof separation and eventual collapse and also as being part of the entire communication system in the mine, are of special importance. The junction's roof behaviour depends on numerous conditions such as:

- Location within the district or main roadway;
- Strata geological structure of the roof rock-mass;
- Distance from the mined-out district or from the depillaring zone;
- Roof rock-mass water condition;
- Roof strata strength characteristic;
- Tectonical disturbances within the roof strata;
- Cover thickness;
- Roof bolting quality; and
- Junction duration time.

All of these factors are taken into account while monitoring the installed stations. Obviously, an objective determination of some of them is a very difficult task. Nevertheless, the roof separation measured from installed devices is a definite value.

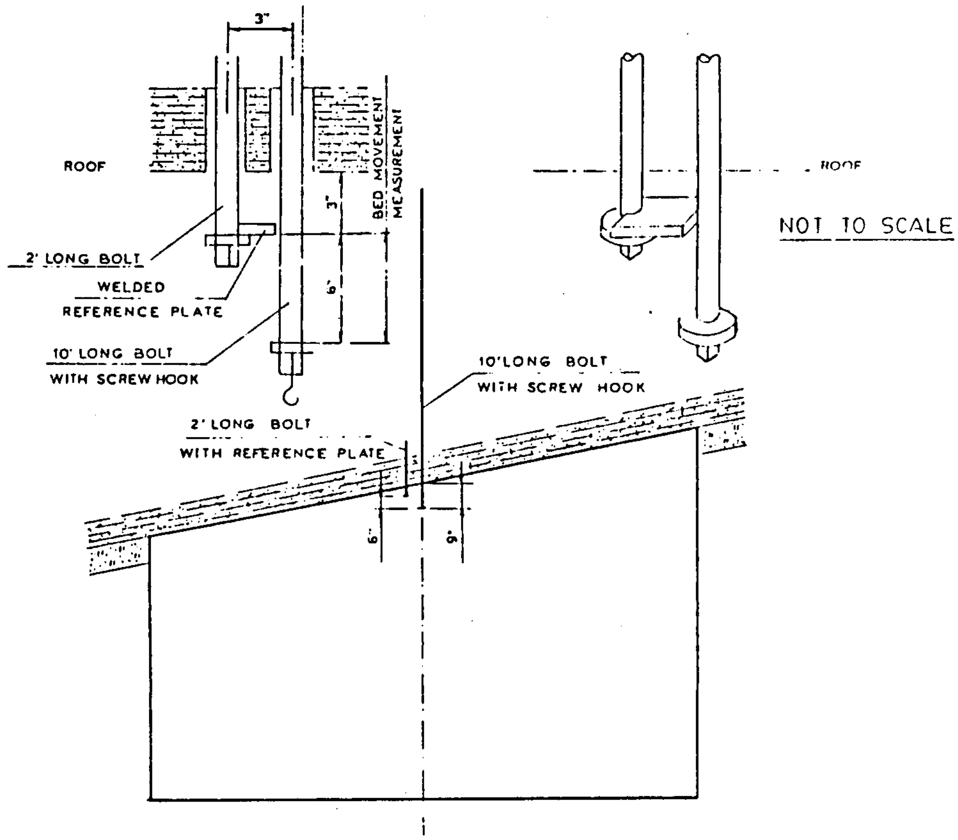
Basically, junction stations designation is to determine the roof strata separation within the first 10 ft of the immediate roof strata. Roof separation values are monitored in a time function. Since monitoring covers the time from the moment of excavation up to the point when the station area is depillared, the observation covers the entire time and all stages of mining activity.

Data already acquired from SW 7, SW 8, SW 9, SE 1, and SE 2 districts allow us to establish a point when roof separation becomes critical. As mentioned before, the results will be analyzed following completion of the project, nevertheless, current experience indicates that the 0.4 in (10 mm) value has to be considered as critical, i.e., the junction may collapse at any time. This pertains to normal geological conditions for No. 4 Seam, SRCL. If any abnormalities occur, they must be taken into account.

Obviously, it does not mean that we have achieved a 100% control of a junction's roof behaviour. An estimation of non-measurable factors has been and always will be a subjective matter. Nevertheless, we have achieved a significant step in more accurate estimation of the junction status. This gives us a much better understanding of the phenomena taking place in the junction roof strata, therefore, a higher confidence in the safety of the operation.

Figure 12 illustrates the standard array of junction stations.

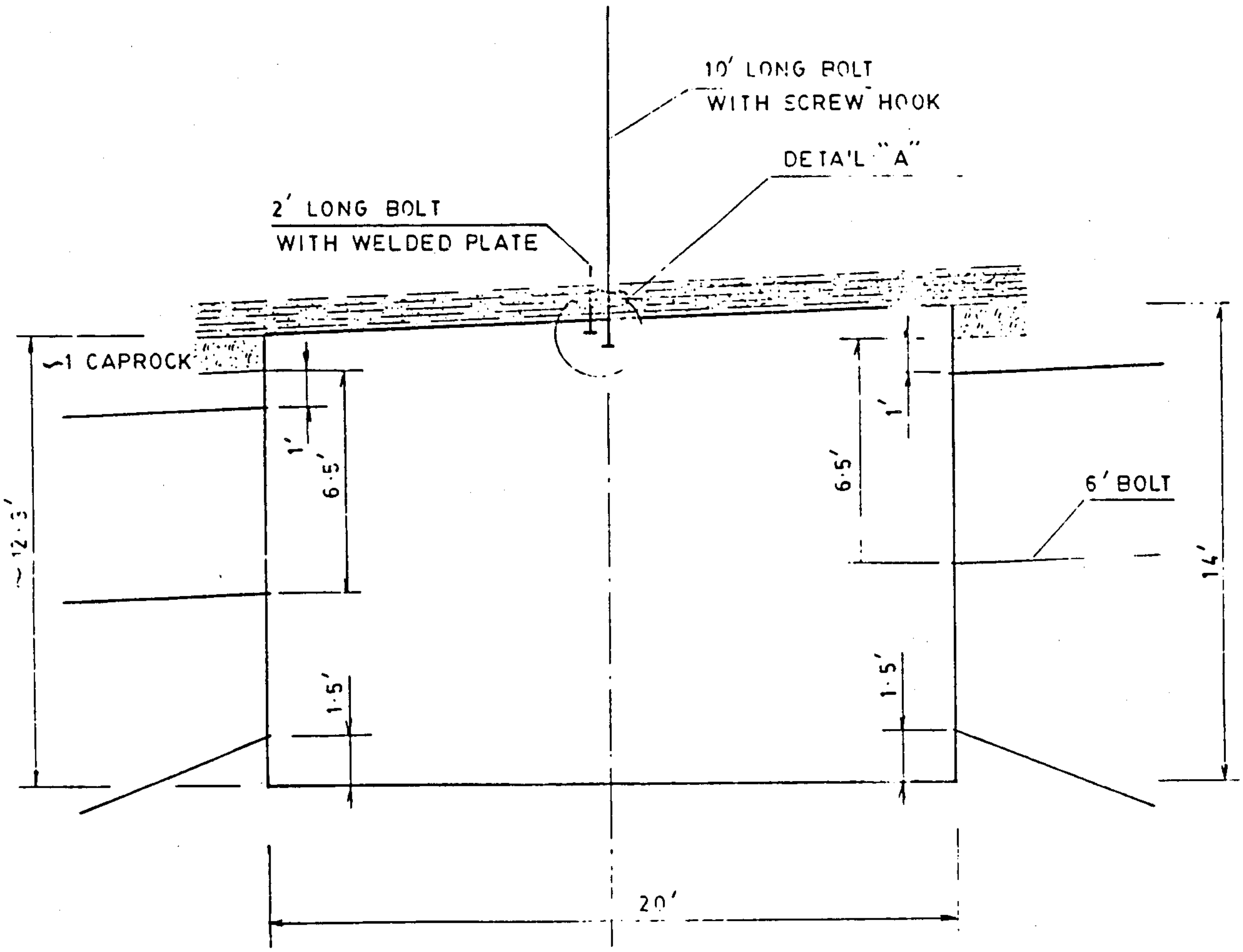
FIGURE 12



STANDARD ARRAY OF JUNCTION ST.

A "Full Station" designation is to monitor numerous aspects of the rock-mass behaviour. They are located between junctions at certain locations before roadways are planned. The rib, roof and pillar behaviour is monitored. These factors represent a complex set of phenomena that are interdependent on each other. The immediate interpretation is a difficult task. All of this data will be input for the full analysis after the program is complete. Nonetheless, some exceptional rib convergence and pillar squeezing values are directly corresponding with roof separation at the adjacent junctions and residing pillar stress. In such evident cases, the observations are also used immediately in daily operations. An interesting observation is that roof separation almost never takes place at normal conditions in the district area over the 20 ft wide roadways. (As mentioned previously, roof separation phenomena is quite common at junction areas.) This leads to the conclusion that the risk of roof collapse within the mining district (duration approximately 1/2 year) in normal conditions is very minimal. Again, as at the junction areas, a careful estimation of the related, non-measurable factors must be done by an experienced person to derive the final practical conclusion.

FIGURE 13



NOTE:– ALL MONITOR BOLTS MECHANICALLY ANCHORED

–ALL RIB BOLTS 6 FT. LONG

–DISTRICT ROADWAY GRADIENT AVERAGE 8°

STANDARD ARRAY OF FULL STATION

5.0 DEMONSTRATION SCHEDULE

The mine plan (Figure 11) shows the location of the test area. The Demonstration is split into several phases.

Phase 1 - included preparatory drivages and the initial entry pillars beyond the first mining section. All of these drivages were driven using existing standard techniques and equipment with the entry roads initially 10 ft (3 m) high, then gradually increased to 12 ft (3.6 m) and finally to 14 ft (4.3 m) high. At this phase, roads were mined with a multi-lift system to achieve the desired height. Once the entry (1 East) drivage was started, the monitoring program was also initiated (June 1989).

Phase 2 - included the drivage of the roadways to form the panel pillars as well as split pillars within the first mining district (SE 1). These roadways were formed by multi-lift methods with the MC in use. Roadways (pillars) were 12 ft (3.6 m) high with split pillars of 40 ft (12 m) wide. As mentioned before, major operational difficulties took place with the MC. Numerous trials and modifications have been necessary. As mining progressed, the monitoring system continued.

Phase 3 - included the depillaring of the first mining district (SE 1). Before it commenced, the high reach continuous miner was expected to be operating and a single lift depillaring method was planned to be used. Unfortunately, the difficulties with the MC postponed this stage.

To date, the second mining district has been depillared and the development of SE 3 district has been commenced. The work is carried out using standard techniques. The MC modifications have been completed and another trial is planned.

The project could have been successfully concluded if the MC had been capable of performing satisfactorily in the development and depillaring stages.

6.0 ACKNOWLEDGEMENTS

The authors wish to acknowledge the support of SRCL management and operating personnel in carrying out this project. Special recognition goes to those individuals who have been working in the project implementation and to their supervisors for their cooperation, hard work and ideas that help to continue a challenging venture in underground mine operations.

The contributions of the staff of Western Diversification Office, Alberta Office of Coal Research and Technology, Alberta Occupational Health and Safety, Energy Resources Conservation Board and United Steelworkers of America (Local 7621) are gratefully acknowledged.

7.0 REFERENCES

BARRON, K., WRIGHT, P.L., and SMALES, T., Guidelines for the Design of Coal Pillars in the No. 4 Seam at McIntyre Mines Limited, Canadian Mining Research Centre, Report 82/20-T, April 1982

BIENIAWSKI, Z.T., Strata Control in Room and Pillar and in Longwall Mining, Penn State University, Seminar, September 1986

BORECKI, M. and KIDYBINSKI, A., Coal Strength and Bearing Capacity of Coal Pillars, Proceedings of the Second International Congress on Rock Mechanics, ISRM, Belgrade, 1970, Vol. 2, Paper 3-21

DENKHAUS, H.G., A Critical Review of the Present State of Scientific Knowledge Related to the Strength of Mine Pillars, Journal of South African Inst. Min. Metall., Vol. 63, 1962, pp 59-75

KULACH, J., Pillar Design Criteria for Mining Activity in No. 4 Seam at Smoky River Coal Limited, Smoky River Coal Limited, April 1989 (not published)

"Novel Application of Oil Agglomeration Technology"

Y. Briker, K. Szymocha, T. Ignasiak, W. Pawlak and B. Ignasiak

Abstract

Laboratory-scale work conducted at the Alberta Research Council on the application of advanced spherical agglomeration led to the development of three independent processes:

- Agflotherm® for simultaneous upgrading of low-rank coals and heavy oils,
- Aglofloat® for desulfurization and deashing of high sulfur bituminous coals and for recovery of combustible material from coal fines and rejects,
- Clean Soil Process for clean-up of soils contaminated with various hydrocarbons.

This work was conducted under the sponsorship of the Canada/US Consortium on Coal Processing. Based on experience from batch studies, two continuous units were designed, a 5 kg/h continuous unit and later a 250 kg/h pilot plant. Over 100 tests were carried out to optimize circuit designs and process parameters.

Introduction

Environmental, technological and economic problems caused by utilization of low quality and low rank coals and also serious concerns related to a growing number of polluted industrial waste sites have significantly increased an interest in further development and application of oil agglomeration technology.

Oil agglomeration is one physical cleaning method which offers economic and efficient ways of converting poor quality fuels into a valuable energy resource. The principle of spherical agglomeration is based on differences in surface properties (contact angle, hydrophobicity) between carbonaceous and inorganic parts of coals. In a three phase (coal/oil/water) system, under a vigorous agitation, oil preferentially covers coal matter particles with a thin layer rejecting mineral particles into a water phase. As a result, new phases, agglomerates and water with the suspended mineral matter, are formed that can be easily separated. Agglomerates represent an improved coal product suitable for further utilization.

The Agflotherm® Process

Subbituminous coals and lignites can usually be surface mined at low cost and quite often are characterized by low sulfur content that makes the utilization of these coals environmentally preferrable. However, high moisture and very often high ash content, with corresponding low heating value, and poor washing characteristics pose certain economic limitations for these materials to be used in distant markets. It has been found that low rank coals, generally considered unsuitable for agglomeration due to their hydrophilicity, can be successfully agglomerated and beneficiated with heavy oil-based bridging oils.[1] Studies on advanced oil agglomeration conducted at the Alberta Research Council resulted in the development of the Agflotherm® process aimed at simultaneous upgrading of poor quality coals and converting heavy oils/bitumen into distillable liquids with properties similar to that of synthetic crudes.

The process includes oil agglomeration of the raw coal followed by thermal treatment of the agglomerated product. Depending on the response to agglomeration and the ash content of the feed coal, the agglomeration step can be either a one-stage process with direct agglomeration of the raw coal feedstock or a two-stage process with microagglomeration and separation followed by size enlargement. The microagglomeration is carried out at 25% to 35% solids concentration and 0.5% to 5% oil addition. Additional quantities of bridging oils (up to a total of 15% to 30%) are used for size enlargement in the agglomeration step. Thermal treatment of low-ash agglomerates is carried out in an inert atmosphere. A simplified flow-diagram of the integrated coal/oil upgrading process is presented in Figure 1. The agglomeration step has been scaled up to a 250 kg/h continuous operation and the thermal treatment to a 5 kg/h continuous process.

More than 20 low rank subbituminous coals have been tested using the Agflotherm® process. Table 1 presents the process conditions and the quality of raw coal feedstocks and product agglomerates generated at different temperatures.

Table 1

Agflotherm™ Process Performance with Subbituminous Coals

Sample	Moisture (wt%)	Ash (wt%)	Calorific Value (Btu/lb)	Moisture Capacity (wt%)
Feed Coal 1	16.3	24.6	7,840	19.4
De-oiled Agglomerates				
at T1	4.1	10.8	11,580	-
at T2	3.5	13.9	11,320	7.0
Feed Coal II	16.1	23.1	8,040	17.1
De-oiled Agglomerates				
at T1	3.9	9.4	11,730	10.8
at T2	3.3	10.2	11,740	8.2
Feed Coal III	14.9	13.0	8,850	18.9
De-oiled Agglomerates				
at T1	3.8	10.9	11,370	12.4
at T2	3.8	11.3	11,350	10.1

The effect of temperature on the recovery of oil for various coal/oil feedstocks is shown in Table 2.

Table 2

Effect of Temperature on Processed Oil Yield

Feed Oil	Temperature [°C]	Yield[a] [%]
Heavy Oil I	A B C	44 65 93
Heavy Oil II	A B[1] C	43 50 74
Bitumen/Condensate Blend (7:3)	A B[1] C	59 70 88

[a]Based on the oil present in agglomerates

Characteristics of oils derived from agglomerates in comparison with feedstock are presented in Table 3.

Table 3

Selected Properties of Feed Oils and Oils Derived from Agglomerates at 350°C

	Heavy Oil		Bitumen/Diesel Blend (4:1)	
	Feed	Processed Oil	Feed	Processed Oil
Yield, wt%	-	44	-	59
Density, g/mL @15.5°C	0.984	0.923	0.969	0.919
API gravity	12.2	21.7	14.4	22.5
Sulfur, wt%	4.1	1.55	3.67	1.61
Nitrogen, wt%	0.5	0.03	0.54	0.19
Atomic H/C	1.58	1.78	1.58	1.66
Distillation Cuts				
Naphtha (<177°C)	1.3	0.0	0.8	0.0
Distillate (177-343°C)	18.5	52.9	33.5	55.3
Gas Oil (343-525°C)	22.4	47.1	27.7	43.7
Resid (>525°C)	57.8	0.0	38.0	1.0

Results from Table 1 indicate that Agflotherm® process yields a high quality combustible material. Product agglomerates have a higher calorific value by 30% to 50% than the parent coal and significantly lower moisture and ash content. Capacity moisture is also greatly reduced. The temperature of thermal treatment does not affect the properties of de-oiled agglomerates as much, as the processed oil yield, which for some oils almost doubles when temperature increases.

Results from Table 3 suggest that the oil recovered from thermally treated agglomerates possesses a much better quality than the original oil used for agglomeration. It has higher API gravity, lower sulfur and nitrogen content, and basically no resid fraction. As a result of thermal treatment, redistribution of a resid fraction occurs which leads to an increase of light component ratio in product oil. Heavy components are precipitated in the agglomerates and contribute to an increase in their calorific value.

The Aglofloat® Process

Inspite of the high heating value, a number of bituminous coals can not be burned directly as mined due to their high sulfur content. Modified oil agglomeration, developed at the Alberta Research Council as the Aglofloat® process, efficiently removes ash and sulfur forming minerals but with high combustible matter recovery [2]. This method is based on the combination of agglomeration, flotation and washing as well as the application of very small quantities of heavy-oil derived bridging liquids. Two options of the process have been developed, single and two-stage, where the later is a combination of two single-stage processes with inter-stage wet grinding of the separated product for further size reduction and liberation of mineral matter and pyrite. Simplified flow-diagrams of both Aglofloat® processes are presented in Figures 2 and 3.

The process has been scaled up to 5 kg/h and subsequently to 250 kg/h continuous operations. The flow-diagram of the 250 kg/h continuous unit is shown in Figure 4.

A number of run-of-mine and precleaned coal samples, as well as several preparation plant streams, have been tested in batch and continuous operations. Results of the beneficiation of some bituminous coals by a single- and two-stage Aglofloat® process are presented in Table 4. Comparison of the Aglofloat® process performance for ROM Illinois No. 6 coal at three different levels (batch - 5 kg/h - 250 kg/h) is given in Table 5.

Table 4

Beneficiation of High Sulfur Coals by Aglofloat Process

	Ash %	Total S %	Performance CMR %	TSR %	ISR %
Run-of-mine Kentucky No. 9	14.6	5.08	-	-	-
Single-stage processing	6.8	3.62	94.5	39	60
Two-stage processing	4.4	2.81	93.8	54	84
Precleaned Illinois No. 6	12.0	3.34	-	-	-
Single-stage processing	6.7	2.93	92.6	23	54
Two-stage processing	6.2	2.89	92.0	25	57
Prep Plant Streams Cyclone Overflow	19.2	1.13	-	-	-
Single-stage processing	6.2	1.08	96.1	21	38
Two-stage processing	5.8	0.78	92.1	45	78

*All values on dry basis

Table 5

Continuous Aglofloat® Process Applied
for Run-of-Mine Illinois No. 6 Coal

	Feed		Product			
				Total Sulfur		
Oil Addition [%]	Ash [%]	Total S [%]	Ash [%]	Actual [%]	Reduction [%]	Coal Matter Recovery [%]
Batch Testing						
1.0	34.8	4.82	11.6	4.28	35	92
5 kg/h						
2.4	32.6	5.05	9.5	4.02	42	77
1.33	34.0	4.32	9.1	3.70	38	74
0.90	34.0	4.64	8.7	3.82	41	88
250 kg/h						
1.60	32.0	4.67	9.1	4.27	32	89
0.57	34.6	4.58	8.3	4.02	37	79

The performance of Aglofloat® process depends on the properties of the feed coal. In general, the process yields 75%-90% of inorganic sulfur rejection for run-of-mine coals and 40%-55% of inorganic sulfur rejection for precleaned coals with over 90% combustible matter recovery.

The ash content of clean coal varies from 4% to 7%. Application of the two-stage procedure for run-of-mine coal improves the performance, on average, from 20% to 40% in terms of inorganic sulfur rejection. There is also a slight reduction in coal matter recovery. This is not observed in case of the precleaned coals, for which the process performance is not influenced by the experimental procedure.

Comparison of the Aglofloat® results for ROM Illinois No. 6 coal, obtained at three different testing levels, suggest a similarity of the performances for batch and continuous operations. Slightly higher values of total sulfur reduction obtained in 5 kg/h continuous unit can be explained by some losses of the combustible material.

<u>Clean Soil Process</u>

A growing number of industrial plant sites polluted with various hydrocarbons generated by the heavy-oil industry, tar pits from MGP operations and also accidental oil spills, have put a considerable demand for research on soil cleaning technologies.

Work conducted at the Alberta Research Council on the application of modified oil agglomeration has led to the development of the Clean Soil Process which would yield a clean soil with 0.1%-0.0% of contaminant and coal agglomerates suitable for combustion [3].

A simplified flow-diagram of the Clean Soil Process is presented in Figure 5.

The process utilizes coal as a cleaning agent and is based on the principles of oil agglomeration with contaminants acting as the bridging liquids.

The results of clean-up of various contaminated soil samples are presented in Table 6.

Table 6

Clean-up of Soil Samples Contaminated with Various Hydrocarbons

Sample #	Soil Contaminant	Concentration of Contaminant (wt%)	
		Feed	Processed Soil
1	Tar	4.0	0.15
2	Tar	1.2	0.0
3	Tar	5.4	0.3
4	Tar	1.6	0.2
5	Heavy Oil	0.4	0.04
6	Residual Oil	33.5	0.08
7	Diesel	7.8	0.06
8	Diesel	24.5	0.05

The results indicate that for most of the soil samples tested, an acceptable level of cleanability can be achieved by application of the Clean Soil Process. The process seems to be very efficient for cleaning soils contaminated with heavy oil/bitumen and light oil types of contaminants (less than 0.1% of remaining contaminant in the processed soil). In the case of some tarry wastes, the level of residual contaminant concentration is still above the desired 0.1%. Recent studies show that by changing the process configuration and by reprocessing of some streams, acceptable levels of soil remediation can also be achieved for the difficult-to-clean tar-contaminated soils.

Conclusions

For the last 10 years, the Alberta Research Council has been extensively working on modification and application of oil agglomeration technology for coal preparation and environmental clean-up. The following independent processes have been developed and extensively studied with the results achieved:

1. Aglotherm® process for upgrading of low rank coals and heavy oils generates:

   - a high quality solid fuel characterized by low moisture content (2% to 4%), high heating value (12,000 Btu/lb), good handleability and combustion characterististics.

   - a distillable oils suitable for pipelining and characterized by higher API gravity, lower heteroatom contents and absence of asphaltene fraction.

2. Aglofloat® process for cleaning high-sulfur bituminous coals generates:

   - a high quality fuel with significantly reduced ash content (4% to 7%) and with inorganic sulfur content being reduced by 70% to 90% for the ROM coals and by 40% to 55% for the precleaned coals. Similar performances in batch and continuous operations are achieved.

3. Clean Soil Process for cleaning of tar/oil contaminated soils results in:

   - clean soil with 0.10%-0.00% of residual hydrocarbon contaminant

   - by-product agglomerates suitable for combustion in industrial boilers.

REFERENCES

1. "Process for the Selective Agglomeration of Subbituminous Coal Fines", Canadian Patent No. 1216551, 1987.

2. W. Pawlak, A. Turak, Y. Briker and B. Ignasiak, "Coal Upgrading by Selective Agglomeration, Proc. Twelfth Annual EPRI Contractor's Conference on Fuel Science and Conversion, May 1987, Palo Alto, California, USA.

3. W. Pawlak, T. Ignasiak, Y. Briker, D. Carson and B. Ignasiak, "Novel Application of Oil Agglomeration Technology", Proc., Thirteenth Annual EPRI Conference on Fuel Science and Conversion, 1989, p.p. 6.1-6.33.

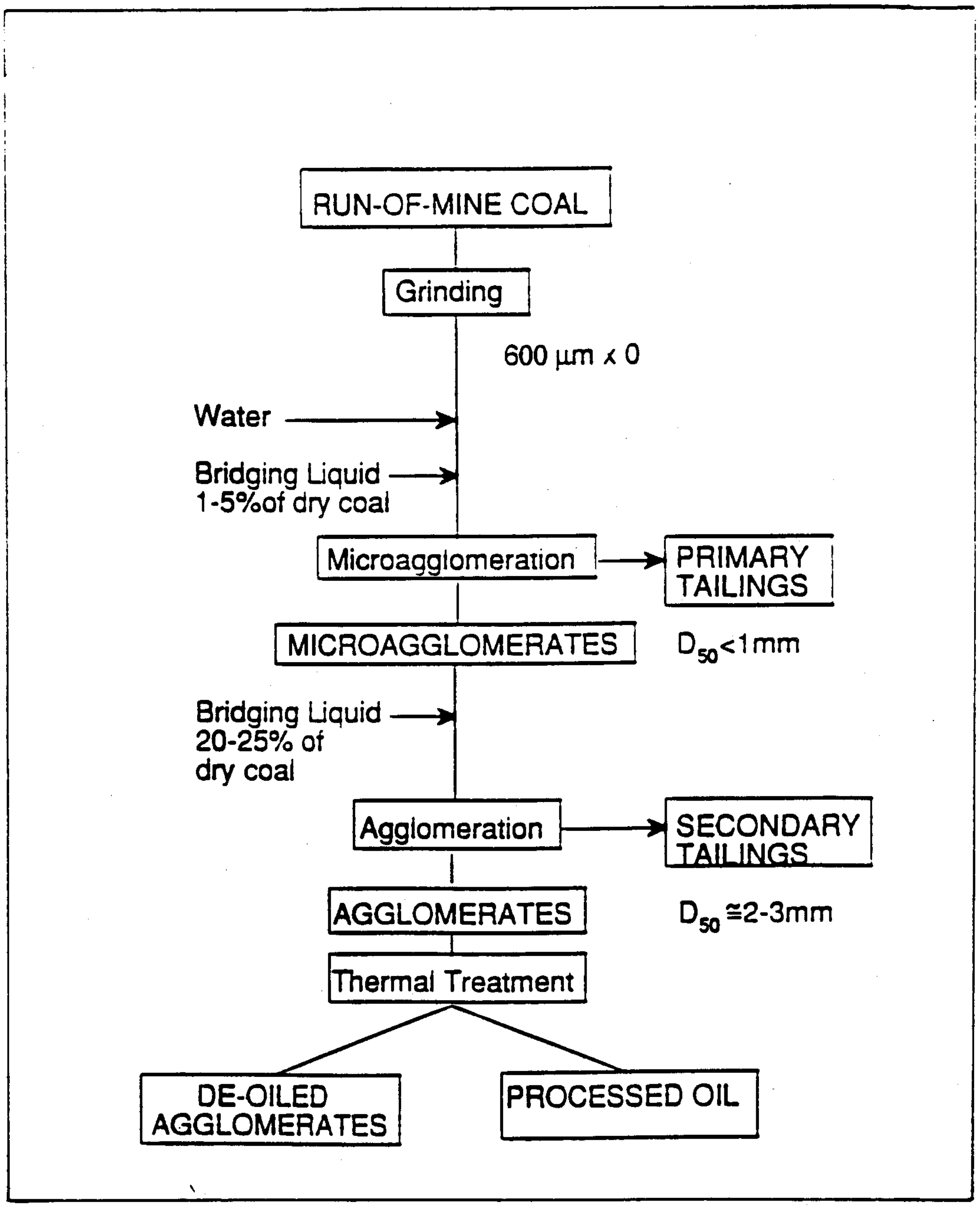


Figure 1

Agflotherm™ Process Flow-Diagram

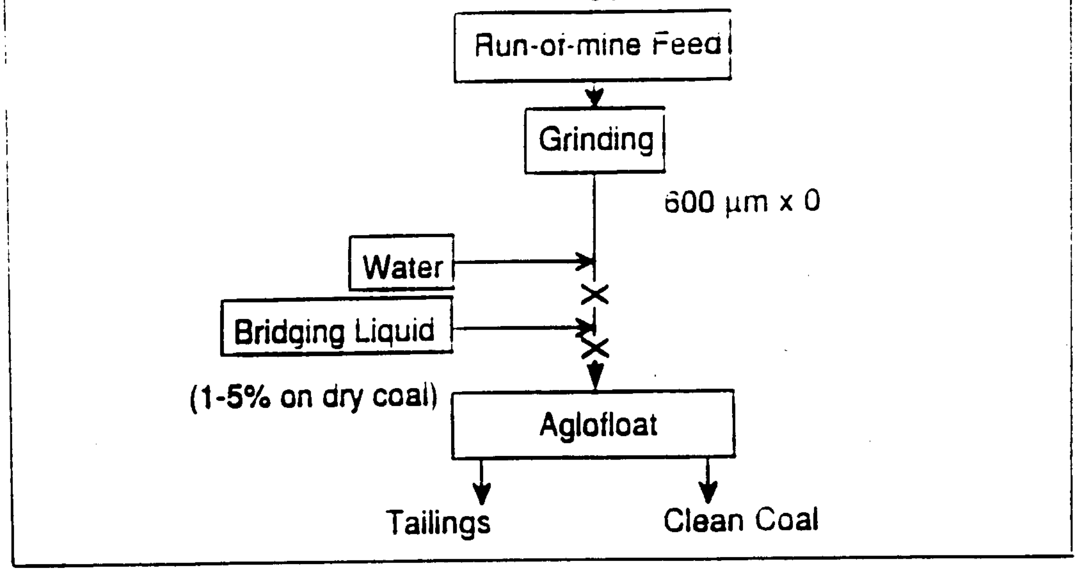


Figure 2

Single Stage Aglofloat™ Process

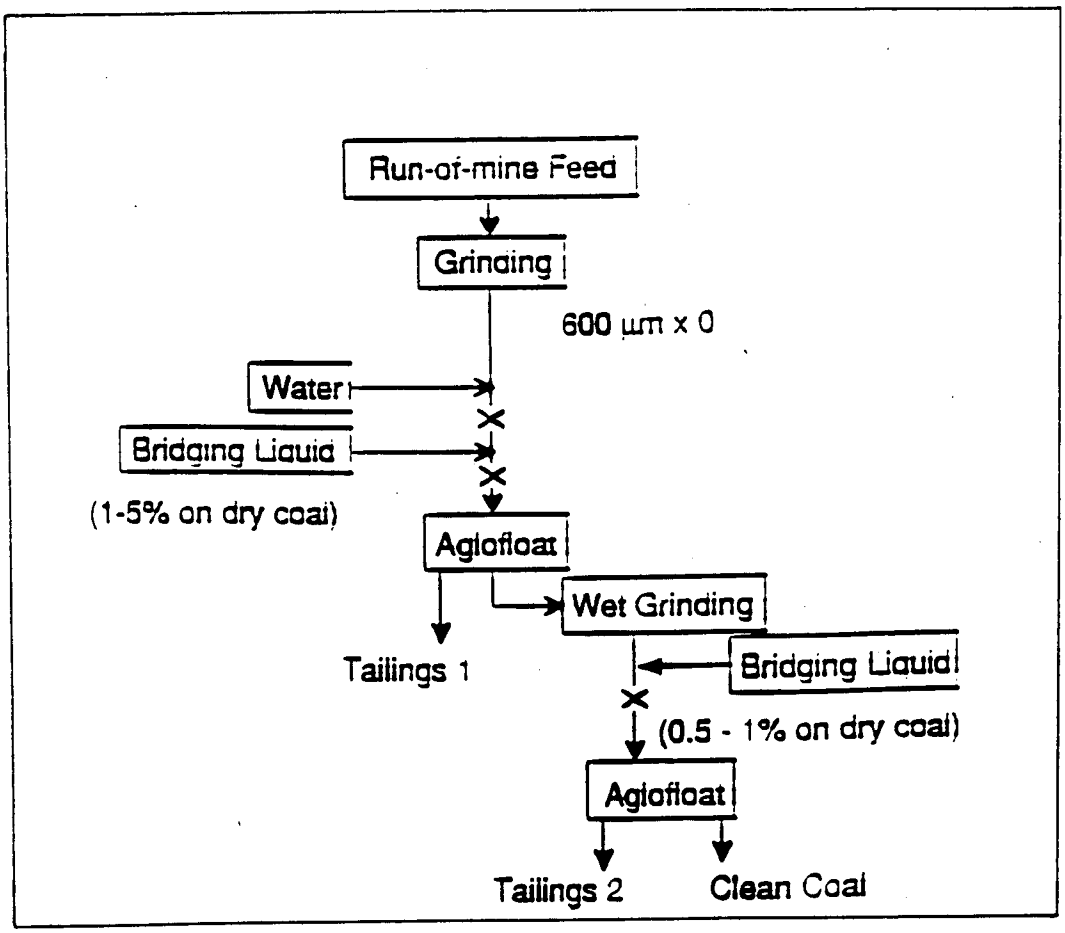


Figure 3

Two-Stage Aglofloat™ Process with Interstage Wet Grinding

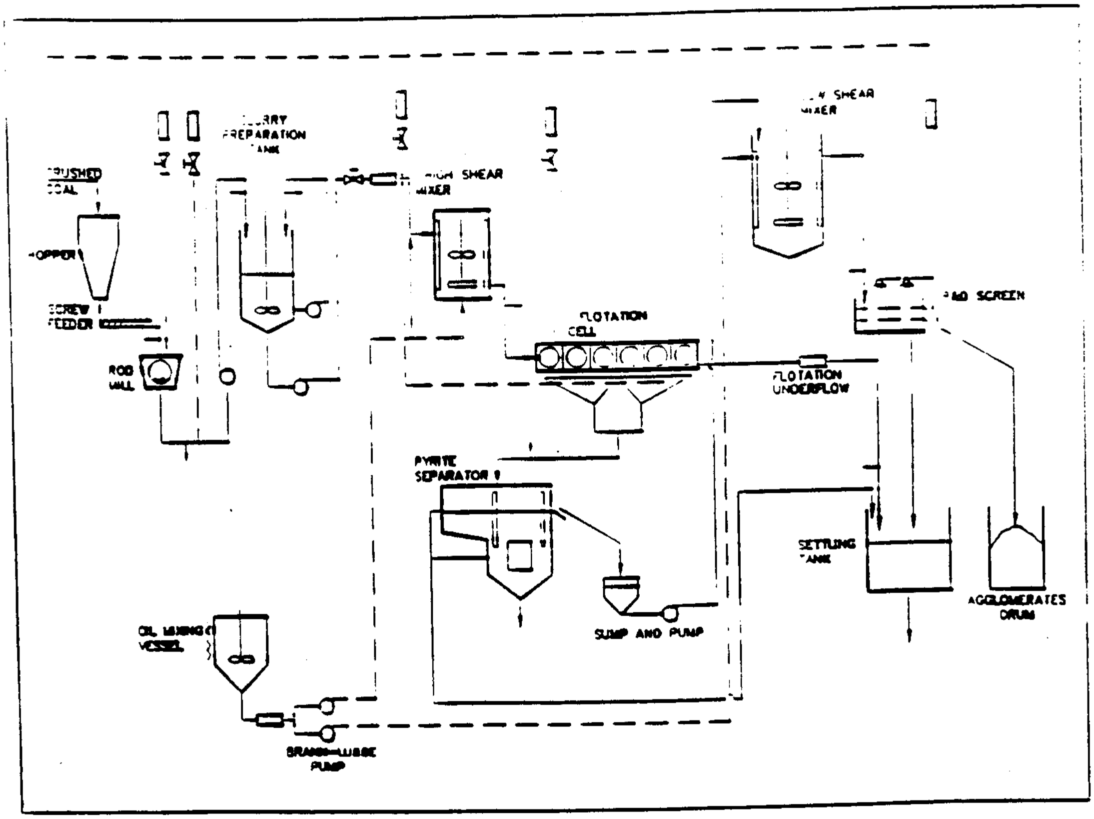


Figure 4

Simplified Flow Diagram of the Continuous 250 Kg/hr System

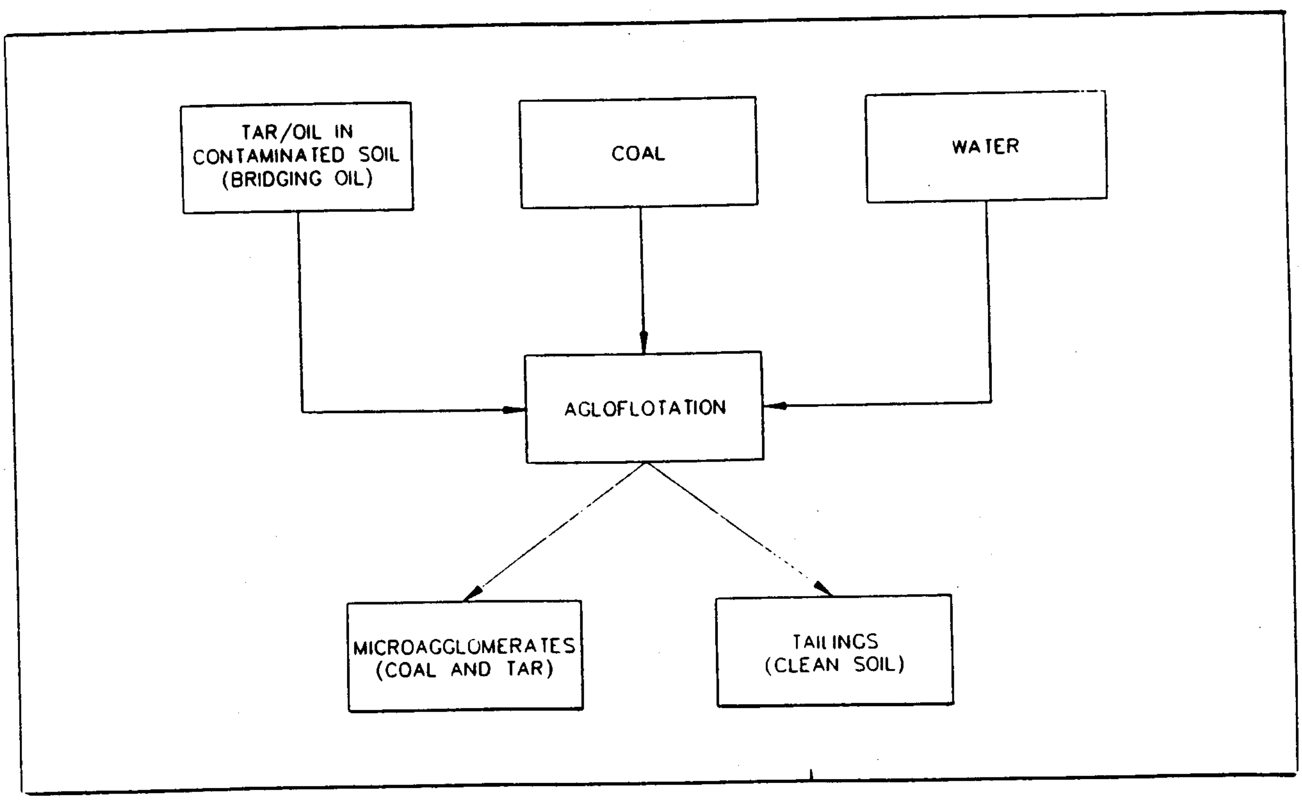


Figure 5

Process Schematics for Cleaning Tar/Oil Contaminated Soils

OIL-AGGLOMERATION DEMONSTRATION PROJECT AT SMOKY RIVER COAL LIMITED

Edgar Horner
Smoky River Coal Limited

David Fawcett
Fawcett Mining Consultants Inc.

October 1991

Prepared For 1991 Alberta Coal Research Contractors' Conference, October 30-31.

OIL-AGGLOMERATION DEMONSTRATION PROJECT AT SMOKY RIVER COAL LIMITED

ABSTRACT

A demonstration oil-agglomeration pilot-plant project was carried out in the preparation plant at Smoky River Coal Limited in 1987. The National Research Council of Canada oil-agglomeration process was demonstrated on Smoky River coal fines. The objectives of the project were to improve the processing plant throughput and yield, while enhancing the handling characteristics and reducing the dustiness of the clean product. The project included oil agglomeration pilot-plant evaluation, pilot-plant spirals testing, pelletizing tests and evaluation of the properties of the agglomerated products.

This paper presents a commentary on the results of the demonstration project, including applications of the technology, the economics of the process, and potential effects of the technology on the environment.

INTRODUCTION

Smoky River Coal Limited (SRC) produces a high-quality, low-volatile, metallurgical coal from its operations near Grande Cache, Alberta.

In 1987, SRC conducted pilot-plant tests using the National Research Council (NRC) oil-agglomeration technology at the Smoky River preparation plant.

The NRC process was evaluated and tested with the objectives of improving the processing-plant throughput and yield while enhancing handleability and reducing dustiness of the clean product.

The project included oil-agglomeration pilot-plant evaluation, pilot-scale spirals testing, pelletizing tests, and a thorough evaluation of the properties of the oil-agglomerated products - including dustiness, handleability and carbonization testing. Following the pilot-scale testing and technical evaluation, an economic evaluation of applications of oil-agglomeration at SRC was undertaken in order to put the process capital and operating costs into commercial perspective.

A project report was prepared and submitted to the Alberta Office Of Coal Research and Technology in November 1988. With the exclusion of the spirals-testing component, this paper summarizes the work done on that project, along with reviewing the results and potential applications of the technology.

BACKGROUND

Smoky River Coal Limited

The original plant at SRC was designed and constructed in the late 1960's. In general, the same process has been used since that time with minimal modification to the equipment and process. The process involves heavy-media cyclones on the plus 0.5 mm material and froth flotation on the minus 0.5 mm material, with the combined product being thermally dried in a gas-fired dryer.

As mining has progressed away from the original mines in the Smoky River valley, the run-of-mine coal characteristics have changed. There has been an increase in fines content as a result of geological factors, increased percentage of surface coal, and increased coal handling. The run-of-mine ash content has trended lower with raw coal ash contents of less than 10% becoming common in recent times.

The typical range for the raw-feed fine-size content considered in this project was:

minus 0.60 mm: 35 to 50%
minus 0.15 mm: 13 to 25%

The standard clean product produced at Smoky River is specified at 7% ash and 7% moisture.

As a result of the increasing fines content, the fines circuit was becoming the bottleneck in maximizing plant throughput and coal recovery. SRC was examining means to remove this bottleneck and hence was persuaded to evaluate the NRC oil-agglomeration process.

Secondary to the process evaluation, SRC, along with other producers of metallurgical coal in western Canada, was also seeking means to reduce dustiness and improve handleability of the coals. The NRC process, perhaps enhanced with pelletizing, offered the potential to treat the fines to reduce the problem.

NRC Oil-Agglomeration Process

The preferential wetting of solid surfaces by immiscible liquids and agitation of the coated particles to form agglomerates has been used in different ways for some time. Coal particles are essentially hydrophobic and they readily agglomerate with oils in an aqueous slurry when subjected to high-shear agitation. The hydrophibic mineral particles remain suspended in the water.

Many papers have been published on the use of oil agglomeration for coal cleaning; however, the recent work by NRC renewed interest in selective oil agglomeration and promoted the examination of the process by SRC for Smoky River metallurgical coal.

The most significant operating and economic parameter in the agglomeration of fine coal is the relative amount of oil used. When, progressively larger amounts of liquid are added to a suspension of fine particles, a variety of agglomerated products can result. At low levels of bridging liquid only pendular bridges form between particles and an unconsolidated two-dimensional floc structure exists. As the funicular region of bridging liquid levels is reached, the flocs consolidate somewhat towards a three dimensional floc structure. Finally, in the capillary wetting region, all the interparticle voids are filled with oil and coal pellets are formed. (Capes et al, 1987)

Early oil-agglomeration work concentrated on the capillary wetting region which involved much higher concentrations of oil (10% by weight and greater). The NRC oil-agglomeration process involves much lower additions of oil and works within the pendular/funicular wetting region to produce microagglomerates, or flocs. By using smaller additions of oil (less than 5% by weight), the NRC process offered potential to be commercially attractive.

As a result of the low oil-addition levels, the NRC process required the development of an efficient mixing system to adequately blend the oil and coal together. In addition, the lower levels of oil do not promote displacement of moisture within the fine coal and an efficient means of drying is necessary to aid in moisture removal. Consequently, the NRC process necessitated the development of innovative coal-processing techniques; the NRC mobile oil-agglomeration pilot-plant has served as an excellent means to demonstrate and adapt this technology to the coal industry.

TEST PROGRAM

Laboratory Evaluation

Laboratory testing of samples of the fine coal feed was carried out both by SRC in its laboratory and by NRC to assess the characteristics of the material and the potential for the process. Several oils were tested for the process.

All the test samples gave favourable results and were successfully separated into clean product and tailings fractions. The minus 0.60 mm froth-flotation-feed samples produced combustible recoveries in excess of 90% with an ash content of 8 to 14%. Both No. 2 and No. 4 fuel oil worked satisfactorily - no difference was observed between the two oils.

Pilot Plant

As a result of the successful laboratory tests, it was decided to proceed with pilot-plant tests using the NRC oil-agglomeration pilot-plant.

After doing preliminary evaluation of economics and process needs, it was decided to classify the froth-flotation feed at minus 0.15 mm and feed the minus 0.15 mm material to the oil-agglomeration plant.

During the operating runs made with the pilot plant, 15 separate tests were made. Of these, 9 tests were done with minus 0.15 mm plant feed and 6 tests were done with thickener underflow (at minus 0.60 mm). No. 4 fuel oil was used for 12 tests and waste oil was used for 3 tests.

Process samples were taken throughout the test circuits for immediate and follow-up laboratory testing. Large samples of the product were collected for coal properties, pelletizing and handleability testing.

Pelletizing

To test a process for preventing the break-up of the oil agglomerates and to further enlarge particle size, test pelletizing was undertaken. Samples of the minus 0.15 mm oil agglomerates and minus 0.60 mm unagglomerated clean coal were sent for testing to Teledyne Readco, a leader in pelletizing technology.

Properties

Two sets of carbonization tests were carried out by CANMET to determine the impact of oil agglomerates on the coking properties of the clean coal product.

Three sets of dustiness/handleability tests were carried out by the Coal Mining Research Company (CMRC) to determine if the product coal containing oil agglomerates favourably impacted dustiness and handleability. The National Coal Board's Durham Cone Test and the ASTM Dustiness Test were used to evaluate handleability and dustiness respectively.

TEST RESULTS

Oil-Agglomeration Process

The following conclusions were made with respect to the NRC oil-agglomeration process for the minus 0.15 mm plant feed coal.

- The oil-agglomeration-froth separator produced 90% yield (96% combustible recovery). Equivalent yields with froth flotation were estimated by SRC at 85%. The ash content, however, was significantly higher affecting potential overall product quality and properties.

- The ash content of the product could not be controlled by adjusting the oil-addition level. The froth-separator product ash varied between 7.3 and 10.5% (average of 9.2%). When considering the amount of minus 0.15 mm material in the product (15 - 25%), this variance should not pose a problem in meeting clean coal specifications.

- The yield did not vary to any considerable extent over the range of oil-addition levels tested. Oil-addition levels of about 1.5% by weight should be sufficient to ensure the high recoveries in evidence with the pilot-plant testing.

- Although only one test with waste oil on the minus 0.15 mm feed was performed, results indicate slightly lower yields as compared to clean oil. Laboratory results indicated that about 25% more waste oil is needed to achieve similar yields to that of clean oil. On this basis, waste oil-addition levels of 2% by weight should be sufficient to ensure high recoveries.

- The pilot-plant testing showed poor recoveries with the screen-bowl centrifuge (average of 60.8% mass yield). Testing done using a commercial-scale centrifuge indicates that the solids recovery would be 99%. However, in order to ensure product moistures of 15% to 20%, laboratory centrifugation data indicates that oil-addition levels of 2.4% to 4.0% are necessary. This is higher than the estimated 1.5% oil addition required to adequately clean the coal. If thermal drying of this material is required, then some of the oil would be evaporated, causing the agglomerated particles to break-up.

  Pilot-plant testing of the minus 0.60 mm thickener underflow showed higher clean coal ash values of 14.5 to 22.2% with yields of 53.8 to 69.0% (not including Test No. 7). The high ash values and subsequent low FSI make this product undesirable for coking coal.

Pelletizing

A pinmixer pelletizer successfully pelletized the minus 0.60 mm unagglomerated coal but problems were encountered with pinmixer pelletizing using minus 0.15 mm oil agglomerates. Further testing, with a conventional pelletizing disc, with the oil agglomerates successfully produced pellets of 10 to 13 mm in diameter.

Although the testing with the conventional pelletizing disc did produce good sized agglomerates, the use of the lignosulfinate binder appeared to cause a deterioration of the rheological properties of the coal. Also, the oil-agglomerated pellets exhibited lower strength than normal non-oily coal pellets as the oil coating apparently reduces the binder strength.

Properties

Results of the carbonization testing indicated that the coking properties, as measured by coke strength, of Smoky River product coal containing agglomerates, were not significantly reduced.

In general, with respect to dustiness and handleability testing, the dustiness was reduced while handleability was unchanged. These properties, however, appeared to be very sensitive to the manner in which the normal plus 0.15 mm product and the minus 0.15 mm oil-agglomerated coal were blended. When air-dried as a blend, significant negative handling properties and some increased dustiness was apparent contrary to what was expected. When the oil agglomerates were pre-dried before blending, no significant change in handling properties, but reduced dustiness, was apparent. Air-drying as a blend may have led to over drying of the plus 0.15 mm material because of the resistance of the agglomerates to drying.

ENVIRONMENTAL FACTORS

In evaluating a new process, or a process modification, for implementation, an assessment of the environmental factors should be made.

The primary environmental factors to be considered in evaluating the application of the oil-agglomeration process versus the present coal-cleaning process are determined to be:

- concerns arising from the use of oil,
- impact on particulates emission, and
- energy conservation.

The use of oil may present several problems in the storage and transportation of coal as a result of the vaporization of a portion of the oil and/or oil contamination of surface water.

Since the NRC process uses a relatively light-weight oil, there could be some vaporization of the oil and hence odour coming from the stockpile sites. This could be particularly sensitive in areas where stockpiles are relatively close to habitation; for example, the port stockpiles at Neptune Terminals in Vancouver. The vaporization could potentially cause problems in the holds of ships used to transport the coal to offshore customers if there was not adequate ventilation.

There is also potential for some oil to contaminate surface water requiring more extensive water-treatment facilities at coal handling and storage sites.

The potential to reduce the dustiness of the product is a positive factor. Coal dusting is an environmental problem in the stockpiling, handling and transportation of coal and means are being actively sought to reduce the emissions.

The maximization of energy recovery from a resource base has long-term positive environmental implications, since:

- less energy would be expended per unit of recovered coal (more tonnes for same, or slightly higher, energy output), and

- less "coal" would be lost to reject material from the process, thereby reducing the size of waste-disposal piles and sites.

There are limits to the recovery, however, when the product quality deteriorates to where it becomes unsuitable for customer requirements, or when energy is lost because of additional requirements for environmental controls, coal handling, or from the loss of oil (energy) through vaporization.

The extent of the environmental impacts of these factors was not addressed in the demonstration project.

ECONOMICS

An economic analysis was carried out to determine the commercial viability of the NRC oil-agglomeration process as a solution to the needs of SRC for improving the present metallurgical-coal process.

The analysis compared the operating and capital costs of modifying the plant and process to include oil agglomeration versus more conventional means of improving the present froth-flotation system.

The primary factors influencing the economics were determined to be:

- capital cost,
- the amount of oil per tonne required and the cost of that oil,
- coal recovery, and
- sensitivity to changes in the above factors.

The capital cost of a plant modification was determined to be very high in comparison to other potential processes. With reference to the SRC requirements, the capital cost of an oil-agglomeration plant versus increasing the froth-flotation capacity was $6.7 million or over 300% higher. This would amount to an increase per raw tonne of $0.75 based on amortizing the investment over 10 years (which is an exceptionally long period for the coal industry today).

The operating costs, excluding oil usage, were determined to be higher for oil agglomeration versus froth flotation - $0.30 per tonne versus $0.10 per tonne.

The major factor influencing the economics of oil agglomeration is the percentage of oil required and the cost of that oil. At an oil addition level of 2% (i.e. 20 kg per tonne) and the cost of oil at $0.15 per litre, the cost-per-tonne processed would be $3.50.

The potential to recover the higher capital and operating costs lies in coal recovery. The test work, although not considered conclusive, showed that a higher recovery of coal could be achieved with the oil-agglomeration process. The higher recovery, if achieved, would have a major positive impact on the revenues per tonne of resource mined and processed. The following is an example of the economic impact.

> Assuming a price of $40.00 per tonne of product is received FOB mine, and conventional process coal recovery (yield) of 85% is achieved, the producer would be receiving $34.00 per tonne run-of-mine. If the yield were improved 5 percentage points (i.e. to 90%), the producer would receive $36.00 per tonne run-of-mine. Hence, the producer could spend up to $2.00 per run-of-mine tonne to improve yield by 5 percentage points.

Not considering the value of the potential secondary benefits (e.g. reduced dustiness), nor considering the potential environmental factors, it appears that the process is not economic for the specific application being considered.

Investment in an oil agglomeration plant for the SRC situation should be considered high risk at this time because of the high up-front capital, the need for more extensive testing to support the yield increases and the volatility of the price of oil.

In some situations, economic application of this technology may be possible in the reprocessing of the fine refuse (tailings) from coal preparation plants. This could be done as an integral part of the system, or in the processing of old tailings. NRC process-based plants have operated at sites in the eastern United States for this purpose. This application is not viable at SRC since the Alberta Power Milner plant is designed to burn tailings without further ash reduction.

ADDITIONAL RESEARCH CONSIDERATIONS

Although the pilot-plant testing of the NRC oil-agglomeration process successfully demonstrated that the process could clean the minus 0.15 mm plant-feed coal with a significant increase in yield compared to conventional froth flotation, there would need to be a substantial amount of additional research and evaluation done before the process could be commercially implemented for processing of western Canadian metallurgical coals.

The aspects to be considered for further research, development and evaluation are:

- capital costs for an oil-agglomeration plant,
- technical and economic comparison of alternative processes,
- use of heavier oil and of waste oils,
- confirmation of oil-addition levels required,
- impact of oil vaporization,
- impact of oil on the total process, impact on plant equipment and oil loss to waste (tailings),
- centrifuge drying process,
- pelletization of oil agglomerates, and
- testing of coking properties of oil agglomerates.

The capital cost of an oil-agglomeration plant appears to be very high for the throughput capacity. A plant and equipment engineering evaluation could be undertaken to verify the designs and examine means to reduce the cost of the major components.

A more in-depth assessment of alternative processes, both conventional and novel, is required to be able to more firmly establish the comparative technical and economic merits of the processes.

Use of a heavier oil would reduce the potential for vaporization of the oil and contamination of surface water, and would potentially be less expensive. On the other hand, heavier oil would require heating to improve viscosity, particularly in cold weather. The impact of a heavier oil on the pelletizing of oil agglomerates would need to be determined through further testing.

Use of waste oil was considered and tested in the SRC demonstration project. It was determined that waste oils (from the mine-maintenance facilities) could be used; approximately 25% more used oil was required than new oil to obtain similar yields. Because the cost of waste oil could be relatively low, research to further address means to "clean" this oil and to further test it in the process is merited. An economic evaluation of waste-oil supply would need to be done to better ascertain longer term availability, cost, and opportunity cost of not recycling the waste oil for other uses.

Since the operating cost using this process is very sensitive to the oil-addition percent, further testing of the impact of the various addition levels on potential variations of the run-of-mine coal is merited.

Test work should be carried out to define the impact of oil vaporization on the environment, coal transportation and economics.

When oil addition is made to a circuit in a plant wherein the water supply is recirculating, consideration must be given to assessing the impact that oil contamination would have on other parts of the process and on the tailings system.

With respect to the NRC oil-agglomeration process as demonstrated, the centrifuge-drying process needs to be closely examined, and further pilot-scale testing undertaken to determine oil-agglomerate product-moisture contents when high centrifuge recoveries (greater than 95%) are achieved. If the moisture content cannot be shown to be significantly reduced (30% to 15% moisture) at oil-addition levels of 2% or less, then the affect of thermal drying on oil loss and hence dustiness properties merits careful evaluation. Heavier oils would show less evaporation and their performance should be evaluated.

In order to take full advantage of the potential to reduce dustiness, further testing of pelletizing processes should be carried out to determine the equipment design and selection parameters and to find a suitable binder.

Finally, further testing of oil agglomerates and pelletized oil agglomerates should be done to better determine the impact of the process and oil addition on coking properties.

Should the additional research and development work outlined above give favourable results supporting further development of the process, the next stage should be a demonstration plant designed at approximately 50 tonnes per hour.

CONCLUSIONS

Preparation of this paper has provided an opportunity to reassess the results of the oil-agglomeration demonstration project carried out at Smoky River Coal Limited.

Although the project demonstrated the potential of the NRC process, it is felt that many questions were not fully answered and additional questions arose from the results of test work. It was beyond the scope and financing of the project to investigate various factors in greater depth. At that time, with the economics being marginal, with the many unknowns, and with limited capital funds, Smoky River Coal Limited was not interested in further pursuing the development and demonstration of the process.

Based on the results of the demonstration project and the evaluation work done, the authors do not believe that sufficient information is available to definitively conclude the economic viability of the NRC oil-agglomeration process for processing of metallurgical-coal fines in all situations.

ACKNOWLEDGEMENTS

The research project, on which this paper is based, was administered by the Alberta Office of Coal Research and Technology, and was funded in part from the Alberta/Canada Energy Resources Research Fund, a fund jointly sponsored by the Government of Canada and the Government of Alberta. Additional funding was also received from Energy, Mines and Resources, Canada and the Industrial Research Assistance Plan (IRAP) of the National Research Council Canada. The encouragement, co-operation, and assistance of management and staff at Smoky River Coal Limited and the National Research Council Canada, Division of Chemistry throughout this project is acknowledged.

REFERENCES

Bennett et al., 1987. Combustion of Agglomerated Coal Fines Blended with Pulverized Coal. Special National Research Council Canada (NRCC) Report No. C-1145-875. December 1987.

Capes, C.E., et al, 1986. Waste Coal Fines as a Source of Slurry Fuels: Preparation and Combustion Aspects. Proc. Powder Bulk Solids Conference, London, 1986.

Capes, C.E., Coleman, R.D., and Thayer, W.L., 1982. Selective Oil Agglomeration: An Answer to Fine Coal Treatment Problems. Proc. 64th CIC Coal Symposium, Ottawa, 1982.

Capes, C.E., Coleman, R.C., and Thayer, W.L., 1985. Oil Agglomeration for fine Coal Processing. Proc. 4th International Symposium on Agglomeration, Toronto, 1985.

Capes, C.E., et al., 1987. The Recovery and Utilization of Fine Coal. 89th General Meeting of the CIM, Toronto, 1987.

Capes, C.E., 1988. "Comments" on Draft Report to Alberta Government on Oil Agglomeration Project. August 26, 1988.

Fuels Evaluation Report, 1986. In-house report for Smoky River Coal prepared by Energy and Environmental Research Corporation. September 1986.

Germain, R.J., and Long, D.F., 1986. Coal Handleability: The Durham Cone Test. CMRC Report 85/67-T2, May 1986.

Smoky River Coal Limited, 1988. Final report on SRCL/NRC Oil Agglomeration Project.

TITLE: Application of Electrocoagulation to Tailings Reclamation

AUTHORS: J.C. Donini, C.W. Angle, D.K. Garand, K.L. Kasperski, C.K. Preston, K.L. Kar, T.A. Hassan, S.S. Thind[1], R. Latimer, D. Miller[2] and K.J. Cochrane[3]

AFFILIATION: [1]CANMET/Coal Research Laboratories (CRL)
#1 Oil Patch Drive, Devon, Alberta

[2]Luscar-Sterco (1977) Ltd.
P.O. Box 5000, Edson, Alberta

[3]Alberta Office of Coal Research and Technology (AOCRT)
11th. Floor, 9945 - 108 Street
Edmonton, Alberta

ABSTRACT: AOCRT, CANMET/CRL and Luscar-Sterco have collaborated in the development and testing of a novel electrocoagulation pilot-scale unit. This study encompassed all stages of development, from the fundamental bench work to the preparation of specifications and testing of the pilot-plant system. CANMET completely funded the construction of the pilot-scale cell and the investigation of the system on test mixtures. The three partners shared the cost of investigating the problems arising from the use of a real-life feed, which was typically an untreated thickener feed from the Luscar-Sterco washery.

The pilot-scale (2-20 L/min) system that is now available can be easily scaled up and is technically successful. It does not meet Luscar-Sterco's specifications for costs of consumable commodities, however, which must not exceed $0.05 for aluminum and electricity per cubic metre of treated water. Thus, the process is unsuitable for full-scale application at the Luscar-Sterco wash plant.

The fundamental parameters that influence the system design and operation, and their relevance to the above conclusions are discussed.

Other applications where the economic specifications are not so stringent, and electrocoagulation is economical, are described.

SOLIDS DISTRIBUTION IN SLURRY FLOW IN A LINEAR MANIFOLD WITH A HORIZONTAL APPROACH

H.A. Nasr-El-Din and J.H. Masliyah

Department of Chemical Engineering
University of Alberta,
Edmonton, Alberta T6G 2G6, Canada.

Abstract

An experimental study was conducted to examine solids distribution in branches of a linear manifold having a horizontal approach. The manifold was designed to maintain approximately a constant slurry velocity upstream of each branch. Solids concentration in the branches was measured for various manifold orientations (upwards, side and downwards) and upstream conditions.

Semi-empirical correlations for the three orientations were developed to predict a branch concentration ratio. This ratio was dependent upon the branch flow ratio, the particle inertia parameter and the vertical solids concentration profile upstream of the branch. The agreement between the experimental and the predicted branch concentration ratio was fairly good.

Introduction

Two phase flow through tees and manifolds is often encountered in chemical and mining industries. Industrial applications include: parallel passes in slurry preheaters in some coal liquefaction processes (Segev and Kern, 1985); and solids distribution in oilfield perforated casings (Haynes and Gray, 1974; Gruesbeck and Collins, 1982). The presence of a non-uniform solids concentration profile upstream of a branch and the difference in inertia between the liquid and solids phases result in maldistribution of solids in the branches, and consequently potential operating problems.

The degree of separation that occurs in a manifold can be assessed using two criteria. The first criterion is the branch concentration ratio or the branch separation ratio, S_i, defined as:

$$S_i = C_{bi}/C_{hi} \qquad [1]$$

C_{bi} and C_{hi} are the solids concentration in branch i and its header, respectively. Due to the axial variation of the solids concentration along the manifold, a branch concentration is normalized using the average solids concentration in the header immediately before the branch.

The second criterion is the transport efficiency, E_i, defined as the solids flow rate in branch i to the total solids flow rate in the main pipe. E_i is expressed as:

$$E_i = C_{bi}Q_{bi}/C_M Q_M \qquad [2]$$

where C_M and Q_M denote the solids concentration and the volumetric flow rate in the main pipe, respectively. Q_{bi} is the volumetric flow rate in branch i. This measure can be used when designing a manifold based on equal solids flow rate in the branches.

Most of the studies to date on slurry flow through a bifurcation junction were conducted with a simple tee with either a vertical or horizontal approach. One of the first studies, by Bugliarello and Hsiao (1964) analyzed phase separation of neutrally buoyant particles. The flow was vertically downwards and the branches were at various angles. They found the branch concentration ratio to be less than unity. This unexpected result was due to uneven solids concentration and velocity profiles upstream of the branch.

Torrest and Savage (1975) studied the collection of particles in small branches with a vertical approach. They found the solids transport efficiency to be independent of the branch size. The transport efficiency was higher for particles with low settling velocity and could be increased by increasing the branch flow rate. An empirical correlation was developed to estimate the transport efficiency in terms of particle settling velocity, upstream bulk velocity and branch flow rate.

Nasr-El-Din et al. (1985) studied particle transport into small branches (sampling ports) in vertical pipelines. They found that the branch concentration ratio was a function of the branch flow ratio and the particle inertia parameter (see Equation 5). The branch concentration ratio increased with decreasing particle size and with increasing the branch flow ratio and solids concentration. Nasr-El-Din and Shook (1986) examined the effects of the branch angle on solids distribution in vertical tees of various diameter ratios. They found that the branch separation ratio was a strong function of the branch angle, especially at a low branch flow ratio.

More recently, Nasr-El-Din et al. (1989a,b) studied particle segregation through a T-junction with a horizontal approach. The concentration ratios in the branch (defined as the branch solids concentration to that in the upstream header) and in the run (defined as the solids concentration downstream of the tee to that in the header upstream of the tee) were measured for various branch orientations and upstream conditions. For the upwards orientation, the branch concentration ratio was always less than unity, whereas the run concentration ratio exceeded one and it reached 3.5 in some cases. The opposite trends occurred for the downwards orientation. A relatively more uniform solids distribution between the branch and the run was obtained using the side orientation, especially with fine particles.

Studies on solids distribution in manifolds are sparse, especially for systems with a horizontal approach. Haynes and Gray (1974) studied particle transport in a manifold with a vertical approach. Sand-water slurries were tested in a 4-inch vertical downwards flow with branches of 1/4 and 1/2-inch diameter. They found that the transport efficiency varied inversely with solids concentration and particle size and increased with the branch flow rate. A similar study was conducted by Gruesbeck and Collins (1982). However, unlike Haynes and Gray, Gruesbeck and Collins found that the transport efficiency was independent of solids concentration in the main pipe.

Segev and Kern (1985) were the first to study solid-liquid separation in a horizontal slurry manifold. A variable diameter manifold was designed to ensure a constant slurry velocity in the header. The separation ratio of branch i in the manifold was defined as:

$$B_i = X_{bi}/X_{hi} \qquad [3]$$

where X_{bi} is the solids mass fraction in branch i and X_{hi} is the solids mass fraction in the header upstream of the branch. They developed the following semi-empirical correlation for the separation ratio in terms of the particle inertia parameter and the vertical solids concentration profile upstream of the branch:

$$B_i = 0.875\ \alpha\ C_{Ri}^{0.356} / K_i^{0.05} \qquad [4]$$

where α is characteristic of the sharpness of the branch corner. $C_{Ri} = c_{hi}(\text{centre})/C_{hi}$ for the side orientation and $c_{hi}(\text{bottom})/C_{hi}$ for the downwards orientation. c_{hi}(centre) and c_{hi}(bottom) are the solids concentration at the centre and bottom of the header upstream of branch i, respectively. These concentrations were predicted using a modified diffusion-type model originally developed by Karabelas (1977). K_i is the particle inertia parameter for branch i and is defined as:

$$K_i = \rho_s d_{50}^2 U_{hi} / 18\mu_f D_{bi} \qquad [5]$$

The objectives of the present study are: (1) to investigate the effects of upstream conditions on the solids distribution in a linear manifold with a horizontal approach; (2) to determine the operating conditions which would give equal solids distribution between the branches; and (3) to develop semi-empirical correlations to predict solids concentration in the branches with known upstream conditions.

Particle Equation of Motion

The linear momentum equation for solids is given by (Wallis, 1969)

$$\rho_s \mathbf{a}_s = - \nabla P + \rho_s \mathbf{g} + \mathbf{f}_{sf} + \mathbf{f}_{sw} \qquad [6]$$

where $\mathbf{a}_s$ is the particle acceleration, P is the fluid pressure, $\mathbf{f}_{sf}$ is the drag force of the fluid on the solids, $\mathbf{f}_{sw}$ is the wall friction force on the solids and $\mathbf{g}$ is gravitational acceleration.

Using a Lagrangian frame of reference, neglecting the effects of the solids on the fluid flow and wall friction, the momentum equation for the solids phase ahead of branch i can be written in a normalized form as:

$$d\mathbf{v}_s'/d\tau_i = - \rho_f \nabla_i' P_i' / 2\rho_s + \mathbf{g} D_{bi} / U_{hi}^2 + C_{DS} Re_{phi} |\mathbf{v}_f' - \mathbf{v}_s'| \, (\mathbf{v}_f' - \mathbf{v}_s') / 24K_i (1-c)^n \qquad [7]$$

where $\mathbf{v}_s$ and $\mathbf{v}_f$ are the local particle and fluid velocities, Re_{phi} is the particle Reynolds number based on the header bulk velocity and n is the Richardson and Zaki (1954) exponent for hindered settling. Other parameters are defined in the nomenclature. Nasr-El-Din and Shook (1986) integrated Equation (7) using a two-dimensional potential flow for the mixture to obtain the branch concentration ratio for a tee with a vertical approach. They found that the model predictions significantly over-estimated their experimental measurements. One of the reasons for this discrepancy was the secondary flow generated in the branch (Iwanami and Suu, 1969a,b) which was not included in their analysis.

From Equation (7), two limits for the branch concentration ratio can be inferred. When K_i approaches infinity (i.e., coarse particles having ρ_s/ρ_f >>

1), the branch concentration ratio approaches zero. This is because the inertial forces dominate, and a solids particle cannot follow the fluid into the branch. When K_i approaches zero (i.e., fine particles having $\rho_s/\rho_f \cong 1$), the branch concentration ratio approaches unity for a uniform upstream solids concentration profile. This is because the inertial forces are negligible and the particle trajectories coincide with the fluid streamlines.

Due to the complex nature of the flow into side branches, it is very difficult to theoretically determine the branch concentration ratio. In this study the following semi-empirical correlation is used to predict the concentration ratio for the branches in a manifold having a horizontal approach.

$$S_i = C_{bi}/C_{hi} = a_1 \, C_{Ri}^{a_2} \, Q_{Ri}^{a_3} \, \exp(a_4 K_i) \qquad [8]$$

where S_i is the separation ratio of branch i and a_1 to a_4 are empirical constants. C_{Ri} = c_{hi}(top)/C_{hi} for the upwards orientation; c_{hi}(centre)/C_{hi} for the side orientation; and c_{hi}(bottom)/C_{hi} for the downwards orientation. C_{hi} is the average solids concentration in the header upstream of branch i. Q_{Ri} is the ratio of the branch flow rate to the header flow rate, i.e., Q_{bi}/Q_{hi}. K_i is the particle inertia parameter for branch i, defined in Equation (5).

Equation (8) is similar to that of Segev and Kern (Equation 4). However, there are two differences between the two equations. Equation (4) was developed for a branch to main pipe flow ratio, Q_{bi}/Q_M, of 0.25. This is presumably the reason for not having the branch flow ratio in their equation. However, examining the flow in a manifold with a constant velocity, where the volumetric flow into each branch is 0.25 that of the main pipe, it can be seen that the flow ratio for various branches are as follows: Q_{b1}/Q_{h1} = 0.25,

Q_{b2}/Q_{h2} = 0.33, Q_{b3}/Q_{h3} = 0.50. Previous experimental studies have shown that the branch concentration ratio is a strong function of the branch flow ratio. To account for the dependence of the branch concentration ratio on the flow ratio, Q_{Ri} was included in Equation (8). For negative values for a_4, Equation (8) takes into account two limiting cases of the branch concentration ratio, namely: as K_i approaches infinity, S_i approaches zero, and as K_i approaches zero, S_i approaches a value that is dependent on Q_{Ri} and C_{Ri}.

Experimental Studies

Figure 1 shows a schematic of the closed loop flow system used in the experiments. The loop consisted of a three-branch manifold, a stainless steel feed tank stirred with a variable speed 1 hp stirrer, a 20 hp variable speed Moyno pump, a double pipe heat exchanger, a 2-in. Foxboro magnetic flowmeter (Model 8002-WCR-AG) for the main pipe, and three 1-in. Foxboro magnetic flow meters (Model 8001-WCR-AG) for the branches. The transparent sections before and after the first branch allowed for visual observation. To minimize the effect of secondary flow generated by an upstream elbow (Sharp and O'Neil, 1971; Nasr-El-Din and Shook, 1987), the first branch was located 80 pipe diameters downstream of the elbow. This allowance exceeded the minimum of 50 pipe diameters recommended by Colwell and Shook (1988). Valves were located in the loop to allow for the slurry to flow clockwise or counter-clockwise around the loop. The manifold was rotatable to allow for three branch orientations to be examined. The magnetic flowmeters were wired with a switch to record flow in any direction through the branches.

The manifold was designed to maintain approximately a constant velocity upstream of each branch (see Segev and Kern, 1985). The header ahead of the three branches were 2, 1.5 and 1.25-in dia., respectively. The flow rate through the branches was adjusted such that the flow in the main pipe was equally divided between the branches, i.e., $Q_{bi}/Q_M = 0.25$. The header velocity ratio (defined as the bulk velocity in the header to that in the main pipe, U_{hi}/U_M) that corresponds to these conditions were: 1.03, 1.27 and 1.15, for branch number 1, 2 and 3, respectively. It should be mentioned that the branch flow ratios for these conditions were: $Q_{b1}/Q_{h1} = 0.25$, $Q_{b2}/Q_{h2} = 0.33$, and $Q_{b3}/Q_{h3} = 0.50$. The fourth branch was a return line and obviously had a branch flow ratio, Q_{b4}/Q_{h4}, of unity.

To provide a range of particle properties, solid polyvinyl chloride particles (PVC) and sands were used in the experiments. The continuous phase used with the sand particles was tap water, whereas because of their low water wettability, a 10% triethylene glycol solution together with a few drops of Triton X-100 (a non-ionic surfactant) was used with the PVC particles. The properties of these phases and the range of parameters examined are listed in Table 1. To ensure fully suspended slurries, the lowest main pipe bulk velocity examined for the sand particles was greater than 1.8 m/s. The loop was maintained at a constant temperature of 25 $^{\circ}$C using the heat exchanger.

The loop was run until steady state was reached. Solids concentration in the branches were measured isokinetically using the technique outlined by Nasr-El-Din et al. (1984). A 4.4 mm inner diameter tapered probe was used to eliminate particle bouncing effect. A peristaltic pump was used to collect samples at predetermined velocities. Samples from the main pipe were withdrawn from the discharge above the feed tank. Solids concentration in each branch and in the main pipe were evaluated from the weight and volume of each sample. Since the particle velocity in a header is in general less than the fluid velocity, the discharge solids concentration is higher than the average in-situ concentration. Material balance closure performed around the branches and the main pipe agreed to within ± 3%.

RESULTS AND DISCUSSION

Branch transport efficiency, E_i

Before examining the experimental results, it is useful to discuss some relations regarding the branch transport efficiency. According to the definition of E_i, the sum of the branch transport efficiency for the manifold should be equal to unity, i.e.,

$$\sum_{i=1}^{4} E_i = 1 \qquad [9]$$

The manifold examined in this study was operated such that the main flow was equally divided among the four branches. Consequently, for a uniform solids distribution the branch transport efficiency should equal to 0.25 for all the branches. Solids maldistribution and lower manifold efficiencies occur if E_i deviates from 0.25.

Figure 2 shows the average solids concentration in the headers for a typical run for the upwards, side and downwards orientations. For given conditions (solids concentration, particle size, bulk velocity and branch flow ratio) it is obvious that a branch with an upwards orientation will receive less solids than that having a downwards orientation.

For the upwards orientation, a branch solids concentration is less than that in the header. Consequently, the solids concentration downstream of the first branch, i.e., the header of the second branch will have a higher solids concentration than that in the main pipe. This would then have the effect of having more solids reporting to the second branch. Similarly, the header of the third branch will have higher solids concentration than the second header, leading once again to a higher solids concentration in the third branch. The opposite trend would occur for the case of downwards branch orientation. It is clear then that an upstream branch affects all the downstream branches. This fact is very important in the design of a linear manifold with a horizontal approach.

Figure 3 shows the variation of the transport efficiency, $C_{bi}Q_{bi}/C_MQ_M$, with the main pipe bulk velocity for the upwards orientation at C_M = 12% and d_{50} = 0.29 mm. At a main pipe bulk velocity of 2.6 m/s, the branch transport efficiency for the first branch is very low, and it increases with the branch number. As was mentioned earlier, the header solids concentration increases along the manifold for the upstream orientation. Moreover, the branch flow ratio Q_{bi}/Q_{hi} increases from 0.25 to 0.5 for branches 1 to 3. These two effects cause more solids to report to a downstream branch. Previous studies with a simple tee and for the same orientation (Nasr-El-Din et al., 1989b) showed similar trends. Figure 3 also shows that a high bulk velocity (4 m/s), the transport efficiencies are closer to 0.25, i.e., a more uniform solids distribution. This is due to the fact that at higher slurry bulk velocities a more uniform solids concentration profile is maintained within a header, thus leading to a higher solids concentration at the top of the pipe.

Figure 4 shows the effect of the main pipe bulk velocity on the transport efficiency for the side orientation for the same conditions shown in Figure 3. Very even solids distribution is observed for the side orientation for all main pipe bulk velocities examined. This is similar to the previous results obtained with a simple tee having a horizontal approach (Nasr-El-Din et al., 1989b).

Figures 5 and 6 show the effects of particle size on the transport efficiency for the upwards and downwards orientations at C_M = 12 % and U_M = 3.2 m/s, respectively. Increasing the particle size has a strong effect on the transport efficiency for both orientations. Fine particles (d_{50} = 0.08 mm) are evenly distributed for all the branches. Figures 5 and 6 also show that the transport efficiency for the first branch is a strong function of the sand particle size. For the sand fraction having d_{50} = 0.39 mm, E_1 = 0.02 for the upwards orientation and 0.65 for the downwards orientation. Such a large deviation from 0.25 is due to two reasons. First, the vertical solids concentration for d_{50} = 0.39 mm would be very skewed, with relatively high solids concentration in the lower half of the pipe. Secondly, the inertia parameter K_i is fairly high for the coarse sand fraction (it is dependent on d_{50}^2 as shown in Equation 5). A large K_i value would tend to make a solid particle **not** to follow a fluid streamline entering a branch. For the upwards orientation, E_4 is higher than 0.25. Such a trend is dictated by the solids mass conservation where the sum of the branches transport efficiencies has to be equal to unity (see Equation 9). Similar arguments can be put forward for the trends obtained with the downwards orientation.

Figure 7 shows the variation of the transport efficiency with the branch number for the PVC particles at a main pipe bulk velocity of 0.8 m/s and 1.2 m/s. The transport efficiency for all branches and orientations is close to 0.25. These trends are reasonable and can be explained as follows: the density of the PVC particles is very close to the fluid density (see Table 1). This implies a uniform vertical solids concentration ahead of the branches and a smaller particle inertia parameter. Both factors enhance a uniform solids distribution between the branches. The results shown in Figure 7 also indicate that for such a slurry system it is possible to obtain a uniform solids distribution even at a low main pipe bulk velocity of 0.8 m/s.

Prediction of a Branch Concentration Ratio, S_i

The vertical solids concentration profile ahead of each branch was first calculated following the procedure of Foster et al. (1991). C_{Ri} values (top, side and bottom) for each branch were then determined. Q_{Ri} values for branch number 1, 2, and 3 were kept constant at Q_{b1}/Q_{h1} = 0.25, Q_{b2}/Q_{h2} = 0.33, and Q_{b3}/Q_{h3} = 0.50, respectively. The particle inertia parameter, K_i, was calculated for each branch and flow condition.

C_{Ri}, Q_{Ri} and K_i values for each branch for various particle size, solids concentration, main pipe bulk velocity and particle density were used to determine the empirical constants in Equation (8) using an IMSL-double precision multi-regression program. Table 2 lists the values of these constants for the three orientations. For all orientations, the constant a_4 is negative. This indicates that the branch concentration ratio diminishes as K_i increases, and it approaches zero for higher values of K_i. Also, as K_i approaches zero, S_i approaches a value, which is a function of C_{Ri} and Q_{Ri}. These trends agree with those inferred to from Equation (7). Most of the data points (75 per orientation) fall within ± 15% of the calculated values.

Conclusions

In the present study, solids distribution in the branches of a linear manifold having a horizontal approach was examined. The following results were obtained:

1. A uniform solids distribution was obtained with the PVC particles (ρ_s = 1180 kg/m^3) and with the fine sand fraction (d_{50} = 0.08 mm) for the three manifold orientations.

2. For the medium and coarse sand fractions, a uniform solids distribution was

obtained only for the side orientation.

3. For the three branch orientations, semi-empirical correlations were developed to predict the branch concentration ratio. The predicted and measured concentration ratios agreed within ± 15%.

Acknowledgements

The authors wish to thank Alberta Coal Research for their financial support. Mr. A. Afacan, Ms. J. Foster and Ms. N.Ruhl assisted with the experiments.

Nomenclature

a_1 to a_4	constants in the semi-empirical correlation (dimensionless)
$\mathbf{a}_s$	particle acceleration (m/s^2)
B_i	mass separation ratio of branch i (dimensionless)
c	local solids concentration (vol.%)
C	solids concentration averaged over the pipe cross section (vol.%)
C_{bi}	solids concentration in branch i (vol.%)
C_{hi}	solids concentration in the header upstream of branch i (vol.%)
C_M	discharge solids concentration (vol.%)
C_{DS}	drag coefficient (dimensionless)
C_{Ri}	concentration ratio of point to bulk (dimensionless)
D	pipe inside diameter (m)
D_{bi}	inside diameter of branch i (m)
D_{bi}	inside diameter of the header upstream of branch i (m)
d_{50}	particle mean diameter (m)
E_i	transport efficiency of branch i (dimensionless)
$\mathbf{f}_{sf}$	drag force of fluid on solids (N/m^3)
$\mathbf{f}_{sw}$	wall friction force on solids (N/m^3)
$\mathbf{g}$	gravitational acceleration (m/s^2)
K_i	particle inertia parameter for branch i, $\rho_s d_{50}^2 U_{hi} / 18\mu_f D_{bi}$
n	Richardson and Zaki exponent (dimensionless)
P	fluid pressure (Pa)
P'	fluid pressure, $P/(0.5\rho_f U_{hi}^2)$ (dimensionless)
Q	volumetric flow rate (m^3/s)
Q_{Ri}	flow ratio of branch i, Q_{bi}/Q_{hi}
Re_{phi}	particle Reynolds number, $\rho_f d_{50} U_{hi} / \mu_f$
S_i	separation ratio of branch i (dimensionless)

t	time (s)
U_M	bulk velocity in the main pipe (m/s)
U_{hi}	bulk velocity in the header ahead of branch i (m/s)
v	local velocity (m/s)
v'	normalized local velocity, v/U_{hi}
X	solids mass fraction (dimensionless)

Greek Symbols

α	constant characteristic of branch corner (dimensionless)
∇	gradient, (m^{-1})
∇'	gradient, $D_i\ \nabla$ (dimensionless)
μ	viscosity (Pa.s)
ρ	density (kg/m^3)
τ_i	characteristic time for branch i, $t\ U_{hi}/D_{bi}$ (dimensionless)

Subscripts

bi	branch i
f	fluid
i	branch number (i = 1,2,3)
M	main pipe
hi	header immediately upstream of branch i
s	solids

Superscripts

′	dimensionless

References

Bugliarello, G. and G.C.C. Hsiao,"Phase Separation in Suspensions Flowing Through Bifurcations: A Simplified Hemodynamics Model", Science **143**, 469-471 (1964).

Colwell, J.M. and C.A. Shook,"The Entry Length for Slurries in Horizontal Pipeline Flow", Can. J. Chem. Eng. **66**, 714-720 (1988).

Foster, J., H.A. Nasr-El-Din and J.H. Masliyah,"Solids Distribution in Slurry Flow in a Linear Manifold with a Horizontal Approach", Can. J. Chem. Eng., in press (1991).

Gruesbeck, C. and R.E. Collins, "Particle Transport Through Perforations", Soc. Pet. Eng. J. **273**, 857-865 (1982).

Haynes, C.D. and K.E. Gray,"Sand Particle Transport in Perforated Casing", J. Pet. Tech. **26**, 80-84 (1974).

Iwanami, S. and T. Suu,"Study on Flow Characteristics in Right-Angled Pipe Fittings: 1st Report on Case of Water Flow", Bull. JSME **12**, 1043-1050 (1969a).

Iwanami, S. and T. Suu,"Study on Flow Characteristics in Right-Angled Pipe Fittings: 2nd Report on Case of Slurries in Hold-up Flow", Bull. JSME **12**, 1051-1061 (1969b).

Karabelas, A.J.,"Vertical Distribution of Dilute Suspensions in Turbulent Pipe Flow", AIChE J. **23**, 426-434 (1977).

Nasr-El-Din, H., C.A. Shook and M.N. Esmail,"Isokinetic Probe Sampling from Slurry Pipelines", Can. J. Chem. Eng. **62**, 179-185 (1984).

Nasr-El-Din, H., C.A. Shook and M.N. Esmail,"Wall Sampling in Slurry Systems", Can. J. Chem. Eng. **63**, 746-753 (1985).

Nasr-El-Din, H. and C.A. Shook,"Particle Segregation in Slurry Flow Through Vertical Tees", Int. J. Multiphase Flow **12**, 427-443 (1986).

Nasr-El-Din, H. and C.A. Shook,"Effect of a 90 Degree Bend on Slurry Velocity and Concentration Distributions", J. Pipelines **6**, 239-252 (1987).

Nasr-El-Din, H., A. Afacan and J. Masliyah,"Wall Sampling from Horizontal Slurry Pipelines", Chem. Eng. Commun. **82**, 203-221 (1989a).

Nasr-El-Din, H., A. Afacan and J.H. Masliyah,"Solids Segregation in Slurry Flow Through a T-Junction With a Horizontal Approach", Int. J. Multiphase Flow **15**, 659-671 (1989b).

Richardson, J.F. and W.N. Zaki,"Sedimentation and Fluidization", Trans. Inst. Chem. Eng. **32**, 35-52 (1954).

Segev, A. and K.C. Kern,"Solid-Liquid Separation in a Slurry Manifold", Presented at the ASME winter meeting, Synfuels and Coal Energy Symposium, Dallas, TX, USA, Feb. 17-21 (1985).

Sharp, B.B. and I.C. O'Neil,"Lateral Diffusion of Large Particles in Turbulent Pipe Flow", J. Fluid Mech. **45**, 575-584 (1971).

Torrest, R.S. and R.W. Savage,"Particle Collection from Vertical Suspension Flows Through Small Side Ports: A Correlation", Can. J. Chem. Eng. **53**, 699-701 (1975).

Wallis, G.B.,"**One-dimensional Two-phase Flow**", McGraw-Hill, New York (1969).

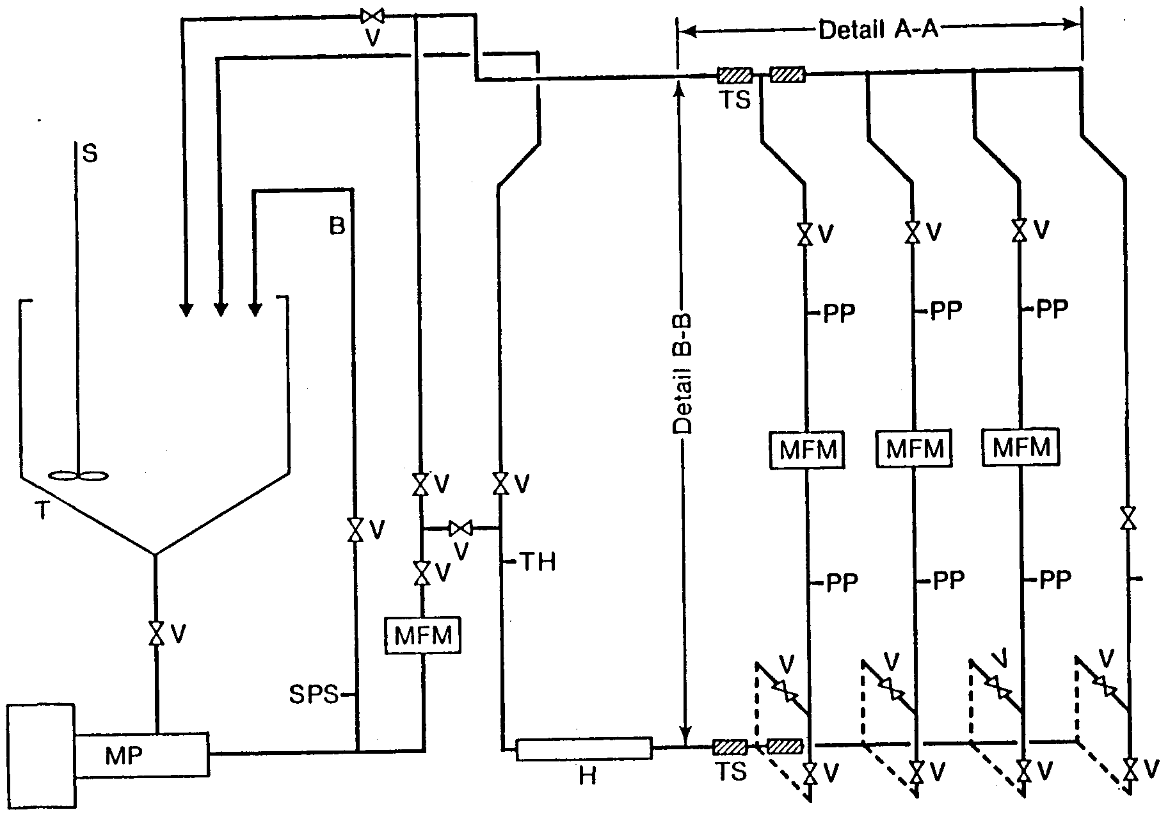
Table 1: Particles and fluids properties* with examined parameters.

parameter	symbol	units	range
sand:			
particle density	ρ_s	kg/m^3	2630
mean particle size	d_{50}	mm	0.08, 0.29, 0.39
solids concentration	C_M	vol.%	6 - 20
bulk velocity	U_M	m/s	1.8 - 4.0
PVC:			
particle density	ρ_s	kg/m^3	1180
mean particle size	d_{50}	mm	0.15
solids concentration	C_M	vol.%	12 - 18
bulk velocity	U_M	m/s	0.8 - 1.2
fluid viscosiy	μ_f	mPa.s	1.265
fluid density	ρ_f	kg/m^3	1008

* measured at 25 °C.

Table 2: Empirical constants for Equation (8).

Orientation	a_1	a_2	a_3	a_4
Upwards	1.3043	0.3838	0.1772	- 0.056
Side	0.9582	- 0.0126	- 0.02999	- 0.021
Downwards	0.6645	0.2955	- 0.3335	- 0.0053



B	System Bypass Line	S	Stirrer
H	Double Pipe Heat Exchanger	T	Feed Tank
MFM	Foxboro Magnetic Flow Meter	TH	Thermometer
MP	Moyno Pump	TS	Transparent Section (Rotatable)
PP	Probe Port for Isokinetic Sampling	V	Valve
SPS	Safety Pressure Switch		

Figure 1 Experimental set-up

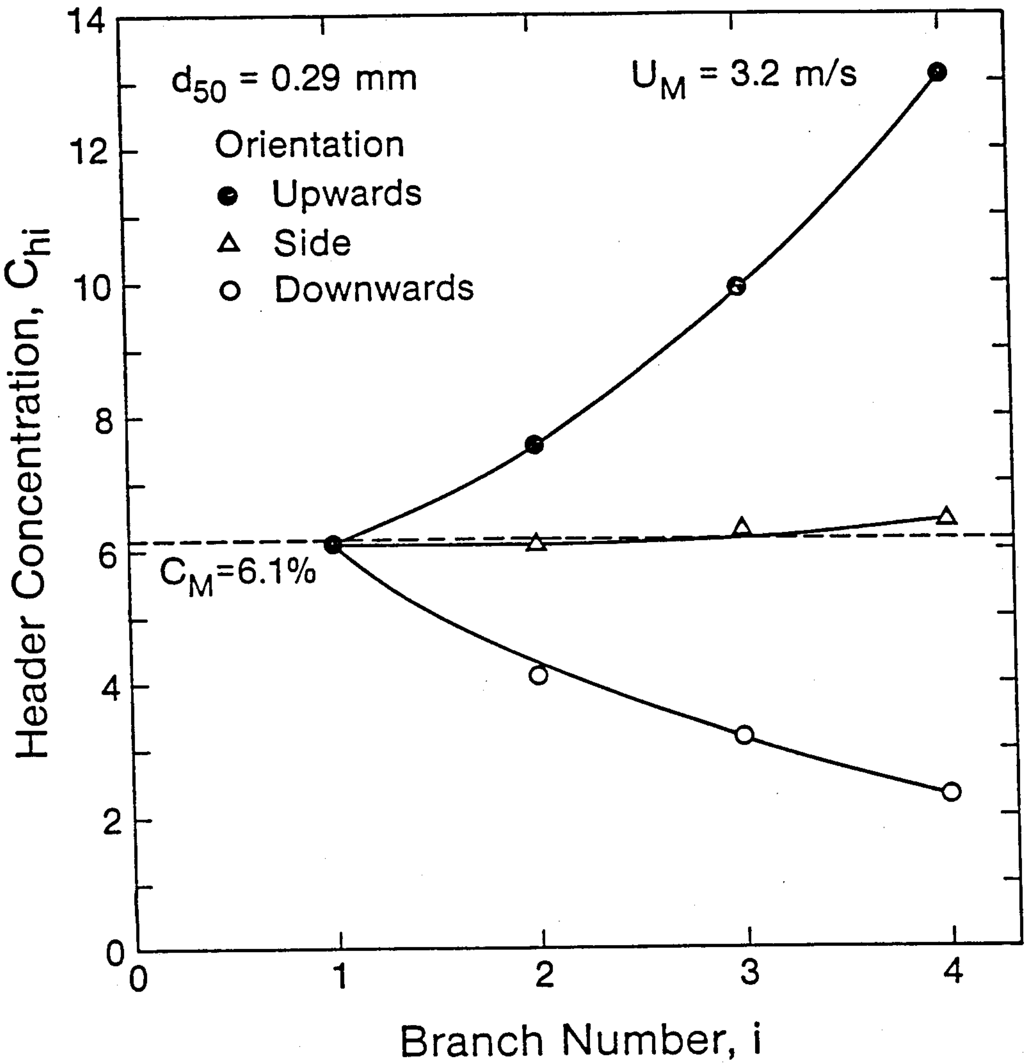


Figure 2 Variation of the solids concentration in the headers of the three orientations, sand particles, d_{50}= 0.29 mm.

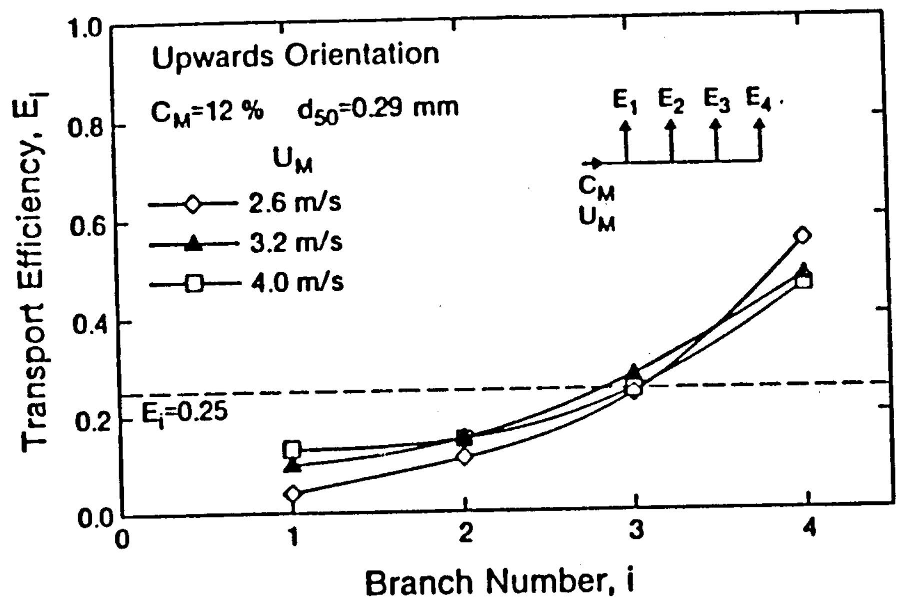


Figure 3 Variation of the transport efficiency with the main pipe bulk velocity for the upwards orientation, sand particles, d_{50}= 0.29 mm.

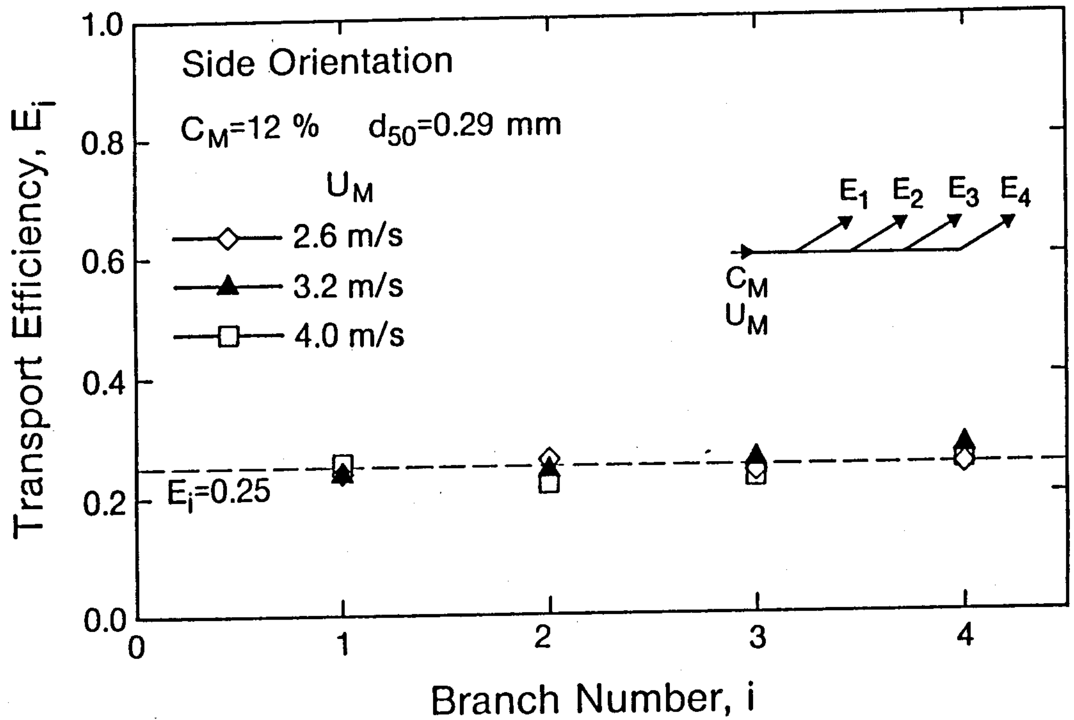


Figure 4 Variation of the transport efficiency with the main pipe bulk velocity for the side orientation, sand particles, d_{50} = 0.29 mm.

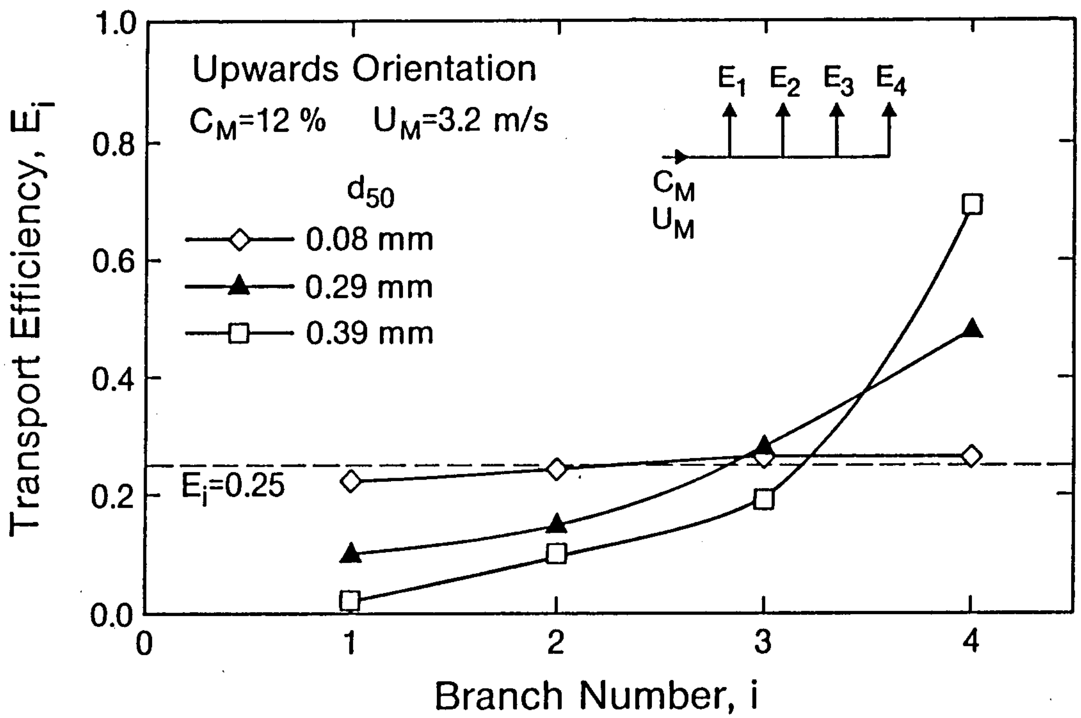


Figure 5 Variation of the transport efficiency with particle mean diameter for the upwards orientation, sand particles.

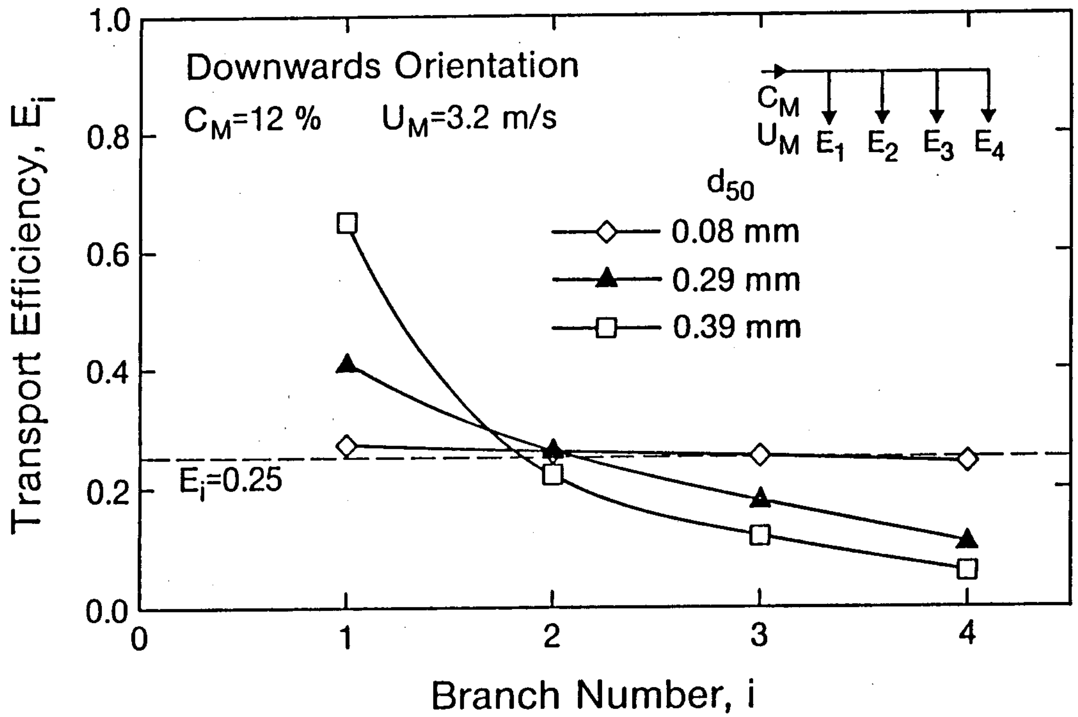


Figure 6 Variation of the transport efficiency with particle mean diameter for the downwards orientation, sand particles.

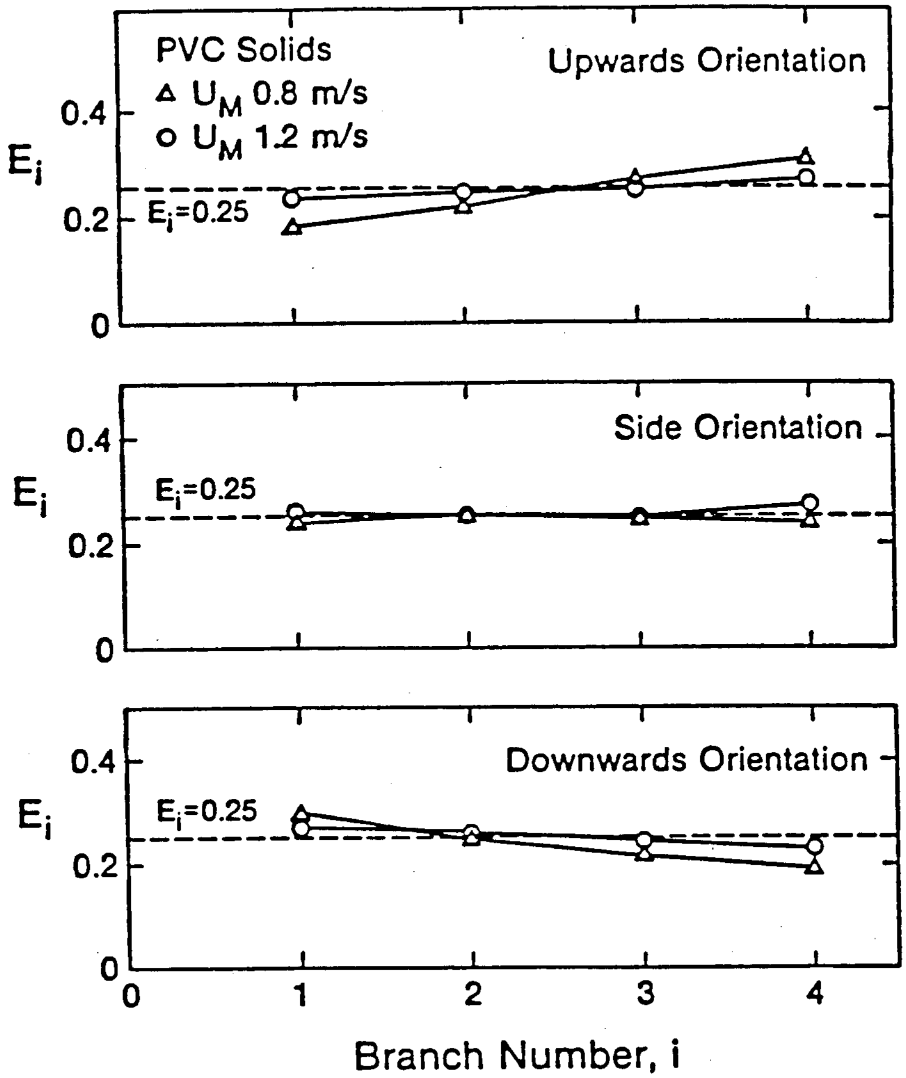


Figure 7 Variation of the transport efficiency for the PVC particles for the three orientations.

1991 Alberta Coal Research Program
Contractor's Conference
October 30-31, 1991

PIPELINE TRANSPORTATION OF ALBERTA COALS AS LIQUID HYDROCARBON SLURRIES

Ted Frankiewicz — Unocal Science & Technology
Glen Perry — Unocal Canada

ABSTRACT

Technology has been developed for the preparation of coal slurries using either light oil or natural gas liquids as carriers. These 50% by weight coal slurries can be transported over long distances via existing crude oil batch or dedicated pipelines.

Using data from pilot scale tests, a process design and cost study was conducted for the preparation and separation of slurries containing approximately 2 MM tonnes/yr. of coal. The process cost numbers were used in an economic analysis and the costs for transporting Alberta coal to Ontario or to Cold Lake by pipeline were estimated.

Results indicate that pipeline transport of coal to Ontario or to Cold Lake can be cheaper than the corresponding rail transport. However, while pipeline transport is cheaper than rail, the use of coal as a fuel at Cold Lake is not likely until natural gas prices in the field increase to above $2.20/MM BTU (1990 Cdn $).

INTRODUCTION AND BACKGROUND

Large reserves of low-sulfur coal exist in western Canada. Alberta's in-place, surface recoverable reserves, for example, exceed 11 gigatonnes. Much of this is lower rank coal, however, and the high cost of transporting the coal to markets has limited the development of this vast Alberta resource. Distance to market, high ash content, and relatively high moisture in Alberta's sub-bituminous coals contribute to the high transportation costs.

Several Alberta coal deposits and large potential users of Alberta coals are only short distances from existing oil or NGL liquid (condensate) pipelines, Figures 1 and 2. These pipelines could carry coal as oil or condensate slurries if technology existed to prepare the slurries at the mine mouth and to separate the coal from the carrier near the consumer.

To develop the required slurry technology, Unocal Canada, Alberta's Office of Coal Research and Technology, the Province of Ontario, and the Canadian Federal Western Economic Diversification Office co-funded the transCOM and Coal-Condensate slurry projects.

Since Unocal Canada owned the Obed Coal Mine while these programs were active, this coal served as the model for developing the slurry preparation and separation technologies. Analytical data on Obed coal is summarized in Table 1. For use in transCOM and condensate slurries, the coal was ground to 60% -200 mesh, the size to which Obed is commercially ground for utility boiler firing.

Table 2 summarizes the physical properties of the light, sweet Alberta crudes and the NGL condensate used in slurry development and testing.

A generalized flow diagram for the transCOM slurry transportation concept is shown in Figure 3. The coal, after grinding, is dispersed in its carrier liquid along with any additives which may be required. The coal-liquid dispersion is then subjected to a second, proprietary mixing step prior to either being stored or injected into a pipeline for transport to the user's location. At the user location, the slurry is recovered from the pipeline and sent to intermediate storage.

Slurry from the storage tanks is separated into its constituent coal and oil by a sequence of centrifuging and drying steps, some details of which are also proprietary. Although most of the oil and/or condensate is recovered from the coal by centrifuging, additional thermal drying is required to improve both the handling and the safety characteristics of the recovered coal.

Drying the coal product has the added benefit of delivering a lower moisture coal to the customer. Together with a small residual hydrocarbon component, this results in a higher energy content coal than he would have received by rail transport. In addition, coal losses during pipeline transport are very low (1). Thus for a Unit Train batch of coal, approximately 10,000 tonnes, 200 - 300 tonnes more coal would arrive at the user site than would be the case with rail transport (hopper car dust and handling losses).

Technical details on the laboratory research, pipeloop circulation tests, field pipeline testing, and pilot-scale, continuous preparation and separation of transCOM and coal-condensate slurries have been presented (1 - 5).

COAL SLURRY PROCESS DEVELOPMENT AND PLANT DESIGN

During the pilot plant phase of the coal slurry programs, the operability of the more important process steps for slurry preparation and separation were demonstrated. Sufficient operating data were acquired that an independent engineering firm (Bantrel Engineering of Calgary and their parent firm, Bechtel Engineering in San Francisco) could design, size, and cost estimate transCOM and coal-condensate slurry preparation and separation plants.

Data from the pilot plant testing, conducted by the Alberta Research Council at their Nisku facility, and from off-site vendor tests were supplied to Bantrel/Bechtel for use in their process design and evaluation study. Included in the information supplied to Bantrel were

* physical characteristics of all process feed materials
* desired physical characteristics of all process products
* performance data for major process equipment
* suggested vendors for lesser equipment such as filters and settlers
* a system operating factor of 0.95

As prepared coal-oil (transCOM) slurries were to contain 49.8% by weight coal while the coal-condensate slurries were to contain 51.8% coal.

Material balances for the slurry separations were calculated for the engineering study using a process simulation spreadsheet developed by Unocal which included all process recycle streams. The data supplied to Bantrel/Bechtel were not optimized, however, and it was later shown that significant opportunities existed for improving process performance and reducing costs (6).

As specified by Unocal, the slurry preparation and separation plants were designed to receive 250 tonnes/hr of 25 mm X 0 Obed coal with 18% moisture (6% over inherent) directly from the coal cleaning jigs, thereby bypassing the kiln dryers at the Obed mine. Oil or condensate used in slurry preparation is as described in Table 2. Slurry preparation consists of pulverizing to a utility particle size distribution, dispersing the coal in the carrier liquid, final slurry preparation, and pre-shipment storage facilities.

Separation includes slurry centrifuging, centrifuge cake drying, centrate cleaning to reduce final solids content of the recovered oil, and oil/water separation of vapors recovered from the dryer. Where appropriate, wet solids recovered from centrate cleaning and dryer vapor recovery were recycled back to the incoming slurry for re-centrifuging. Coal solids recovered from oil slurries were sufficiently warm that post-dryer coal cooling was required. Such cooling was deemed unnecessary for coal being recovered from condensate slurries.

For the base case study, oil losses to the coal were assumed to be 5%. Condensate losses were set at 0.5%. Both numbers are supported by the pilot plant data, but may be conservative since oil and condensate losses were substantially lower under some operating conditions. Process optimization studies on the spreadsheet model also suggested that carrier liquid losses could be reduced with minor changes in operating conditions (6).

COAL SLURRY PROCESS ECONOMICS

Preparation and Separation Plant Costs

Capital cost estimates for the facilities were prepared by Bantrel in second quarter 1990 Canadian dollars. Table 3 summarizes the capital costs for both the coal-oil and the coal-condensate slurry plants. For preparation, coal-oil slurry costs are higher because more intermediate storage is required for accumulating slurry before injecting it into a batch pipeline. Coal-oil slurry separation costs are higher because of increased storage requirements, slower centrifuging rates, and more complex processing to insure that clean, solids-free oil leaves the separation plant.

For the oil slurry preparation plant, direct equipment costs were $23.7 MM. To this were added costs for other materials and systems such as piping, steam, electrical, process control, etc., totaling $14.4 MM or 60% of equipment costs. The preparation plant building cost was $6.4 MM. Direct labor cost was $6.2 MM based on equipment cost factors using 5 man-hours per $1000 of equipment.

Also added to the preparation plant cost were indirects (temporary facilities, site office, warehouse, consumables, construction equipment, etc.) at 90% of the direct labor cost, home office engineering at 10% of the total plant cost, and a $12.4 MM or 20% contingency. An off-site cost for water, fire, sewer, administration, etc., added $7.4 MM.

In each case, the capital cost for the slurry preparation plants includes five Combustion Engineering Model 1023 bowl mills. For the oil slurry plant, these constitute 61% of the process equipment costs, $23.7 MM Cdn. However, CE guidelines for sizing these mills (7), confirmed by pilot scale grinding tests for Obed coal (6,8), indicated that only four such mills would be required, This saves some $2.9 MM in direct material costs and reduces total preparation plant costs by about 12%.

Additional preparation plant cost savings could also be effected by reducing the size of the building specified by Bantrel/Bechtel. In total, process improvements and optimization could reduce the capital cost estimate for slurry preparation by 30%.

For the coal-oil slurry separation plant, major equipment costs were calculated to be $36.9 MM. Materials, labor, indirect, home office costs, off-site costs, and a 20% contingency were then added to the equipment costs in the same manner as described for the preparation plants.

It should be noted that whereas the transCOM separation plant was designed and cost-estimated as a single, large facility, the coal-condensate slurry separation plant was designed as three stand alone facilities which could be sited where needed at, for example, the Esso Resources Cold Lake heavy oil field. The cost for one stand alone unit was determined and then multiplied by three to arrive at the final cost. Obvious savings on the design and engineering costs for the 2nd and 3rd units were not taken in this study and represents a significant potential cost reduction.

Annualized operating cost estimates for the two slurry plant cases are shown in Table 4. Note that operating costs for oil and condensate slurry preparation are identical. However, the operating costs for slurry separation are significantly higher when oil is used as the carrier liquid. The higher cost for separating oil slurries results mainly from reduced centrifuging efficiency and the need to pre-heat the slurry prior to separation. Also, the centrifuge cake dryer operates at a higher temperature in the transCOM oil slurry separation plant.

Slurry Transport Costs: Coal to Ontario

Using the appropriate capital and operating costs shown in Tables 3 and 4 along with the economic assumptions in Table 5, the present value (PV) for a project to deliver Alberta coal to Ontario via pipeline was estimated. For a project yielding a 10% real rate of return, the present value pre-tax was estimated to be $111.6 MM Cdn. This would provide the investors a net cash flow after taxes of $30.9 MM Cdn (9).

The transportation cost for delivering Alberta coal to Ontario via pipeline is estimated on this basis to be $36.90/tonne. However, by reducing capital costs by 30% as discussed above, reducing oil losses to the coal from 5% to 3%, and reducing additive costs by 50%, the cost for delivering Alberta coal to Ontario could be as low as $29.50/tonne (9). This compares with a current estimate of $40/tonne for delivering the same coal to Ontario by rail.

While the above deals with strictly economic issues, there are other major benefits associated with pipeline delivery of coal. These other benefits have economic impact to a varying degree depending on the specific power plant requirements. The benefits include:

1. Pre-grinding coal saves the power plant this cost.
2. Higher BTU value in the delivered coal minimizes boiler de-rating.
3. Open stockpiles of coal are eliminated.
4. Lower coal inventory requirements due to reduced interruption of deliveries from weather, strikes, etc.
5. Reduced land area requirements for power plant site.
6. Less front-end coal handling.
7. Reduced acid gas emissions from low sulfur Alberta coal.
8. Large potential cost reduction from economics of scale.
9. Access to multiple coal sources allows the easy blending of coals for optimum boiler performance.

Slurry Transport Costs: Coal to Cold Lake

Recently, a major study was conducted on the use of coal to provide steam for heavy oil production at Cold Lake. In this study (10), 1.674 MM tonnes/yr. of Highvale coal were determined to be sufficient to supply the energy needs for Phases 1 - 6 of the Esso field development project.

On a bone dry coal basis, the volume of coal required is 78.5% of that used in the original coal slurry design and cost engineering study. Accordingly, to obtain a better estimate for the cost of delivering the required coal to Cold Lake, the Bantrel capital and operating cost estimates were revised to reflect the lower coal throughput and a source of coal such as Highvale.

Table 6 shows the capital cost adjustments made to the Bantrel study. The cost numbers were also adjusted to reflect the need for four bowl mills instead of five, to capture the cost savings from the phased construction of three separation plants, and the reduced site costs due to the pre-existence of some site infrastructure.

Adjustments to the coal slurry plant operating costs were also made and these are summarized in Table 7. Labor and supply costs were held constant. The power and gas requirements were scaled with volume at 78.5%. However, the gas cost is a function of price and this is one of the variables in the economic calculations. Therefore, gas is treated as a fuel requirement and its cost is calculated with the computer model. Power costs were based on 3.9 cents per KWH in the Bantrel study, which is higher than the 2.8 cents paid by large industrial users in Alberta during early 1990. Thus power costs were adjusted by a factor of 0.785 for reduced coal volume and 0.72 for price. An owner's cost was then added at 2% of capital to cover municipal taxes, insurance, and overhead.

Using adjusted plant costs, the cost for conversion to coal combustion and for transporting Highvale coal to Cold Lake as a condensate slurry is shown in Table 8A.

At Cold Lake, coal competes with natural gas. To calculate a price at which coal is competitive with gas, a rate of return of 10% real rate of return (i.e., after taxes) was assumed. It was also assumed that transporting gas from the wellhead to Cold Lake costs $0.20/MM BTU. Highvale coal heating value was taken to be 18.04 MM BTU/tonne and that the coal cost at the mine mouth was set at $11/tonne. The 0.5% loss of condensate to the coal was retained. Table 8B shows that for coal to be competitive, the field value of gas in Alberta must be $2.21/MM BTU.

Although the calculated competitive price required for natural gas is substantially higher if the coal is transported by rail (10), the switch to the use of coal for raising steam in Cold Lake will not occur until gas prices rise from their current bargain basement levels. This study does illustrate, however, that when the switch is made, there will be a strong economic incentive to transport that coal by pipeline. The assumption that the displaced gas would find export markets implies large social benefits in the form of jobs, foreign exchange, and value added by displacing gas with coal.

SUMMARY AND CONCLUSIONS

The technology to prepare coal slurries using either light crudes or NGL condensate was developed. The slurries consist of approximately 50% by weight coal and 50% carrier liquid. The viscosity of these slurries is sufficiently low that transport through conventional pipelines over long distances is feasible. The continuous preparation and separation of slurries was demonstrated at the pilot plant level. A coal-oil (transCOM) slurry was also injected into and recovered from an operating light oil field pipeline.

Data obtained from the pilot scale tests were used to design and cost estimate commercial sized plants for preparing and separating the coal slurries using both condensate and light oil as the carrier liquid. The engineering study was conducted by Bantrel Engineering (Calgary) and Bechtel (San Francisco) using data and performance specifications supplied by Unocal.

An economic analysis using the plant costs calculated by Bantrel indicates the transporting of coal via existing crude oil pipelines from Alberta to customers in eastern Canada can be less expensive than the transporting of the same coal from and to the same locations by rail. In addition to showing the potential for cost savings, the pipeline transport option has other potential environmental and system benefits as well.

The transporting of coal such as Highvale to Cold Lake where it can be burned in place of natural gas for producing steam is also feasible. It appears, however, that a natural gas value of $2.21/ MM BTU will be required before coal will be competitive with natural gas combustion at Cold Lake.

Having completed the initial pilot scale test campaign and the engineering study, the transCOM and Coal-Condensate slurry projects are currently inactive. The next step toward commercializing this technology would be to conduct a continuous demonstration scale test involving equipment somewhat larger than that used in the pilot tests. The demonstration plant should be fully integrated and operated over a period of several months. This level of activity is justified only by the willingness of a coal supplier and a coal customer to fund a substantial fraction of the project's cost.

REFERENCES

1. T. Frankiewicz, G. Pangracs, C. A. Shook, R. G. Gillies, and M. Small, Pipeloop and Pipeline Flow Tests on Coal-Oil (transCOM) and Coal-Condensate Slurries, Proc. 16th Int. Conf. on Coal and Slurry Technology, Clearwater, FL, April 1991.

2. T. Frankiewicz, R. E. Miller, G. Pangracs, A. K. Chambers, and A. Turak, Pilot Scale Equipment Testing for the Continuous Production and Separation of Coal-Oil Slurries, Proc. 16th Int. Conf. on Coal and Slurry Technology, Clearwater, FL, April 1991.

3. T. Frankiewicz, R. E. Miller, G. Pangracs, R. G. Gillies, and M. Small, Laboratory and Vendor Scale Testing of Unit Operations for Separating Coal and Oil from Coal-Oil (transCOM) Slurries, Proc. 15th Int. Conf. on Coal and Slurry Technology, Clearwater, FL, April 1990.

4. T. Frankiewicz, G. Perry, R. G. Gillies, C. A. Shook, and M. Small, Pipeloop Studies on the Rheology of Coal-Oil (transCOM) Slurries Intended for Transport in Long Distance Pipelines, Proc. 15th Int. Conf. on Coal and Slurry Technology, Clearwater, FL, April 1990.

5. T. C. Frankiewicz and G. F. Perry, Transporting Coal in Long Distance Pipelines as an Oil Slurry, 39th Canadian Chemical Engineering Conference, Hamilton, Ontario, Canada, October 1989.

6. T. C. Frankiewicz, Unocal S & T Project Report 90-26: Coal-Condensate Slurry Phase I and transCOM Phase II, Volume I, October 1990.

7. C. E. Raymond Bulletin No. 813R, Raymond Bowl Mills, 1986.

8. T. C. Frankiewicz, Unocal S & T Project Report 89-2: transCOM Phase I, Volumes I and II, February 1989.

9. G. F. Perry, Unocal Canada Ltd. Project Report on transCOM Phase II: Logistics and Economics, October 1990.

10. G. F. Perry and References cited in Unocal Canada Ltd. Project Report on Coal-Condensate Slurry, Phases I and II: Logistics and Economics, October 1990.

TABLE 1: ULTIMATE AND PROXIMATE ANALYSES FOR OBED MT COAL USED IN transCOM SLURRY PIPELOOP TESTS.

	AS RECEIVED	MOISTURE FREE	MOISTURE/ASH FREE
MOISTURE	12.4%		
ASH	13.0	14.8%	
VOLATILE MATTER	32.1	36.6	43.0%
FIXED CARBON	42.5	48.5	57.0
C	57.7	65.9	77.3
H	3.9	4.5	5.3
N	1.23	1.4	1.7
S	0.64	0.74	0.86
O	11.1	12.7	14.9

TABLE 2: ANALYTICAL DATA ARE SUMMARIZED ON CRUDE OILS AND ON NATURAL GAS CONDENSATE USED TO MAKE COAL SLURRIES FOR PIPELOOP AND FIELD PIPELINE TESTS.

SAMPLE	PEACE RIVER	PEACE PIPELINE	KIDNEY CRUDE	NATURAL GAS CONDENSATE
VISCOSITY (cP)				
6°C	15.50	10.00	Non-Newt.	0.55
11°C	11.50	6.80	Non-Newt.	--
20°C	5.40	5.70	6.7 @ 15 C	0.49
30°C	3.90	4.30	--	--
35°C	3.00	3.40	--	0.44
40°C	2.80	3.10	3.01	--
Density (g/cc)				
15°C	0.835	0.831	0.838	0.728
40°C	0.822	0.811	--	--
Asphaltenes (%)	1.1	1.1	--	--
Toluene Insol (PPM)	17	27	--	--
Sulfur (%)	0.30	0.20	--	--
BS & W (%)	0.40	0.20	--	--
KF Water (%)	0.10	0.01	--	--

Table 3: CAPITAL COST SUMMARY FOR COAL SLURRY PREPARATION AND SEPARATION PLANTS.

	Coal-Condensate Slurry		Coal-Oil Slurry	
	Preparation	Separation	Preparation	Separation
Direct Cost	48.1	51.6	55.1	83.4
Indirect Cost	5.8	7.2	6.8	14.4
Total Field Cost	53.9	58.8	61.9	97.8
Home Office	5.4	5.7	6.2	9.8
Contingency (20%)	11.8	12.9	13.6	21.5
TOTAL	71.1	77.4	81.7	129.1

Note: Costs expressed in millions of Canadian dollars, 2nd quarter 1990 basis.

Table 4: OPERATING COST SUMMARY FOR COAL SLURRY PREPARATION AND SEPARATION PLANTS.

	Coal-Condensate Slurry		Coal-Oil Slurry	
	Preparation	Separation	Preparation	Separation
Power	1,977	1,174	1,977	2,305
Natural Gas	--	1,570	--	3,643
Labour	1,396	2,452	1,396	1,572
Supplies	622	559	622	1,255
Sub Total	3,995	5,755	3,995	8,775
Contingency (10%)	405	575	405	875
TOTAL	4,440	6,330	4,400	9,650

Note: Costs are in thousands of Canadian dollars/year, 2nd quarter 1990 basis.

TABLE 5: SUMMARY OF ECONOMIC ASSUMPTIONS USED IN THE ECONOMIC EVALUATION OF THE USE OF PIPELINES TO TRANSPORT COAL AS A HYDROCARBON SLURRY.

Design Basis:

* 2.29 Million short tons per year of 18% moisture coal produced, 2.15 Million tons of 12.5% moisture delivered to pipeline, 1.88 Million tons of bone dry coal delivered to Buyer.

* Storage - preparation plant: 125,000 Bbls oil feed
  300,000 Bbls slurry out
  - separation plant: 1,500,000 Bbls slurry feed

* Two stage centrifuging, drying and cooling of coal.

* Coal stored as slurry, fed from dryer/cooler direct to burner.

* Coal product is 95% coal, 5% oil, 0% water. (Hydro test has about 3% H_2O).

* For Edson: Oil delivered by Trans Mountain, at a tariff based on current cost of $0.25/Bbl, at a throughput of 150,000 BPD. This tariff scales with volume (i.e., as the slurry preparation plant requires 41,000 BPD of oil, the tariff reduces to (150/191 times 25 cents) or 19.6 cents per barrel. A new 12 inch pipeline is built from Edson to Edmonton for slurry delivery.

Economic Parameters

1) Oil On Coal Penalty of 5% per weight of bone dry coal:
   - Oil Price $22.50/Bbl (Cdn)
   - Coal price delivered $2.50/MMbtu + $.15/MMbtu operating credits.
   - Oil on Coal Penalty - 5% oil
     = (5/95) X (39 MMBTU/t) X ((22.50/5.64) - (2.65))
     = $2.75/ton of bone dry coal or $5.2 MM/yr

2) Grinding, handling, inventory operating cost savings $3.25/ton of raw coal -$3.25 X 2.29 MMT = $7.4 MM credit.

3) IPL Pipeline Tariff Edmonton to Sarnia declining at 2.5 % per year (real) $6.46/ton of pipeline delivered coal, i.e., 6.46 X 2.15 MMT = $13.9 MM.

4) Inventory (incremental to rail for coal = 0) and (incremental to IPL for oil = 5.6 days or 230,000 Bbls).

5) Working capital and start-up of 4 months operating, excluding additive.

6) CCA rule - 25% D.B. on plant, 6% D.B. on pipelines, 1/2 year rule, 43.84% tax.

TABLE 5 CONTINUED: transCOM ECONOMIC ASSUMPTIONS

7) Plants construction period 2 years, total cost of $204 MM - (50/50%) $102 MM in each of two years, pipeline construction period one year.

8) Project Life of 20 years.

9) No inflation, all economics are real in 1990 Canadian dollars.

NOTE on Rates of Return:

From analyzing utility rates of return, it appears that using a 10% (real) after tax rate of return on total investment is above the typical utility cost of capital assuming 15% (nominal) after tax on equity, 12% (nominal) before tax on debt, a 70/30 debt/equity ratio for a 20 year project life and 5% per year inflation. For calculating economics in a fashion similar to the government, Ontario Hydro, or other utilities, a discount rate of 6% real appears to be more appropriate.

TABLE 6: UNOCAL ADJUSTMENT OF CAPITAL COSTS FOR COAL SLURRY PLANT USED TO TRANSPORT COAL TO COLD LAKE AS CONDENSATE SLURRY

$MM - 1990 - CDN	Preparation	Separation 1 Plant	Separation 3 Plants
Total Field Costs	48.3	15.6	45.7* (save $1.1MM)
4 vs 5 Bowl Mills	41.4*		
Home Office 10%	4.1	1.5	2.6* (save $1.9MM)
Contingency 20%	9.1	3.4	4.8
Process Sub Total	54.6	20.5	53.1
Offsites	7.4	5.3	8.9* (save $7MM)
Total Plant	62.0	25.8	65.0
Volume adjustment			
- 78.5%	48.7*		
- 85%			55.3*

* Denotes where adjustments have been made for phased construction, integration into existing sites and lower volumes of coal.

TABLE 7: UNOCAL OPERATING COST ADJUSTMENTS FOR COAL SLURRY PLANTS USED TO TRANSPORT HIGHVALE COAL TO COLD LAKE AS A CONDENSATE SLURRY (MM $ Cdn/yr).

	Preparation		Separation	
	Bantrel	Revised	Bantrel	Revised
Power	1.977	1.117	1.174	.664
Gas	0	0	1.570	shrinkage
Labour	1.396	1.396	2.452	2.452
Supplies	.622	.622	.559	.559
Contingency @ 10%	.405	.314	.575	.368
Owners Cost @ 2% cap	N/A	.974	N/A	1.106
Total		4.423		5.149 + gas

TABLE 8A

TABLE 8A: TRANSPORTING HIGHVALE COAL TO COLD LAKE AS A CONDENSATE SLURRY. SCOPING STUDY BASED ON GROUPS OF 6 BOILERS, EACH BOILER SIZED AT 180 MM BTU/HR. ($MM Cdn 1990)

	Capital	Operating
Field Conversion	184.5	21.6
Slurry:		
Preparation	48.7	4.4
Pipeline to Edm	11.7	.5
Pipe Edm-Cold Lk	2.4	0.2
Separation	55.3	5.1 (ex gas)
Additive		1.8
Condensate Loss		0.6
Sub-Total for Slurry:	118.1	12.6
Total	302.6	34.2/yr

TABLE 8B: TRANSPORTING HIGHVALE COAL TO COLD LAKE AS A CONDENSATE SLURRY. SCOPING STUDY BASED ON GROUPS OF 6 BOILERS, EACH BOILER SIZED AT 180 MM BTU/HR.

	$/Tonne	$/MMBTU
Coal @ minemouth	11.00	.61
Boiler/transport @ above	44.83	2.48
Sub-Total		3.09
Steam cost less gas boiler cost		.76
less gas transport cost		.20
Plus gas cost (1.09 BCF @ $2.20/MMBTU)		.08
Equals Gas Field Price		2.21

Note: 1990 Alberta Market Price (AMP), average of all sales is $1.65/GJ or $1.75/MMBTU. Prices to export core market, 80% load factor, are closer to $1.90/MMBTU.

File: Sep13b91.wp

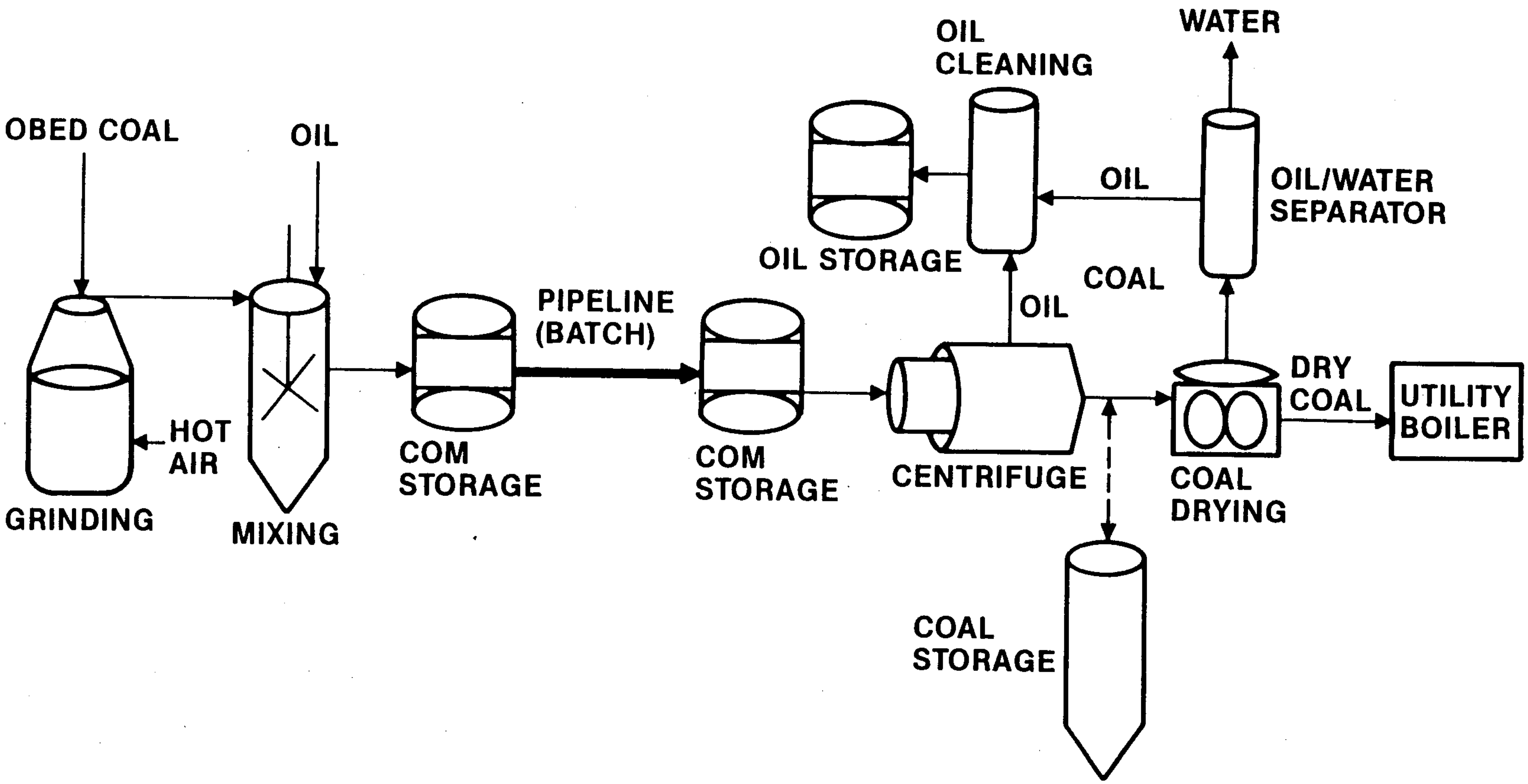
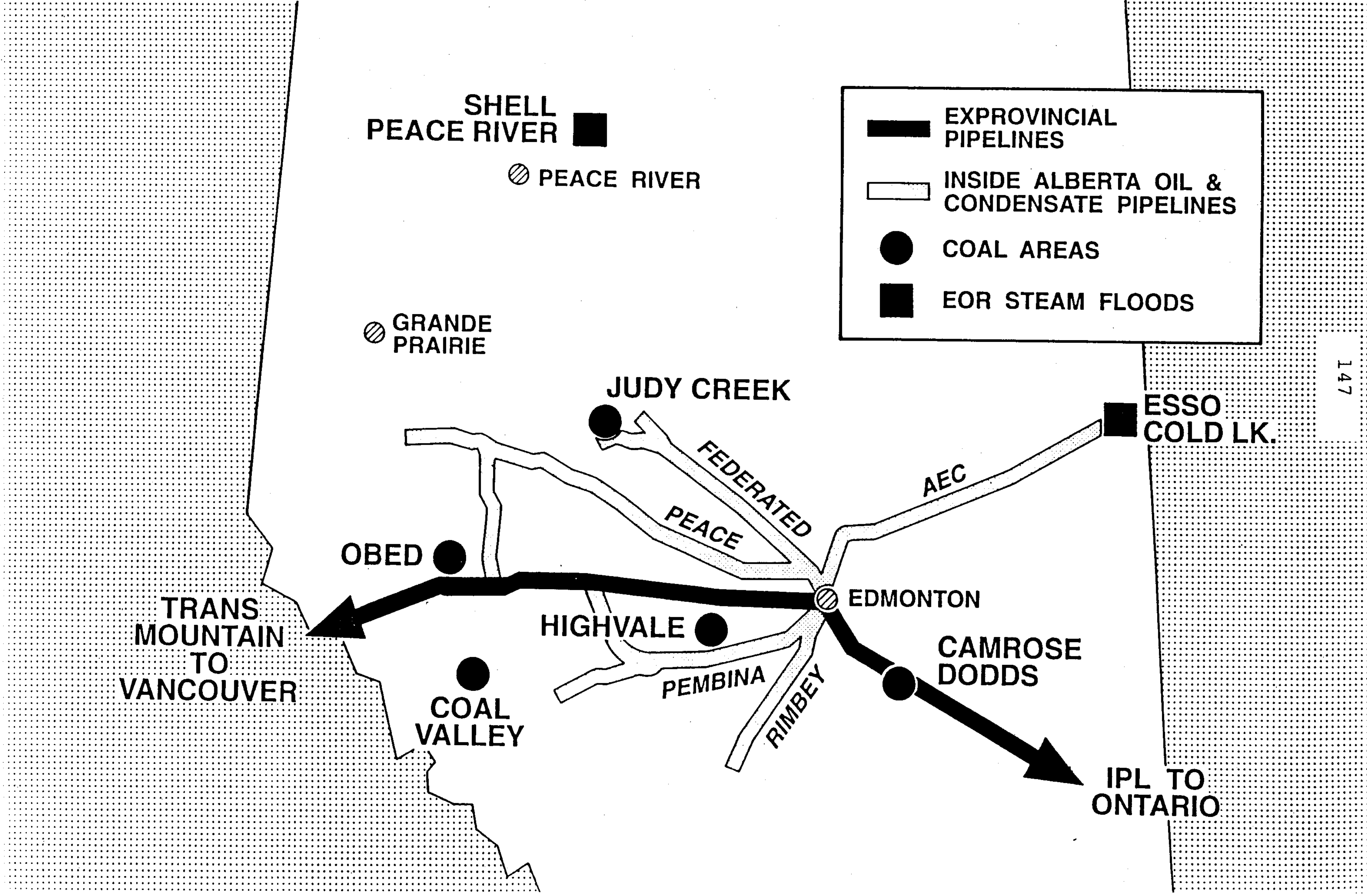


FIGURE 3. COAL-OIL MIXTURE TECHNOLOGY FOR PIPELINE SLURRY TRANSPORTATION.

FIGURE 2. ALBERTA PIPELINES FOR MOVING LIGHT OIL, NATURAL GAS CONDENSATES, AND COAL-CONDENSATE SLURRIES.



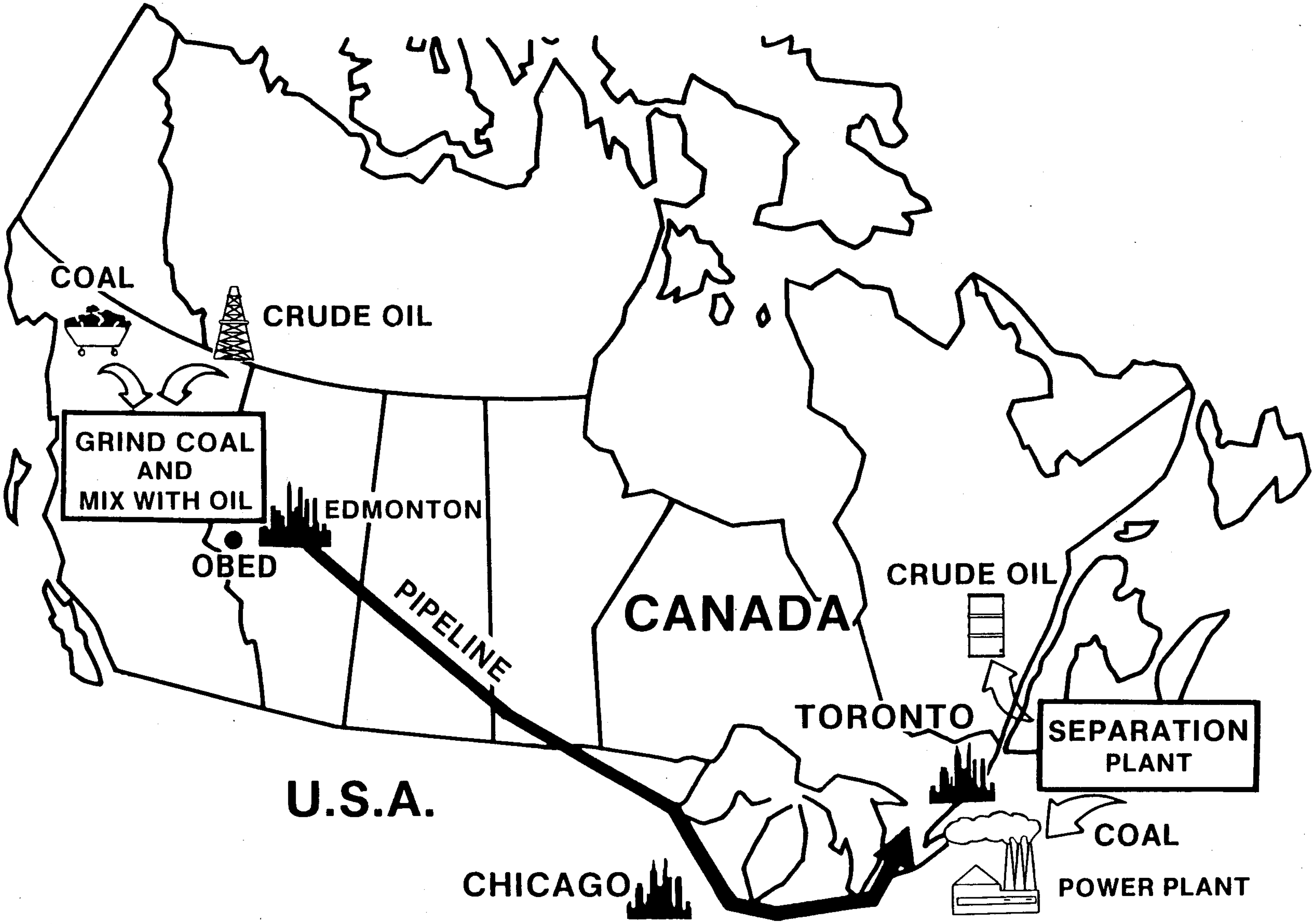


FIGURE 1. EXISTING OIL PIPELINE FOR TRANSPORTING COAL AS SLURRY TO EASTERN CANADA.

H_2S Capture in Coal Gasification Using Calcium Based Sorbents

G. Kovacik, A. Chambers, M. Oguztoreli, D. Ungarian, B. Croy
Alberta Research Council, Coal and Hydrocarbon Processing Department

1.0 INTRODUCTION

Coal gasification processes have a gas clean up (acid gas removal) stage to refine the gas before its end use. Conventional gas cleaning requires that the gases be quenched and the H_2S removed using a cooled solvent. The sulphur is subsequently recovered for sale. Gas cleaning adds considerable cost and complexity to the gasification plant. This results in decreased thermal efficiency when applied to integrated gasification combined cycle (IGCC) for electric power generation.

Hot gas cleanup is estimated to increase overall IGCC power plant thermal efficiency by up to 3% (Gallaspy and Sears, 1990). Most hot gas cleanup research has been aimed at developing exotic, regenerable sorbents working in gases produced from high sulphur content coals (Pitrolo and Bechtel, 1988). The use of inexpensive sorbents for capturing H_2S from hot product gas streams from low sulphur content (<1.0 wt%) coals could improve the efficiency and economics of IGCC electricity generation.

Experiments were performed using both simulated gasification conditions, in a laboratory fixed bed unit and sorbent injection into an entrained flow gasifier. The results of these tests confirm the ability of calcium based sorbents to remove H_2S from gasification product gases. Over 90 wt% sulfur capture was observed in both laboratory fixed bed experiments and in pilot scale sorbent injection tests.

2.0 EXPERIMENTAL RESULTS and DISCUSSION

2.1 Equilibrium Calculations

To gain a better understanding of sorbent injection applications, equilibrium calculations of sulfur capture using calcium based sorbents were performed. The chemical equilibrium was calculated on the basis of the minimization of Gibb's free energy using F*A*C*T[1] software. Product gas compositions predicted for the gasification of low sulfur subbituminous coal in Shell and Texaco gasification processes were used for the calculations (Kovacik, 1990; Kovacik, 1989). The equilibrium sulfur capture was calculated for a Ca/S ratio of 3.0, temperatures of 800 to 1500 K, and pressures up to 2 MPa. The predicted sulfur capture, for low sulfur coal using calcium based sorbents, was up to 95%, Figure 1. High steam concentration in the product gas was predicted to suppress sulfur capture due to the calcium oxide/hydrogen sulfide equilibrium:

$$CaO + H_2S \longleftrightarrow CaS + H_2O$$

[1] F*A*C*T is a copyright product of Thermfact Ltd/Ltee, Mount Royal, P.Q.

Thus, the potential for sulfur capture using calcium based sorbents in gasifiers that produce high concentrations of water vapour may be limited. High water vapour concentrations are typical of coal/water slurry fed gasifiers, such as Texaco and Dow processes. The equilibrium studies also indicated that the optimal temperature range for sulfur capture was 1000 to 1300 K, Figure 1. Therefore, sorbent injection was expected to be most effective at temperatures below 1300 K.

2.2 Simulated Gasification Sorbent Injection Tests

A fixed bed sorbent testing unit was used to study sulfur gas removal from simulated coal gasification product gas. The unit consisted of a gas flow control assembly, a reactor, an electric furnace, condenser train, and a gas collection and analysis assembly (see Figure 2). Mass flow controllers were used to control the gases entering the reactor. The unit operated at pressures up to 1.5 MPa and reactor temperatures up to 1300 K. The product gases were collected and analyzed on-line with a gas chromatograph equipped with flame photometric (sulfur specific) and thermoconductivity detectors. A systematic investigation of sorbent performance over a range of pollutant gas concentrations and operating temperatures was performed (see Table 1). Gas temperature, input gas composition and product gas composition were controlled and measured. The tested sorbents were selected based on their low cost (all are waste materials) and availability.

Table 1 Sorbent Test Conditions

Sorbent Tested	Input Gas Composition	Temperature Range
Water Treatment Sludge (>95 wt% $CaCO_3$)	60 mol% CO, 40 mol% H_2, 1760 and 600 ppmv H_2S	800-1300 K
Metallurgical Red Mud (>90 wt% Fe_2O_3)	60 mol% CO, 40 mol% H_2, 1760 and 600 ppmv H_2S	700-1200 K
Power Plant Fly Ash (>8 wt% Ca, >4 wt% Fe)	60 mol% CO, 40 mol% H_2, 1760 and 600 ppmv H_2S	800-1000 K

Experimental results confirmed the ability of water treatment sludge, metallurgical red mud, and coal power plant fly ash to remove H_2S from dry synthetic gasification product gases (Table 2). Water treatment sludge was selected for furthur study in CANMET's entrained bed gasifier based on these results.

Table 2 Preliminary Sorbent Sulfur Capture Results
Input gas - 60 mol% CO, 40 mol% H_2
600 ppmv H_2S

Sorbent	Sulfur Capture	Effective Temperature Range
Water Treatment Sludge (>95 wt% $CaCO_3$)	over 90 wt%	800-1000 K
Metallurgical Red Mud (>90 wt% Fe_2O_3)	over 90 wt%	700-800 K
Power Plant Fly Ash (>8 wt% Ca, >4 wt% Fe)	up to 75 wt%	700 K

2.3 Sorbent Injection Tests at CANMET's Bells Corners Gasification Laboratory

Sorbent injection experiments were performed using CANMET's gasifier. In these experiments dried water treatment sludge (>95 wt% $CaCO_3$, average particle size 28 microns) sorbent was introduced into the gasifier separate from the coal feed where conditions (temperature) were more favourable for sulfur capture. The design of CANMET's system was based on a downscaled design of the Brigham Young University high pressure entrained coal gasifier. It is a down-fired, axisymmetric reactor consisting of 2 or 3 ceramic-lined sections. These sections are 60 cm in height with an inside diameter of 13 cm. The overall length can be varied from 120 to 180 cm depending whether 2 or 3 sections are installed. Castable ceramic refractory with a total thickness of 9 cm was used to line the inside of the reactor. A schematic of the gasification system and the gasifier are presented in Figures 3 and 4, respectively.

The coals used in these experiments were a low sulfur subbituminous and a higher sulfur bituminous. Coal analyses are given in Table 3. Gasification experiments were performed at oxygen to coal ratios of 0.75 kg/kg for the subbituminous, and 0.90 kg/kg for the bituminous, on an as-received coal basis. These oxygen to coal ratios were selected based on earlier experiments to give high gasification cold gas efficiency. Calcium to sulfur ratios (Ca/S), based on added calcium, were 3.5 and 6.0 mol/mol for the low sulfur subbituminous coal and 1.0 mol/mol for the bituminous coal. The subbituminous coal had an inherent calcium to sulfur ratio of about 6.0 mol/mol based on sulfur and ash elemental analyses. Two sorbent injection locations were used for the subbituminous coal. One was through the burner secondary gas supply tube, surrounding the coal/oxygen flame, at a Ca/S of 3.5. The other was through a sight window about 90 cm after the burner and about 30 cm before the reactor exit, at a Ca/S of 6.0. The sorbent was injected through the burner secondary only for the bituminous coal tests.

Table 3 Coal Analyses
(wt% as received)

	H_2O	Ash	C	H	N	S	O (by diff)
Subbituminous Coal	6.0	15.5	59.6	3.4	0.7	0.2	14.6
Bituminous Coal	2.5	7.4	74.6	4.8	1.4	2.9	6.4

The results of the sorbent injection tests for the low sulfur subbituminous coal are summarized in Table 4. These data show over 95 wt% sulfur retention when the sorbent was injected through the burner secondary. Lower sulfur capture, about 60 wt%, was observed for sorbent injection through the reactor sight window, even though the calcium to sulfur ratio was much higher for this case. This reduced capture was likely due to less effective mixing of the sorbent with the product gases and the short contact time, the sight window was only about 30 cm from the reactor outlet. The relatively high sulfur retention, 50 wt%, observed without sorbent injection was likely due to the inherent calcium in the coal ash. The inherent calcium to sulfur ratio is approximately six to one for this coal. It should be noted that for these gasification tests carbon conversions were 80 to 88 wt%. Therefore, about 12 to 20 wt% of the sulfur would be expected to remain in char/ash in the absence of any sorbent.

Table 4 CANMET Sorbent Injection Results for low sulfur subbituminous coal

	Sorbent Injection Point: Burner Secondary	Sight Window	None
Carbon Conversion, wt%	83	88	83
Ca/S, mol/mol	3.5	6.0	0.0
Sulfur Retained in Char/ash, wt%	95	60	50
Estimated SO_2 Emission,* ng/J	10	80	100

* SO_2 Emission if the product gas was combusted in an IGCC plant, ng of SO_2 per Joule of input fuel HHV.

A preliminary test of the effectiveness of sorbent injection for higher sulfur coals was also performed, as shown in Table 5. With a calcium to sulfur ratio of one, 40 wt% of the sulfur was retained in the char/ash. Without sorbent injection the sulfur retained in the char/ash was 20 wt%, which was in line with the 80 wt% carbon conversion. Thus, about 25% of the sulfur released to the gas was captured by the sorbent.

Table 5 <u>CANMET Sorbent Injection Results for high sulfur bituminous coal</u>

	Sorbent Injection Point: Burner Secondary	None
Carbon Conversion, wt%	82	80
Ca/S, mol/mol	1.0	0.0
Sulfur Retained in Char/ash, wt%	40	20
Estimated SO_2 Emission,* ng/J	500	1000

* SO_2 Emission if the product gas was combusted in an IGCC plant, ng of SO_2 per Joule of input fuel HHV.

Using data from a Coal Association of Canada IGCC study (R.F. Geosits, 1991) the effect of utilizing sorbent injection, as an alternative to conventional sulfur recovery, on IGCC plant performance and economics was estimated. Table 6 details the impact of sorbent injection on plant performance. The thermal efficiency gain of about 0.3% is achieved due to reduced internal power consumption by elimination of the acid gas removal and sulfur recovery sections. This gain in efficiency would reduce carbon dioxide emissions slightly, even after the carbon dioxide release from the calcium carbonate was accounted for. If the calcium to sulfur ratio can be reduced to below three to one greater performance gains could be achieved. This analysis also assumed that the gasifier product gas was quenched in both systems. If sorbent injection was used as a hot gas cleaning technique, further plant thermal efficiency increases would result.

Table 6 <u>Estimated IGCC Performance</u>

Sorbent Injection Versus Conventional Gas Cleaning for Low Sulfur Coal

	Sorbent Injection	Conventional Acid Gas Treating
IGCC Plant Efficiency, % of fuel HHV	40.7*	40.4
SO_2 Emissions, ng/J	10	2
Relative CO_2 Emissions (CO_2/Power Output)	0.998**	1.0

* Sorbent injection was estimated to increase overall thermal efficiency by 0.3% by elimination of acid gas treating and sulfur recovery plants.

** Based on molar Ca/S of 3.0 and 40.7% thermal efficiency.

A preliminary estimate of the impact of sorbent injection on IGCC plant cost is shown in Table 7. The cost of sorbent injection includes limestone costs and increased solids disposal costs. It was assumed that these solids would be landfilled with the coal ash. Sorbent injection was estimated to cost 13% of conventional acid gas removal and sulfur recovery for low sulfur subbituminous coal.

Table 7 Comparison of Relative Cost of Sulfur Capture

Sorbent Injection Versus Conventional
Low Sulfur Bituminous Coal

	Sorbent Injection	Conventional*
Capital Cost	0.02	0.57
OEM Cost	0.11	0.43
Total Relative Cost	0.13	1.00

* Includes sulfur credit.

3.0 CONCLUSIONS

The following conclusions can be drawn:

1. Sulfur capture of up to 95% was demonstrated using calcium based sorbents in both simulated and actual gasification of low sulfur subbituminous coal.

2. Calculated emissions of SO_2 and CO_2 with sorbent injection were similar to those observed for conventional IGCC power generation using low sulfur subbitumious coal as the fuel.

3. Overall thermal efficiency of the IGCC plant could increase by more than 0.3% (from 40.4% to 40.7%, for example) by using sorbent injection instead of conventional sulfur recovery for cold gas cleaning.

4. Estimated cost of sulfur removal was about 13% that of conventional sulfur recovery in the low sulfur subbituminous coal case.

4.0 RECOMMENDATIONS

The following recommendations can be made as a result of this work:

1. Controlled parametric experiments should be performed to determine optimum conditions for sorbent injection in terms of temperature, residence time and calcium to sulfur ratio.

2. Potential problems with solids disposal from sorbent injection application should be investigated.

3. Applicability of sorbent injection to higher sulfur coals should be determined.

ACKNOWLEDGMENT

This work was administered by the Alberta Office of Coal Research and Technology and funded in part by the Alberta/Canada Energy Resources Research Fund and CANMET. The authors would also like to acknowledge the contributions of technical staff of the Coal and Hydrocarbon Processing Department of the Alberta Research Council and the Gasification Laboratory of CANMET.

REFERENCES

D.T. Gallaspy, R.E. Sears, "Site Specific Study of Air-Blown Gasification Power Plants with Hot Gas Cleanup", Proc. Ninth Annual EPRI Conference on Gasification Power Plants, (1990).

R.F. Geosits, et. al., "The Coal Association of Canada IGCC Feasibility Study Update", Canadian Electrical Association Fall Meeting, (1991).

G. Kovacik, M.Oguztoreli, A. Chambers, B. Ozum, "Equilibrium Calculations in Coal Gasification", The International Journal of Hydrogen Energy, Feb. 1990.

G. Kovacik, A. Chambers, "Evaluation of Canadian Coals for Gasification with Texaco, British Gas, and Shell Technologies", (1989)

A.A. Pitrolo, T.F. Bechtel, "Simplified IGCC: Coal's 'Adam Smith' Response to a Changing World", Proc. Seventh Annual EPRI Conference on Gasification Power Plants, (1988).

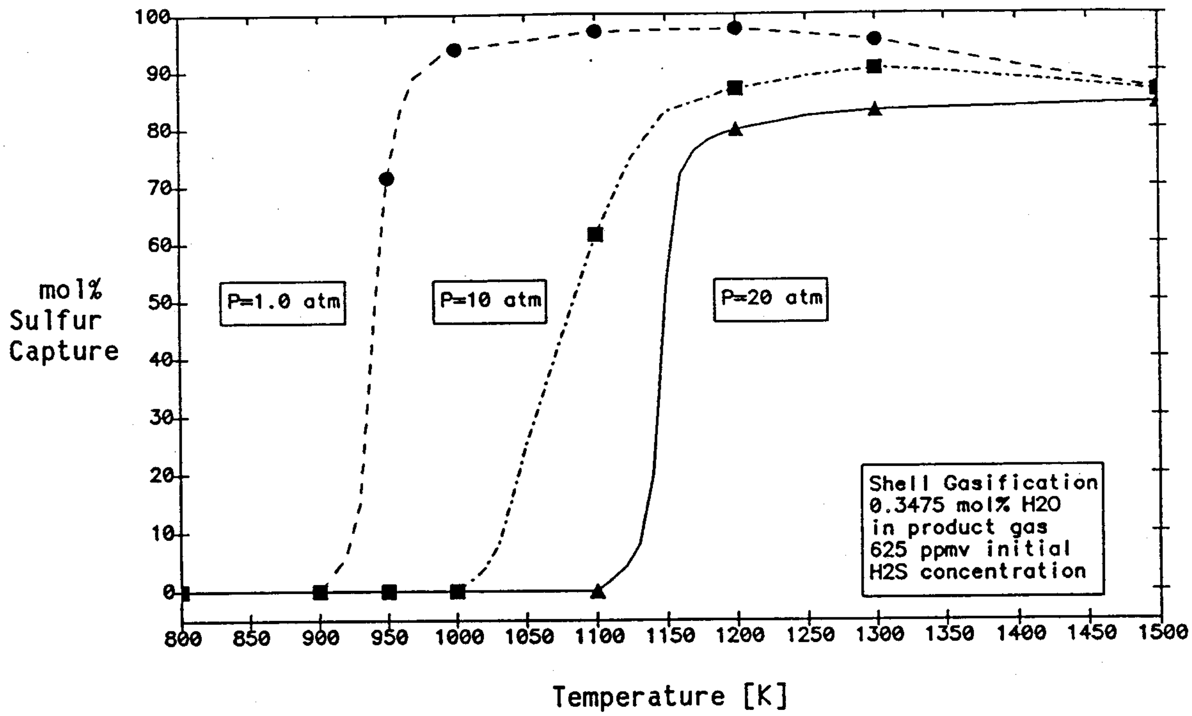


Figure 1 - Predicted Equilibrium Sulfur Capture using F*A*C*T Software
Calcium/Sulfur = 3.0 mol/mol

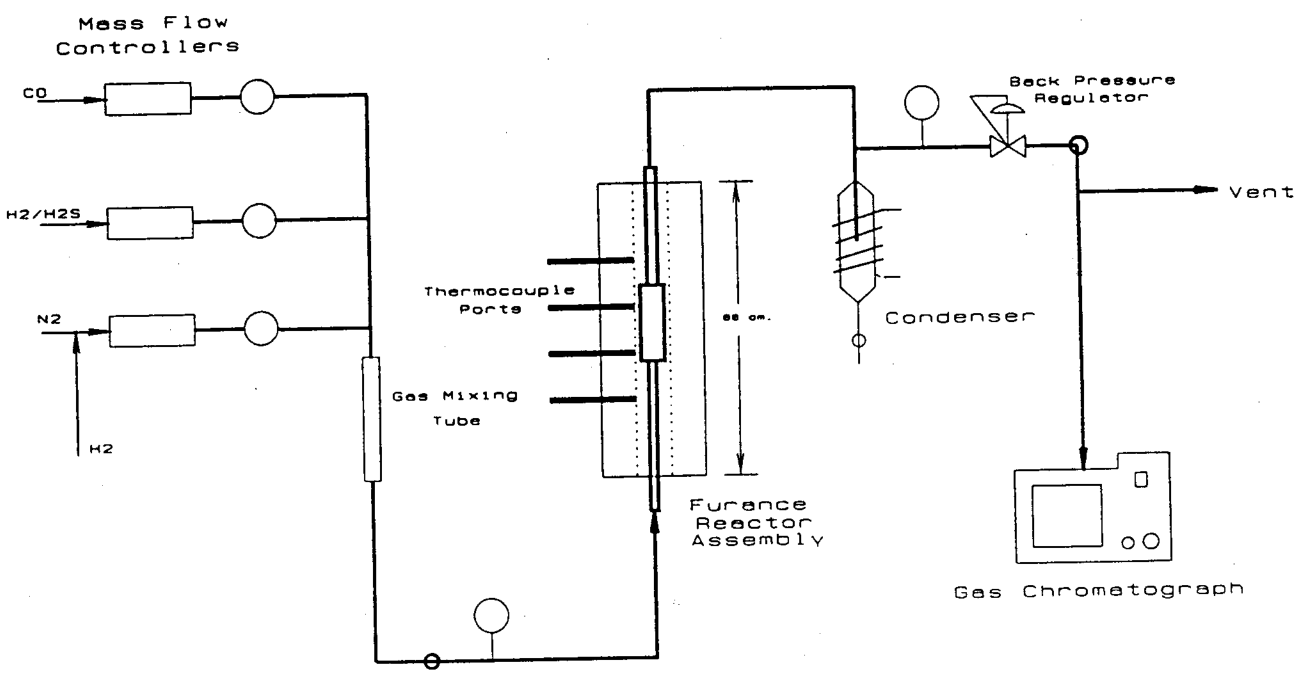


Figure 2 - Fixed Bed Sorbent Testing Unit

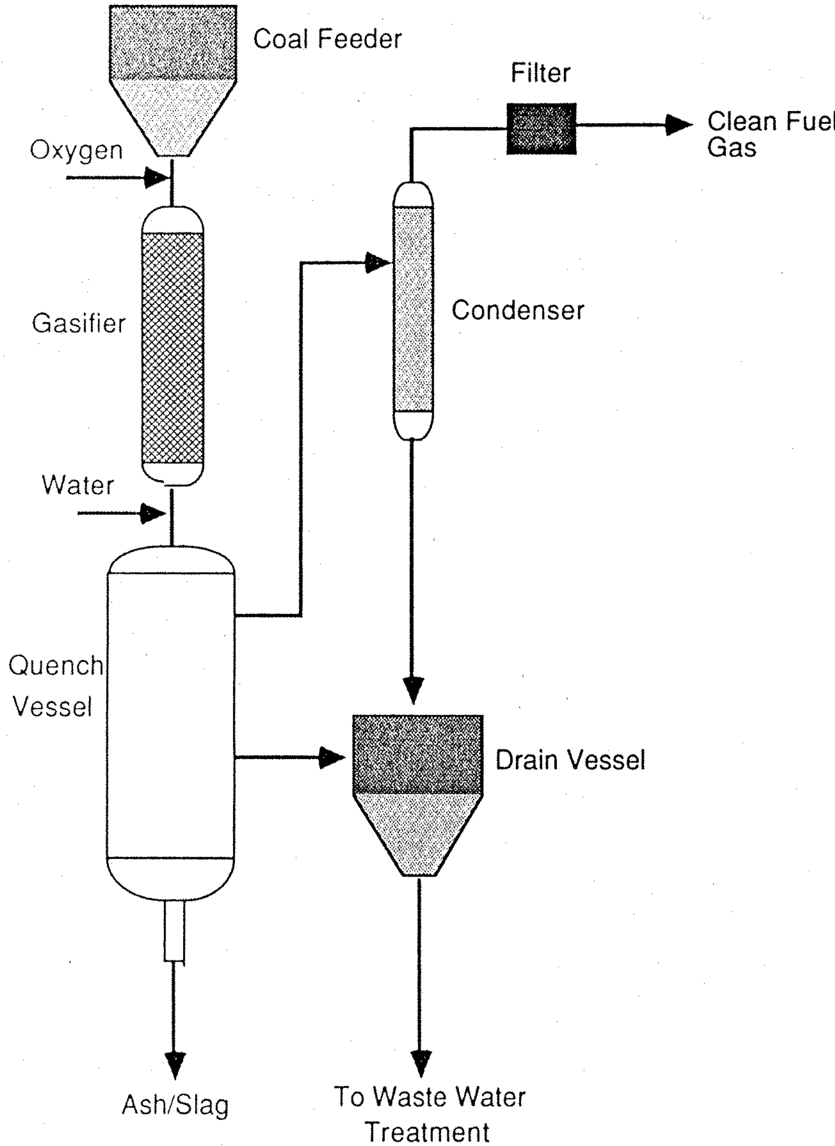


Figure 3 - CANMET Entrained Flow Gasifier System

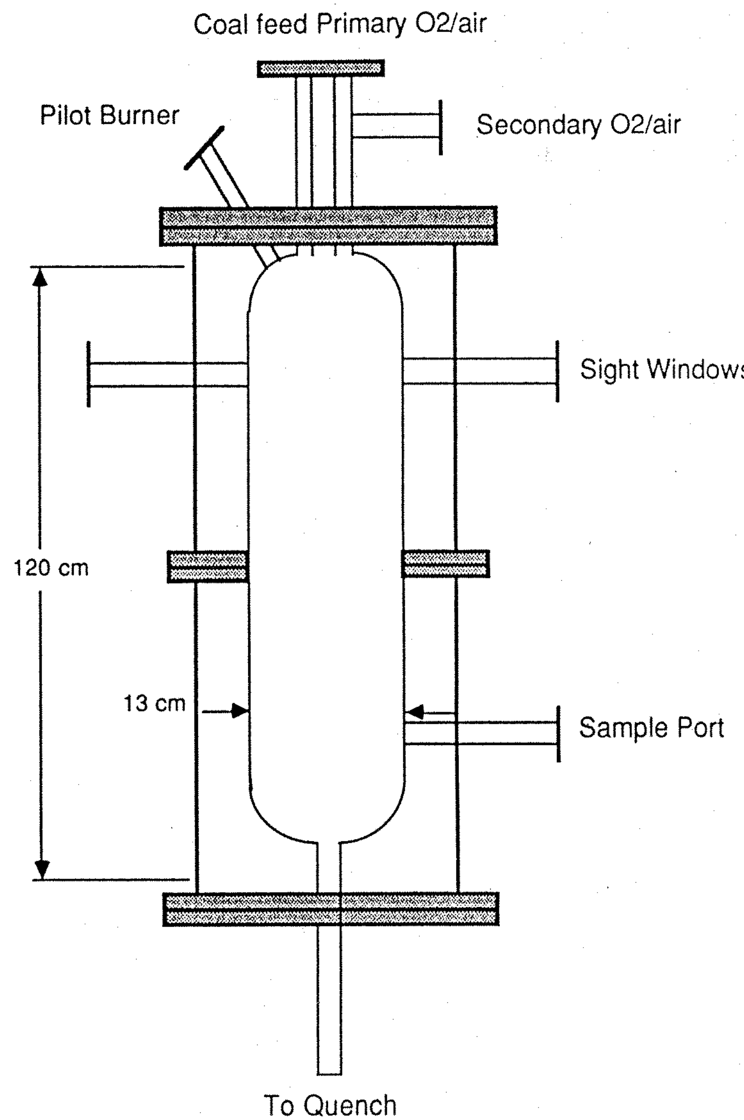


Figure 4 - CANMET Entrained Flow Gasifier

MORPHOLOGICAL AND MINERALOGICAL CHARACTERIZATION OF CHARS OF HIGH VOLATILE BITUMINOUS COAL VALLEY COAL FROM ALBERTA

Prepared By:

R.C. JOSHI

and

P.V. SIVAPULLAIAH

1.0 INTRODUCTION

Noncombustible residue generated by power plants using pulverized-fuel furnaces are collected as fly ash and bottom ash. Eighty percent of the ash is collected as fly ash, about 10% of the ash is used as an admixture in cement concrete. In the past few decades air pollution problems have become very severe, therefore, attempts are being made to burn coal effectively as well as without producing too much NO_x. Such attempts also effect the coal combustion residue. The resultant ash, therefore, is likely to have different properties than the one produced by modern power plants. This study is aimed at characterization of the coal combustion residue or ash particles. Such characterization should hopefully allow evaluation of the modified no NO_x ashes for use in cement concrete.

At the International Flame Research Foundation, Knill et al (1989) carried out devolatalization and char combustion tests on high volatile bituminous (HVB) coals using a isothermal plug flow reactor. The objective of their study was to increase the coal properties data base to support burner design and modelling research, and to develop a simplified coal characterization test. Eleven of the chars obtained after pyrolysis and char combustion of coal valley coal at temperatures between 950-1400°C and residence times between 100-500 ms were used in the present study.

Four chars, obtained by firing the same coal valley coal in Aerodynamically Air Staged Burner in different low No_x flames, were also used for the present study. These chars and residues (ashes) were produced by Smart et al (1989) for semi industrial scale combustion experiment. Less than 10 g of each of these fly ashes or chars were available for study. The laboratory studies were, therefore, limited to morphology and mineralogy of particles and their reactivity with lime.

1.1. Objective

The objectives of the present study were to obtain data on:

(a) Morphological and mineralogical characterization of the chars and noncombustible mineralic residue of HVB coal valley coal after pyrolysis and char combustion at different temperatures and residence times.

(b) Characterization of the chars and noncombustible residue obtained by firing the HVB coal in different low NO_x flames in the Aerodynamically Air Staged Burner.

(c) Reactivity of the noncombustible residue used for the study in (a).

2.0 BACKGROUND

The design and performance prediction of burners, used in furnaces burning pulverized coal, requires a knowledge of the properties of the fuel. The rate of pyrolysis, total volatile yield and rate of char combustion are important parameters for such studies. To increase the coal property data base and to develop a simplified coal characterization test for high volatile bituminous coals, Knill et al (1989) at the International Flame Research Foundation carried out the devolatalization and char combustion tests on seven bituminous coals. The coals were pyrolysed at 1400°C for 150 ms. Combustion of the chars was carried out in 5% oxygen at temperatures of 950°C, 1100°C, 1250°C, and 1400°C. The residence times varied from 100 ms to 500 ms. Isothermal Plug Flow Reactor was used for char combustion. Based on the results of these tests, a simplified coal characterization technique was proposed. This adequately describes the pyrolysis and char combustion properties up to 80% burn out. Changes in surface properties during combustion were thought to significantly affect the burning rate of the particles in the final ten percent weight loss. Hence, further studies are recommended on these residues to determine the cause of reduced reaction rate at high conversion. One of the seven coal combustion residues was selected for the present study.

Semi-industrial scale combustion experiments have been a key area of research at IFRF. Because of strict pollution emission legislation, many combustion equipment manufacturers have been expending considerable effort designing low NO_x combustion systems. A detailed understanding of the NO_x formation process in conjunction with dominant fluid dynamic and thermochemical aspects of the combustion system at a reduced scale is required. For this study, Smart el al (1989), fired different coals in the Aerodynamically Air Staged Burners at IFRF using low NO_x flames. Four of these chars residues from coal valley coal were also selected for the present study.

2.1 Materials and Methods of Analysis

Only very minute samples, ie. less than 50g, were available at IFRF for detailed study. These samples were barely sufficient to conduct x-ray, scanning electron microscopic, and thermo gravimetric analysis. Yet an attempt was made to treat some of the samples with lime to determine reactivity of the residue or ash particles.

Chars or Residues Used: The chars used for this study along with their sources are given in Table 1.

X-ray Diffraction Patterns: Powder x-ray diffraction patterns of all the samples were recorded on Phillips pw 1139 automatic diffractometer using cobalt radiation.

Scanning Electron Microscopic Studies: Char and ash surface characteristics were examined using a Cambridge 150 Scanning Electron Microscope. The samples were sputter coated with a thin layer of gold. A series of microphotographs from several orientations were taken on each of the samples. The magnifications were varied as considered necessary.

Thermo Gravimetric Analysis: Thermo Gravimeters Analysis of the samples were conducted using Dupont 951 Thermal Analyser. A uniform heating rate of 5°C/min. was used for the entire range of heating from ambient room temperature to 800°C. The furnace was purged with nitrogen gas during testing.

Pozzolanic Reactivity: The chars and ashes produced by combustion or heat treatment of coal at different temperatures with varying residence times were mixed with 10% by weight of lime and water. The samples were cured for one week at room temperature. The new cementitious compounds formed if any along with morphological changes were studied by x-ray diffraction and scanning electron microscope.

3.0 RESULTS AND DISCUSSION

3.1 X-ray Studies

Mineralogy of the chars was determined using x-ray diffraction data. The mineralogy of the chars does not seem to be effected by the temperature or the duration of combustion as is evident from x-ray traces in Figure 1. The type of low NO_x flames also does not seem to impart significant and detectable changes in the mineralogy of the ashes. The major minerals present in the ashes obtained by firing the high volatile bituminous coal are mullite, quartz, magnetite, hematite and calcite.

The glass phase of ashes cannot give a distinct set of Bragg reflections; but, some structural information can be obtained in the broad diffused maximum present in the diffractograms. The position of this diffused maximum occurs at larger Bragg angles in high calcium ashes compared to low calcium bituminous ashes (Diamond, 1984). In all the ashes of the present study, this broad maximum is seen at Bragg angles corresponding to 3.86 Å d-spacing confirming that all these ashes are low in calcium content. The general shape of this broad peak is similar for all the ashes studied. This indicates that the composition of the glassy phase does not vary with combustion temperature, residence time, or type of flame used. However, the area of the peak varies indicating the variations in the amount of glassy phase present in the ashes. The data in the X-ray diffractograms indicates that the amount of glassy phase decreases with increasing firing temperature. At any temperature increase of residence time further decreases the glassy phase; but, there seem to be exceptions to this observation. At 1250°C and 1400°C increase of residence time from 100 ms to 300 ms, decreases the glassy phases. The amount of the glassy phase is important, because it has been correlated with pozzolanic activity of the ashes.

In commercial furnaces of the type used by utility companies in Alberta, the residence time for coal particles varies from 100 ms to 25. The data is, therefore, also applicable to commercial fly ashes.

3.2 Scanning Electron Microscope Studies

Scanning electron microscopic examination of ashes is an extremely useful technique for characterization of morphology, as well as, size and shape of particles present in the ashes. The ash particles formed are generally spherical in shape, as can be seen in Figure 2. The spherocity results from surface tension effects on the localized gangue material (Raask, 1982). As seen from SEM photographs, the number of spheres and their sizes and surface texture vary in different ashes. A large number of unburnt carbon particles are seen in the ashes produced by coal combustion at 950°C, at both the residence times of 150 ms and 500 ms. Though reduced in number, ashes obtained at 1100°C still show a large number of carbon particles at all residence times. Spherical ash particles are beginning to form at 150 ms and increase substantially at 500 ms. Ashes produced at 1250°C show less number of carbon particles. This reduction of carbon particles is further improved with increase in residence time. Corresponding to a decrease in carbon particles found in these ashes, there is an increase in the formation of the spherical ash particles. However, further increase in temperature of firing to 1400°C does not improve the formation of these spherical particles. The general spherical shape of the ashes indicate that water demands of concretes made with these ashes may not be high. Plero spheres, hollow spheres, packed with smaller spheres, were observed in the ash samples obtained at 1400°C with a residence time of 300 ms.

The morphology of the ashes obtained in different low NO_x flames does not show noticeable changes. The number of spherical ash particles definitely are less in the ashes obtained in flames 1 and 2 compared with the ashes obtained in flames 3 and 4, see Figure 2. It should, therefore, be deduced that low NO_x flames in commerical furnaces also will tend to produce ash particles of low spherocity and possibly high crystallinity.

4.0 THERMO GRAVIMETRIC ANALYSIS

The TGA curves of the ashes in Appendix C show that most of the weight loss occurs between 400°C and 700°C for all the ashes. For most ashes the weight loss occurs in two stages. One between 400°C and 450°C corresponding to the removal of adsorbed water, and the other between 600°C and 700°C corresponding to the unburnt carbon content.

TGA curves of the ashes produced at different firing temperatures (Figure 3) show that the total weight loss decreases with increase in firing temperature. However, there is little change as the temperature is increased from 1250°C to 1400°C (Table 2). Apparently, all the carbon and other combustibles are burned at about 1250°C.

At any firing temperature increase in residence time also decreases weight loss of the ashes. At 1250°C and 1400°C, increase of residence time from 200 ms to 300 ms does not reduce the weight loss further. Thus, firing at 1250°C with residence time of 200 ms brings about optimum combustion of coal particles.

TGA curves of the ashes obtained in different low NO_x flames show that the weight loss is more for ashes produced in flames 1 and 2 than the ashes obtained in flames 3 and 4.

5.0 POZZOLANIC REACTIVITY

SEM pictures clearly show the formation of new cementitious compounds in all the ashes obtained at various temperatures and residence times. The compounds are either needle like ettringite particles or hexagonal CSH or other calcium or aluminum hydrate particles. Formation of ettringite shows that significant early compressive strengths may be achieved with concretes made with these ashes. Formation of CSH particles suggests that long term compressive strength may also be obtained with concretes containing these ashes. There are no qualitative variations in the type of compounds formed with ashes obtained at different temperatures or residence times. Further studies are required to assess quantitative variations in these ashes. Such studies should be conducted with the aim of producing enough fly ash to evaluate its pozzolanic activity.

Results of x-ray diffraction of these ashes show that the ratios of intensity of lime peak at d = 2.77 to that of the intensity of the mullite peak at d = 2.54 are not different for ashes without and with lime. This indicates that most of the lime added has reacted with ashes to form cementitious compounds. Also, the intensity of the broad peak indicating the glassy phase has reduced in lime treated ashes. This confirms that the glassy phase is involved in reactions with lime. But, ettringite peaks are not observed in the x-ray diffraction patterns of lime treated ashes at the end of one week.

The observations suggest that low NO_x ashes may also be pozzolanic; but, further research is required to confirm this observation.

6.0 CONCLUSIONS

1. There are no significant differences in the mineralogy of the ashes produced at different combustion temperatures and with different residence times. The amount of glassy phase varies in different

ashes. The glassy phase generally decreases with increase in firing temperature or residence times.

2. SEM studies show that the carbon particles decrease with increase in firing temperature or residence time. Correspondingly, there is an increase in the formation of spherical ash particles. However, there is improvement in the production of spherical particles as the temperature is increased from 1250°C to 1400°C. At 1250°C, increase of residence time from 200 ms to 300 ms also improves the formation of the spherical particles.

3. TGA studies support the observations made from x-ray diffraction and SEM studies.

4. Pozzolanic reaction products are formed in all the ashes obtained by coal combustion at different temperature and varying residence times. Further studies are required to assess the quantitative differences in the activity of commercial ashes.

7.0 FURTHER STUDIES

The present studies have been confined to one type of HVB coal ash. All the tests were conducted on minute samples obtained from IFRF. Since combustion studies at IFRF have indicated that differences exist in chars of different coals, the ashes of other coals should be subjected to similar studies.

Evaluation of quantitative differences in the pozzolanic reactivity of the ashes requires further studies. To meet this end research needs to be conducted on modification of ash by burning coal at different temperatures, and also adding calcium carbonate to the pulverized coal.

REFERENCES

1. S. Diamond, "Cem. Concr. Res.", 14, 455-462, (1984).

2. K.J. Knill, T.C.J. Maalnan, and M.E. Morgan, "Development of a Combustion Characterization Technique for High Volatile Bituminous Coals", IFRF Doc. No. F88/a/10, 1989.

3. E. Raask, "In International Symposium on the Use of PCA in Concrete", edited by J.G. Cabrera and A. Cusens, pp. 5-16, (1982).

4. J.P. Smart, W.L. Van de Kamp, and M.E. Morgan, "The Effect of Scale on the Performance of the Aerodynamically Air Staged Burner", IFRF Doc. No. F37/a/19, 1989.

Table 1. Chars Used

Sl No	Pyrolysis Temperature °C	Res. Time (ms)	Furnace	Reference
1	950	150	IFRF	Knill et al (1989)
2	950	500	IFRF	Knill et al (1989)
3	1100	150	IFRF	Knill et al (1989)
4	1100	300	IFRF	Knill et al (1989)
5	1100	500	IFRF	Knill et al (1989)
6	1250	100	IFRF	Knill et al (1989)
7	1250	200	IFRF	Knill et al (1989)
8	1250	300	IFRF	Knill et al (1989)
9	1400	100	IFRF	Knill et al (1989)
10	1400	200	IFRF	Knill et al (1989)
11	1400	300	IFRF	Knill et al (1989)
	Burner	Flame		Reference
12	AASB	1		Smart et al (1989)
13	AASB	2		Smart et al (1989)
14	AASB	3		Smart et al (1989)
15	AASB	4		Smart et al (1989)

Table 2. Summary of TGA Data on Ashes Produced by Heating to Different Temperature and Residence Times

Temp.	Res. Time ms	Total loss	Loss Between 600-700°C
950	150	46.0	3
	500	24.0	3
1100	150	16	3
	500	7	5
1250	100	15	3
	200	4	4
	300	3.5	3.5
1400	100	11.5	3.5
	200	4	3
	300	3.0	2.0
Flame	1	3.5	3.0
	2	5.0	4.5
	3	2.5	2.0
	4	2.0	1.0

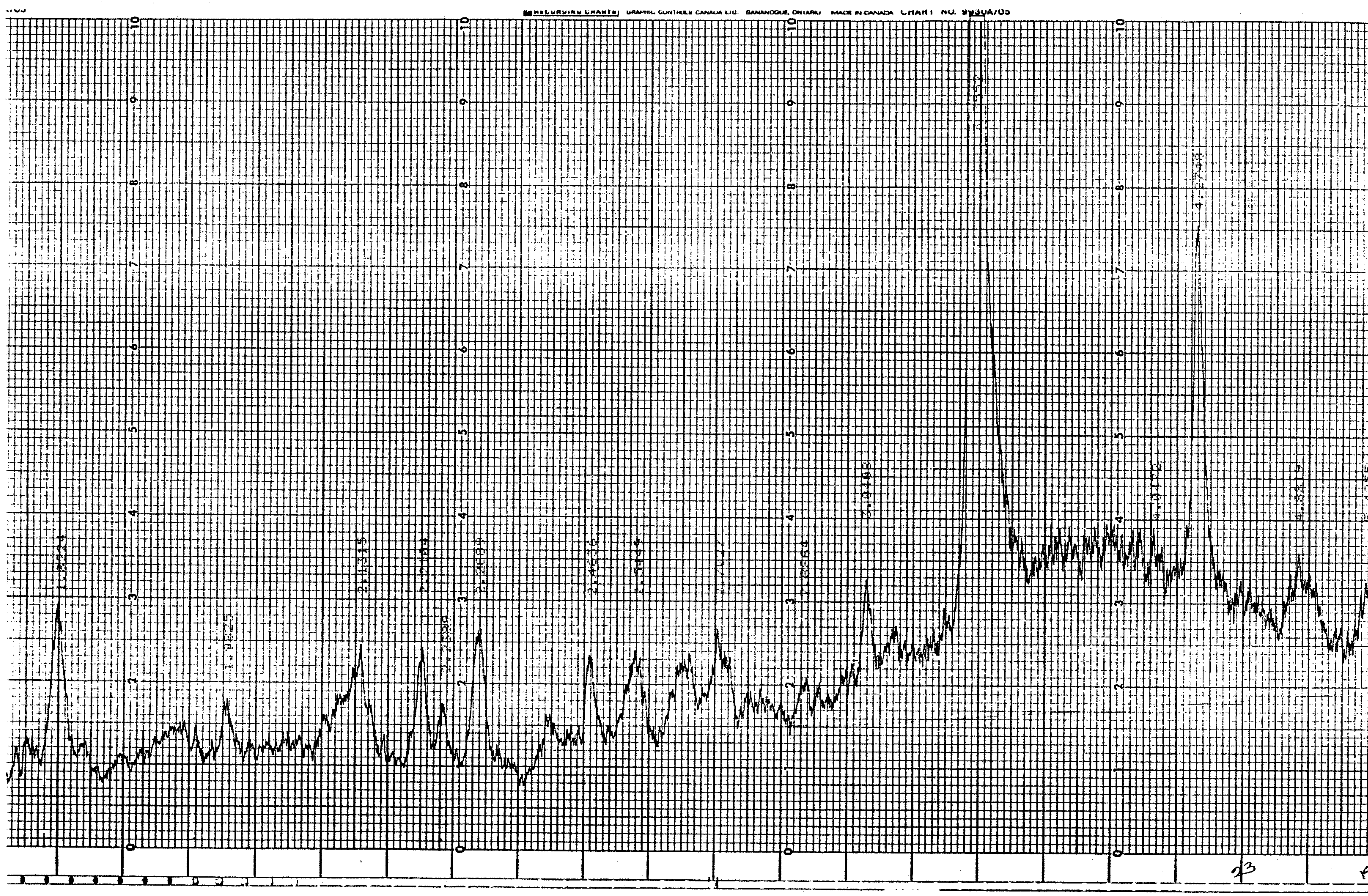
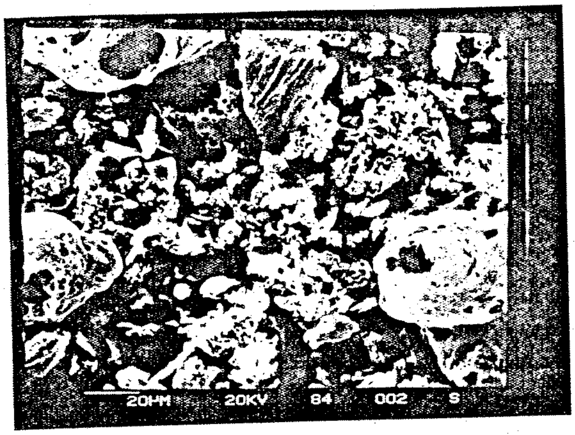
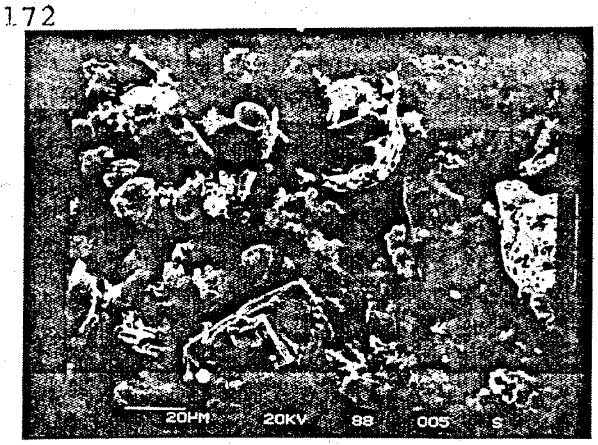


Fig. 1. X-ray powder diffraction patterns of char obtained after combustion at 950 °C with a residence time of 150 ms.

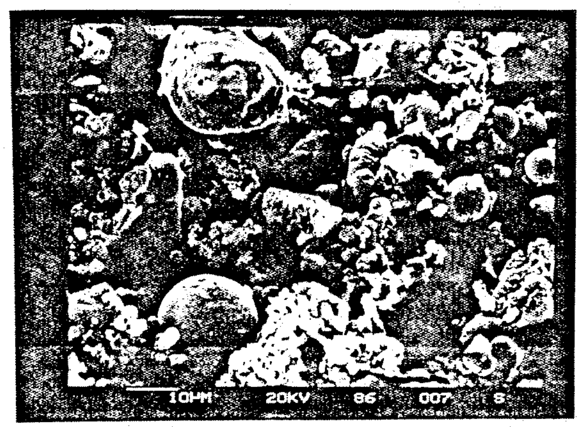


a) char 950° C, 150 ms.

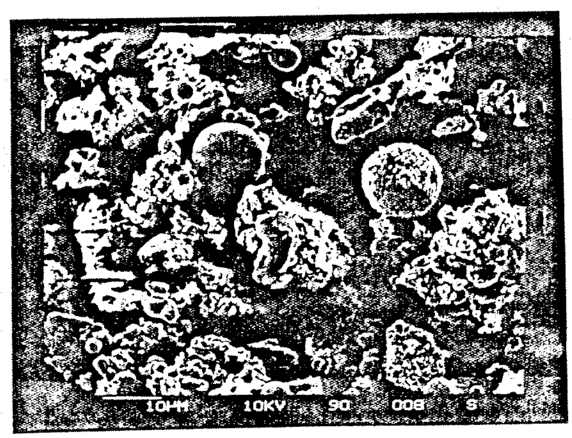


b) char 950° C, 500 ms.

c) char 1100° C, 150 ms.



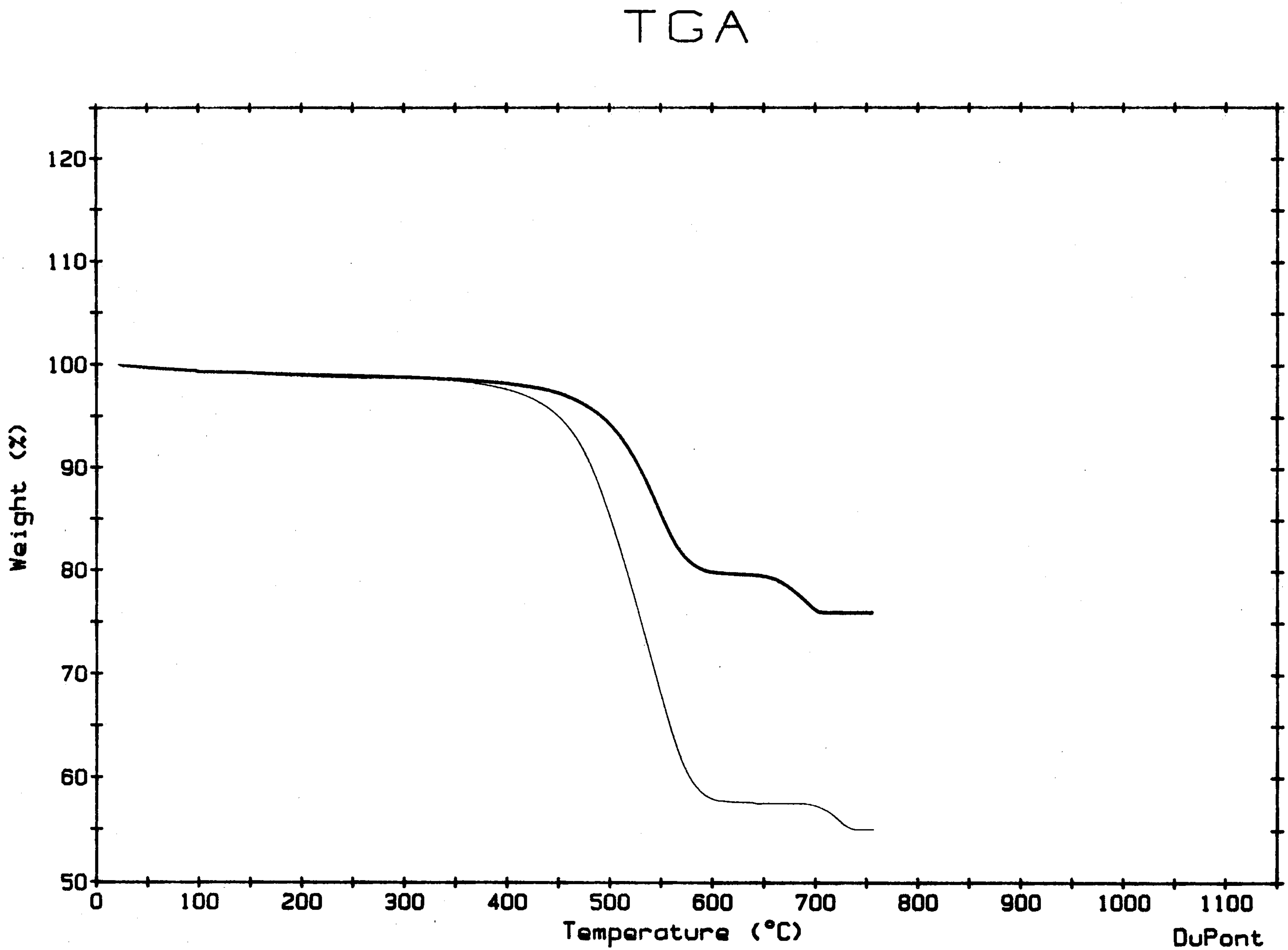
d) char 1100° C, 300 ms.



e) char 1100° C, 500 ms.

Fig. 2 . SEM Photomicrographs of chars obtained at 950° C and 1100° C with different residence times.

Fig. 3. TGA curves of chars obtained after combustion at 950°C with different residence times.



TITLE: Evaluation of IGCC for Canadian Utility Applications

AUTHOR: M.M. McDonald

AFFILIATION: TransAlta Utilities Corporation
110 - 12 Avenue S.W.
Calgary, Alberta
T2P 2M1

ABSTRACT: A study was carried out to provide performance and cost data to allow Canadian utility companies to evaluate the merits of Integrated Gasification Combined Cycle (IGCC) power generation.

The study included gasification process selection, conceptual design and performance estimation, and capital and operating cost estimation for three Canadian sites. They were at Point Aconi, N.S., Wesleyville, Ontario and Blackfoot, Alberta. The evaluation assumed bituminous Prince coal would be used at the Point Aconi plant, while eastern U.S. bituminous coal would be used at the Wesleyville power station and subbituminous Blackfoot coal would be used at the Alberta plant.

The Shell IGCC process was selected for a 250 MW option in Nova Scotia, while Texaco technology was chosen for the 785 MW Ontario case, and the Dow Process was selected for the 525 MW Alberta case. Sulphur emissions from the plants were estimated to be in the range of 30 to 50 ng/J, and NO_x emissions were approximately 50 ng/J. Plant efficiencies were 38.2%, 38.5% and 37.6%, respectively, for the N.S. Ontario and Alberta cases.

Capital costs for the three sites were estimated to be in the range of $1,700 to $2100/kW.

The Coal Association of Canada IGCC Feasibility Study Update

by

R. F. Geosits and L. A. Schmoe
Bechtel Power Corporation

J. H. Westsik
Bechtel Research & Development

Brian W. Raymond, Project Manager for
The Coal Association of Canada

For presentation at the
CANADIAN ELECTRICAL ASSOCIATION
Thermal and Nuclear Power Section
1991 Fall Meeting

ABSTRACT

The Coal Association of Canada recently completed a feasibility study which provides cost and performance data for an IGCC plant, using the Shell Coal Gasification Process and the advanced GE-7F gas turbine. The study is based on a site-specific design using Highvale Western sub-bituminous coal, and includes performance and cost estimates for several alternate coals and design configurations. The study highlights the ability of IGCC technology to reduce gaseous emissions, liquid wastes and solid wastes. This paper summarizes the results of this study and provides an updated status report on an advanced IGCC demonstration plant in Canada.

Keywords: IGCC, Coal Gasification, Combined Cycle

NOTICE

This technical paper and its contents do not necessarily reflect the views of the companies or government agencies that contributed funding to this project.

ACKNOWLEDGEMENT

The research project for which this report is submitted was funded within the scope of the Western Canadian Low-Sulfur Coal to Ontario Program by:

- The Province of Alberta;
- The Province of Saskatchewan
- The Province of Ontario
- The Federal Government of Canada through the Office of Western Economic Diversification;

and, was administered by the Province of Alberta and the Coal Association of Canada.

Other contributors were:

- The Province of British Columbia
- The Canadian Electrical Association
- Cape Breton Development Corporation (DEVCO)
- Nova Scotia Power Corporation
- SaskPower Corporation
- Crows Nest Resources Ltd.
- Manalta Coal Ltd.
- Fording Coal Limited
- Luscar Limited
- TransAlta Utilities Corporation
- Edmonton Power Limited
- Westar Mining Ltd.
- Alberta Power Limited
- Ontario Hydro

1.0 INTRODUCTION

Canada has very large reserves of low cost coal. Environmental concerns and recent discussion on the Green Plan, may require that new ways be found to utilize these reserves with reduced impact to the environment. Integrated coal gasification combined cycle (IGCC) is one of several new technologies which show promise in efficiently generating power while minimizing emissions of gaseous, liquid and solid effluents. A large and diverse group of Canadian partners has undertaken to evaluate the feasibility of this technology.

Phase I of this project is a feasibility study to predict the performance and cost of a 240 MW IGCC facility tailored to Canadian resources and requirements. The Coal Association of Canada (CAC) managed the feasibility study and Bechtel Canada Inc. was selected to perform the engineering. The specific objectives of this study are to:

- develop a cost-effective design which has a target 40% thermal efficiency based on the Higher Heating Value of the coal or lignite feed to the plant.
- minimize plant emissions, solid waste, and liquid discharges.
- select a gasification process, gas turbine, and other system components which best meet the target efficiency and environmental objectives for several Western Canadian coals.
- develop a site and coal specific base case design in sufficient detail to allow preparation of a $\pm$ 20% capital cost estimate.
- estimate performance and cost for several alternate coals and alternate design configurations including pipelining fuel gas to a combined cycle plant in a remote location and potential recovery of CO_2 from the fuel gas for use in enhanced oil recovery (EOR).

The results from the feasibility study can be used by the various association members to evaluate the merits of an IGCC plant at their site.

The feasibility study was completed in mid-September 1991, and is based on target plant operation by the end of 1996 (Reference 1). The early results of the study including technology selection and a description of the process components are discussed in a recent paper presented by the Coal Association of Canada, (Reference 2). This paper discusses the performance and preliminary cost projections for the various coals and design options.

2.0 STUDY BASES

A base case IGCC plant design was developed in sufficient detail to support the $\pm$20% capital cost estimate. In addition, six alternate design cases were evaluated as sensitivity cases to the base case design. Table 1 shows common design criteria and Table 2 summarizes the design requirements for the various cases. Tables 3 and 4 show the analysis of the four coal and lignite feedstocks assumed for this study. Feedstocks range from high grade bituminous coal, normally sold on the export market, to low sulfur strip-mined sub-bituminous coals typified by Highvale and Brooks coal, and lignite with high moisture and ash content.

A technology screening evaluation led the CAC to select the Shell Coal Gasification Process, a conventional air separation unit and the advanced GE-7F gas turbine as the key hardware components. Other design bases selected during early screening studies include:

- a design ambient temperature of 6°C
- a moderate degree of thermal integration
- use of hot boiler feedwater and minimal syngas firing to provide the heat required for coal and lignite drying

The gasification plant is sized to provide the fuel required by the gas turbine at 6°C, which is close to the average ambient temperature. Figure 1 shows the corresponding projected gas turbine output. The gas turbine is flat rated at ambient temperatures below 6°C (constant output limited by the capacity of the gasifier) and output drops off at higher ambient temperatures (limited by air flow through the gas turbine air compressor).

Table 1
Common Design Criteria

Target 40% efficiency
Nominal 250 MW generating capacity
85% Equivalent Availability Factor
Minimum steam superheating in the gasification plant
Zero discharge of process water
10,100kPa 538°C/538°C reheat steam cycle
ASME Code
Ambient temperature range: mean - 21 to +27°C
extreme - 52 to +46°C
Atmospheric pressure 93 kPa (94.7 for the Shand site)

Table 2
Study Cases and Design Criteria

Case Description	Base Case	Grassroots Plant	CO_2 Recovery	Remote Combined Cycle	Shand	Brooks	British Columbia
Coal	Highvale	Highvale	Highvale	Highvale	Shand Lignite	Brooks	B.C. Bituminous
Location	Keephills	Keephills area	Keephills	Keephills or Genesee & Rossdale	Shand	Brooks	-
CO_2 Recovery	Pilot Plant 200 tonne/day	-	50% Recovery	-	-	-	-
Cooling System	Pond	Tower	Pond	Pond & River Water	Tower	Tower	Tower
Shared Services	Yes	No Grassroots	Yes	Yes	Yes	No Grassroots	No Grassroots
Other Requirements	-	-	Liquid CO_2 Recovery	Syngas Pipeline	-	Construction Camp	-

Table 3
Coal Analysis

Analysis	Highvale Average	Highvale Std. Deviation	Shand Typical	Shand Range
Proximate Analysis: (Wt% - As Received)				
Moisture	20.00	—	34.3	31.5-38
Ash	13.84	3.76	10.6	7.3-17.3
Fixed Carbon	38.42	2.90	29.2	26.36-32.57
Volatile Matter	27.74	2.29	25.9	23.99-28.13
Ultimate Analysis: (Wt% - As Received)				
Carbon	48.48	3.69	39.57	37.17-42.07
Hydrogen	3.27	0.54	2.42	1.93-2.69
Oxygen	13.49	3.19	11.92	9.93-15.29
Nitrogen	0.62	0.07	0.58	0.48-0.65
Sulphur	0.20	0.07	0.57	0.22-1.02
Chlorine	—	—	0.01	0.006-0.17
Ash	13.84	4.47	10.58	7.26-17.28
Moisture	20.00		34.35	31.59-38.06
Higher Heating Value (kJ/kg)	18,996.5	1.13	15,095	—
Hargrove Grindability Index:	53	12.6	48	40-57
Ash Analysis: (Wt%)				
SiO_2	54.64	8.51	39.76	31.0-54.35
Al_2O_3	21.29	4.22	19.74	13.8-24.4
Fe_2O_3	3.59	1.81	5.07	2.5-9.9
TiO_2	0.70	0.24	0.92	0.4-5.5
CaO	11.01	3.66	12.16	5.9-20.5
MgO	1.20	0.34	2.73	1.7-3.4
SO_3	3.12	1.04	10.30	6.1-16.5
K_2O	0.58	0.54	0.52	0.2-1.7
Na_2O	2.45	0.73	7.38	2.1-10.4
P_2O_5	0.19	0.08	0.43	tr-1.5
Undetermined	1.23		0.99	–
Ash Fusion Temperature (°C):	**Reducing**	**Oxidizing**	**Reducing**	**Oxidizing**
Initial Deformation	1,244	1,293	1,010-1,288	1,099-1,378
Softening	1,289	1,228	1,099-1,343	1,193-1,454 +
Hemispherical	1,336	1,371	1,118-1,387	1,218-1,454 +
Fluid	1,432	1,460	1,171-1,443	1,316-1,454 +

Table 4
Alternate Coal Properties

	Brooks Coal	British Columbia Coal
Proximate Analysis: (Wt%)		
Basis	as received	air dried
Moisture	17.4	0.33
Ash	11.4	19.23
Fixed Carbon	39.3	56.53
Volatile Matter	31.9	23.91
Ultimate Analysis: (Wt%)		
Basis	moisture & ash free	air dried
Carbon	75.5	79.4
Hydrogen	6.7	4.94
Oxygen	15.0	4.22
Nitrogen	1.7	1.55
Sulphur	1.0	0.78
Ash	NA	8.32
Moisture	NA	0.78
Higher Heating Value (kJ/kg)	23,115	26,379[(b)]
Hardgrove Grindability Index:	40[(a)]	90.3/85.3[(c)]
Ash Composition: (Wt%)		
SiO_2	55.23	55.78
Al_2O_3	24.28	22.43
Fe_2O_3	3.87	13.05
TiO_2	0.55	—
CaO	5.90	1.76
MgO	1.13	0.34
SO_3	3.64	—
K_2O	0.66	0.58
Na_2O	2.63	0.07
P_2O_5	0.15	1.86
Undetermined	−1.76	-

Notes:
(a) With 10.4% moisture
(b) As-received with 9% moisture
(c) At 20% and 8% ash respectively

Ash Fusion Temperature (°C):	Oxidizing	Reducing	Oxidizing	Reducing
Initial Deformation	1,274	1,246	1,289	1,310
Softening	1,327	1,282	1,380	1,390
Hemispherical	1,349	1,299	1,426	1,392
Fluid	1,463	1,457	1,450	1,435

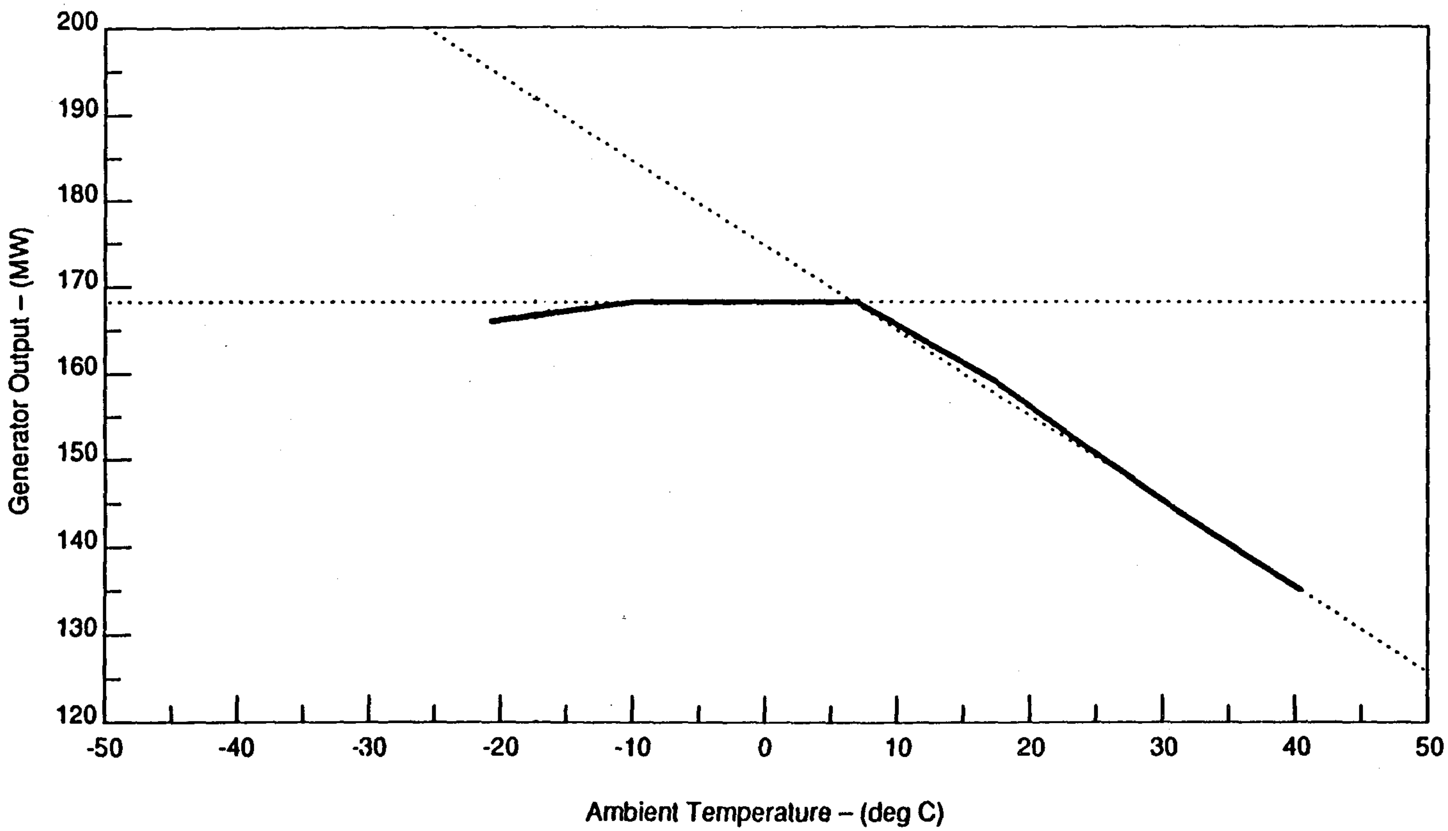


Figure 1 GE MS 7001F Gas Turbine Performance vs. Ambient Temperature

3.0 BASE CASE IGCC DESIGN

The various systems in the base case design are shown in the overall block flow diagram, Figure 2. The IGCC plant consists of a single Shell coal gasification and gas cooling train, and a single combined cycle power block. The power generation system includes a GE-7F gas turbine, a three pressure heat recovery steam generator, and a reheat steam turbine. A CO_2 removal pilot plant is also incorporated, to demonstrate the viability of extracting CO_2 and to make CO_2 available for sale to potential EOR users in a liquid form.

The IGCC plant is located close enough (1.3 km) to the Keephills plant to allow convenient access to the coal storage and makeup water systems. The cooling pond and switchyard are also shared with the Keephills plant. The site arrangement and major structures are shown in Figure 3.

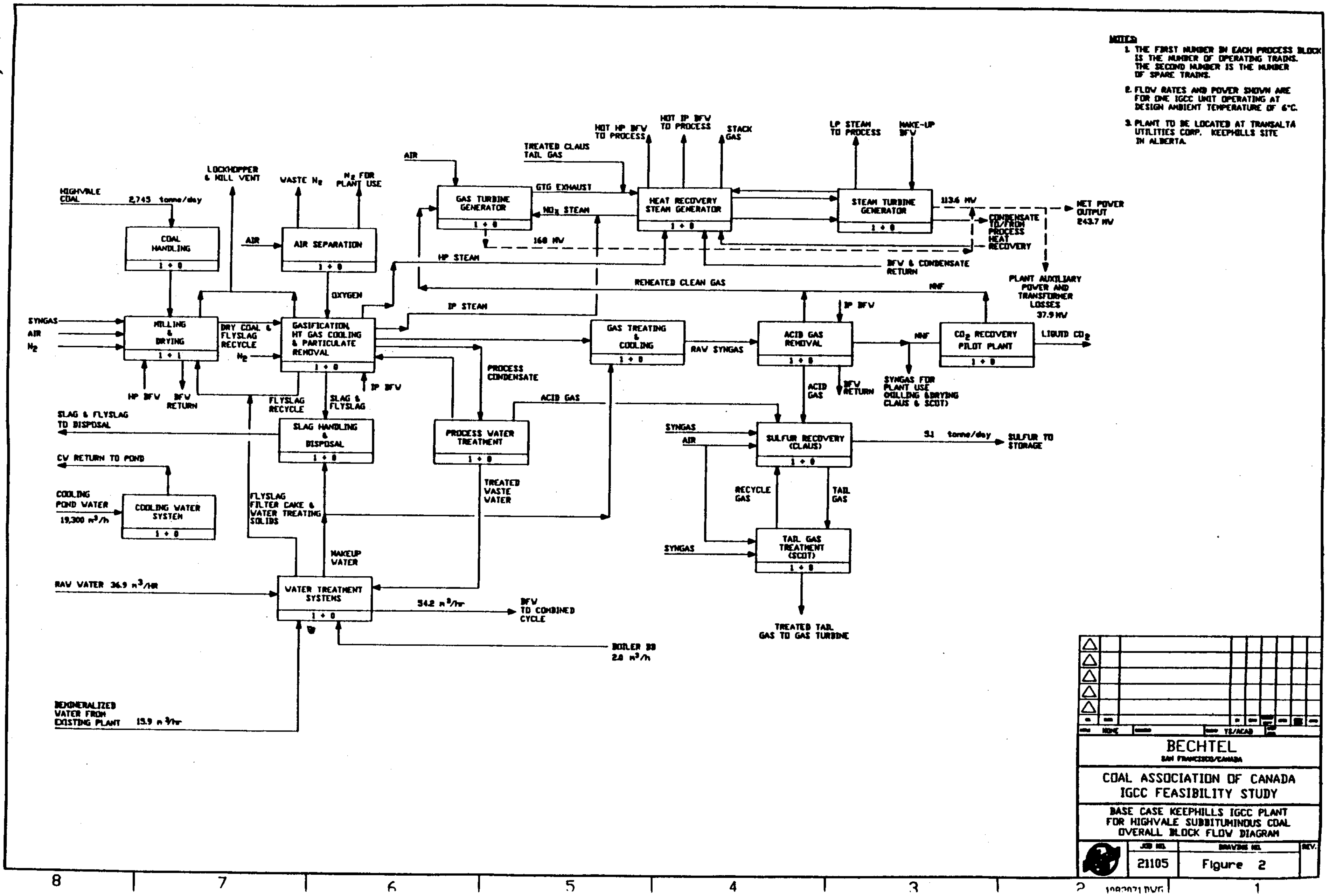
Shell International Petroleum Maatschappij B.V. provided performance and cost data for the Shell Coal Gasification Process. The data was based on Shell's design and performance database updated to reflect the results from the Deer Park demonstration plant and the Demkolec design. General Electric provided data on their 7F gas turbine which was also updated to reflect the operation at Virginia Power's Chesterfield 7 combined cycle power plant. Liquid Air Engineering provided data on a conventional high efficiency air separation unit.

Plant performance and auxiliary power requirements are summarized in Tables 5 and 6, respectively. At the design temperature point, the plant will produce a gross output of about 282 MW. The in-plant power consumption, including the ASU compressor load, is 38.0 MWe, yielding a net output of about 244 MW. The net heat rate is 8910 kJ/kWh (HHV), corresponding to a net overall efficiency of 40.4%. Thus the targeted 40% efficiency has been achieved. The Sankey Energy Distribution diagram in Figure 4 shows the various paths of energy dissipation. The cooling curve for the heat recovery steam generator is shown in Figure 5. Hot BFW is used for process heating and coal drying which tends to maximize the use of low level energy.

The net power output and plant efficiency as a function of the ambient temperature are shown in Figure 6. At temperatures below the ambient design point, the plant output is essentially flat. In this regime the output is limited by the availability of syngas. At temperatures above the design point, the generation capacity declines due to the temperature sensitivity of the gas turbine output. Plant efficiency at part load operation is shown in Figure 7.

The power output and efficiency calculated for the base case design do not represent maximum attainable values for the IGCC plant. GE has recently raised the claimed performance of the GE-7F turbines which would increase the power output from 168 MW to 182 MW. Also further thermal integration may yield performance benefits. Overall, the efficiency may increase by one to two points.

Analysis of the original plant configuration had a projected equivalent availability factor well below the targeted 85%. Additional redundancies in the coal pressurization system and spare gas filtering have raised the availability to 84%. Provisions for liquid oxygen and nitrogen storage would be required to raise the availability above 85%. However, liquid oxygen and nitrogen storage add significantly to the capital cost.


NOTES:
1. THE FIRST NUMBER IN EACH PROCESS BLOCK IS THE NUMBER OF OPERATING TRAINS. THE SECOND NUMBER IS THE NUMBER OF SPARE TRAINS.
2. FLOW RATES AND POWER SHOWN ARE FOR ONE IGCC UNIT OPERATING AT DESIGN AMBIENT TEMPERATURE OF 6°C.
3. PLANT TO BE LOCATED AT TRANSALTA UTILITIES CORP. KEEPHILLS SITE IN ALBERTA.
HIGHVALE COAL
2,743 tonne/day
COAL HANDLING
1 + 0
LOCKHOPPER & MILL VENT
WASTE N2
N2 FOR PLANT USE
AIR
AIR SEPARATION
1 + 0
OXYGEN
SYNGAS
AIR
N2
MILLING & DRYING
1 + 1
DRY COAL & FLYSLAG RECYCLE
N2
HP BFW
BFW RETURN
GASIFICATION, HT GAS COOLING & PARTICULATE REMOVAL
1 + 0
IP BFW
FLYSLAG RECYCLE
SLAG & FLYSLAG
SLAG & FLYSLAG TO DISPOSAL
SLAG HANDLING & DISPOSAL
1 + 0
CW RETURN TO POND
COOLING POND WATER
19,300 m3/h
COOLING WATER SYSTEM
1 + 0
FLYSLAG FILTER CAKE & WATER TREATING SOLIDS
MAKEUP WATER
RAW WATER 36.9 m3/HR
WATER TREATMENT SYSTEMS
1 + 0
54.2 m3/hr
BFW TO COMBINED CYCLE
BOILER BD
2.0 m3/h
DEMINERALIZED WATER FROM EXISTING PLANT
15.9 m3/hr
AIR
GAS TURBINE GENERATOR
1 + 0
GTG EXHAUST
NOx STEAM
160 MW
HP STEAM
IP STEAM
TREATED CLAUS TAIL GAS
HOT HP BFW TO PROCESS
HOT IP BFW TO PROCESS
STACK GAS
HEAT RECOVERY STEAM GENERATOR
1 + 0
LP STEAM TO PROCESS
MAKE-UP BFW
STEAM TURBINE GENERATOR
1 + 0
113.6 MW
NET POWER OUTPUT
243.7 MW
CONDENSATE TO/FROM PROCESS HEAT RECOVERY
BFW & CONDENSATE RETURN
REHEATED CLEAN GAS
MNF
PLANT AUXILIARY POWER AND TRANSFORMER LOSSES
37.9 MW
PROCESS CONDENSATE
PROCESS WATER TREATMENT
1 + 0
ACID GAS
TREATED WASTE WATER
GAS TREATING & COOLING
1 + 0
RAW SYNGAS
ACID GAS REMOVAL
1 + 0
IP BFW
MNF
CO2 RECOVERY PILOT PLANT
1 + 0
LIQUID CO2
ACID GAS
BFW RETURN
SYNGAS FOR PLANT USE (MILLING &DRYING CLAUS & SCOT)
SYNGAS
AIR
SULFUR RECOVERY (CLAUS)
1 + 0
51 tonne/day
SULFUR TO STORAGE
RECYCLE GAS
TAIL GAS
SYNGAS
TAIL GAS TREATMENT (SCOT)
1 + 0
TREATED TAIL GAS TO GAS TURBINE
BECHTEL
SAN FRANCISCO/CANADA
COAL ASSOCIATION OF CANADA
IGCC FEASIBILITY STUDY
BASE CASE KEEPHILLS IGCC PLANT
FOR HIGHVALE SUBBITUMINOUS COAL
OVERALL BLOCK FLOW DIAGRAM
JOB NO.
21105
DRAWING NO.
Figure 2
REV.
8
7
6
5
4
3
2
1
10B2071.DWG

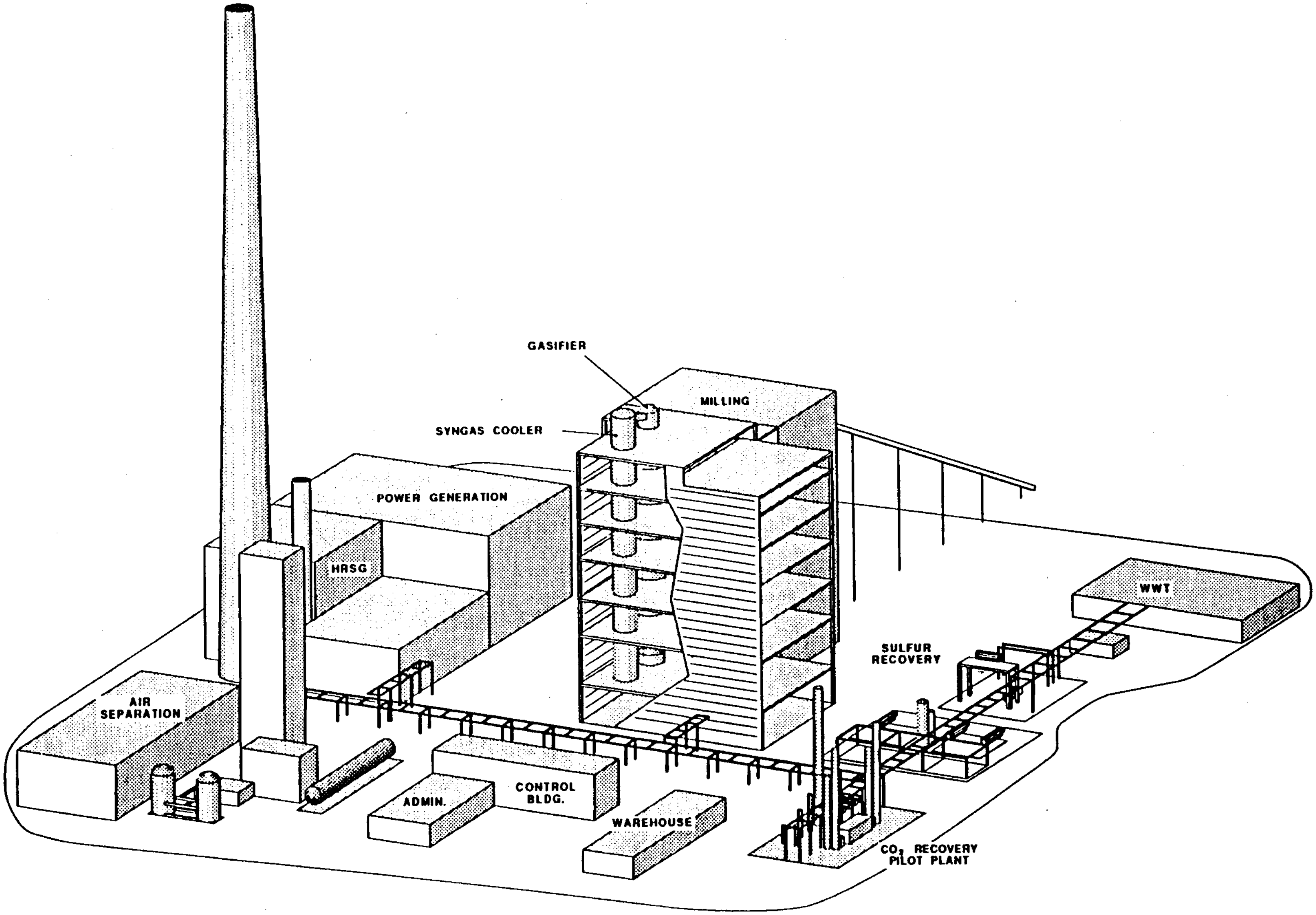


Figure 3 Base Case Design General Equipment Arrangement

Table 5
Base Case Design
Design Point Plant Performance Summary(1)

Fuel energy input GJ/h	2,173
Gas turbine gross output, MW	168
Steam turbine, gross output, MW	113.6
Total gross power, MW	281.6
In-plant consumption, MW	37.9
Net power to grid, MW	243.7
Net heat rate, (HHV), kJ/kW	8,910
Overall plant efficiency, %	40.4

(1) Best estimate based on Bechtel and vendor performance data. Does not represent commercial warranties.

Table 6
Base Case Design
In-Plant Power Consumption at Design Point

Plant Area	kW
Coal receiving and storage	200
Coal milling and drying	3,600
Air separation	24,800
Coal gasification(1)	3,400
Combined cycle	2,500
Cooling water system	1,500
Miscellaneous and transformer losses	1,900
Total	37,900

(1) Includes gasification, gas cooling, ash handling, acid gas removal, sulfur recovery, and wastewater treatment.

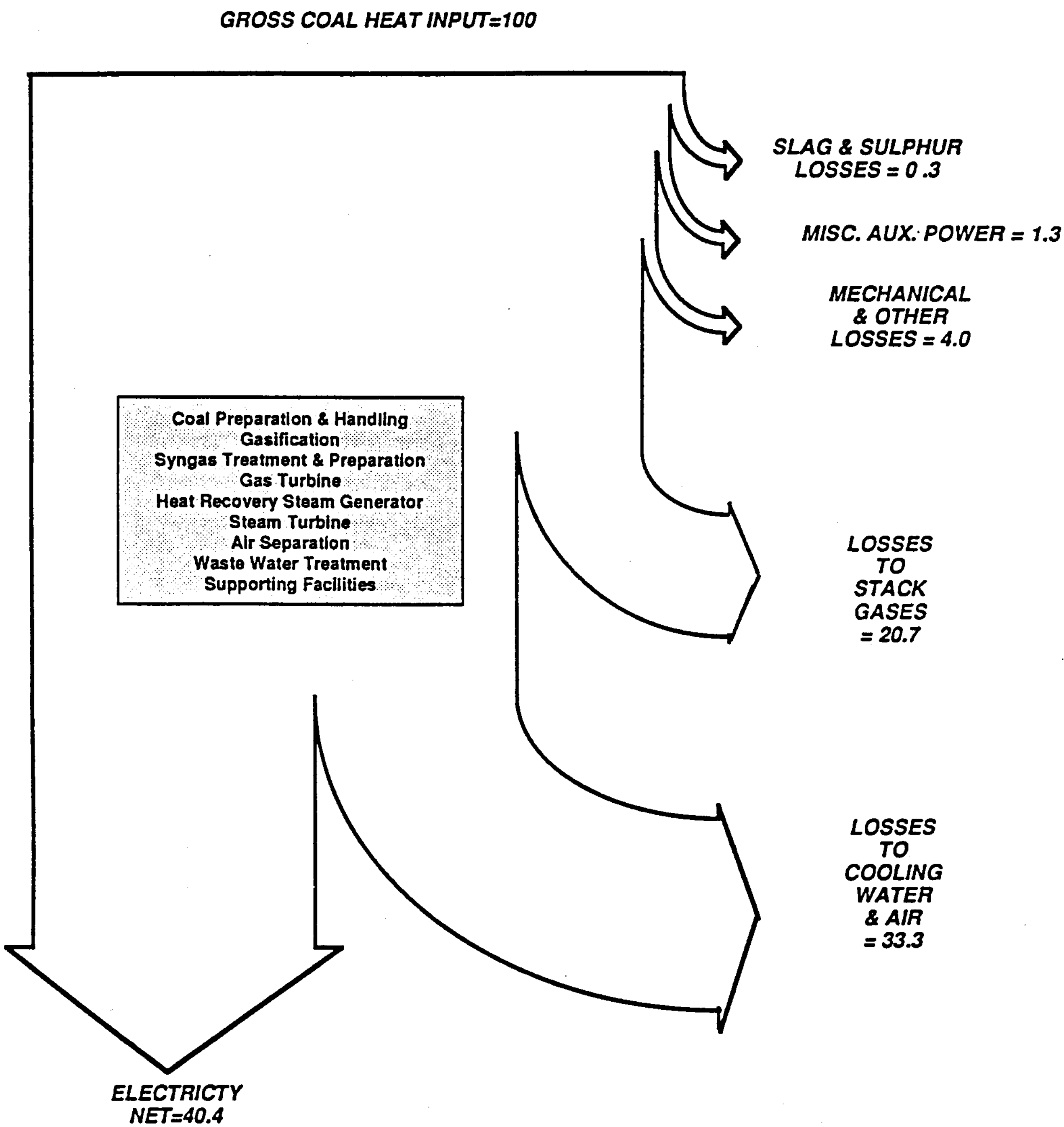


Figure 4 Base Case Design Sankey Energy Distribution Diagram

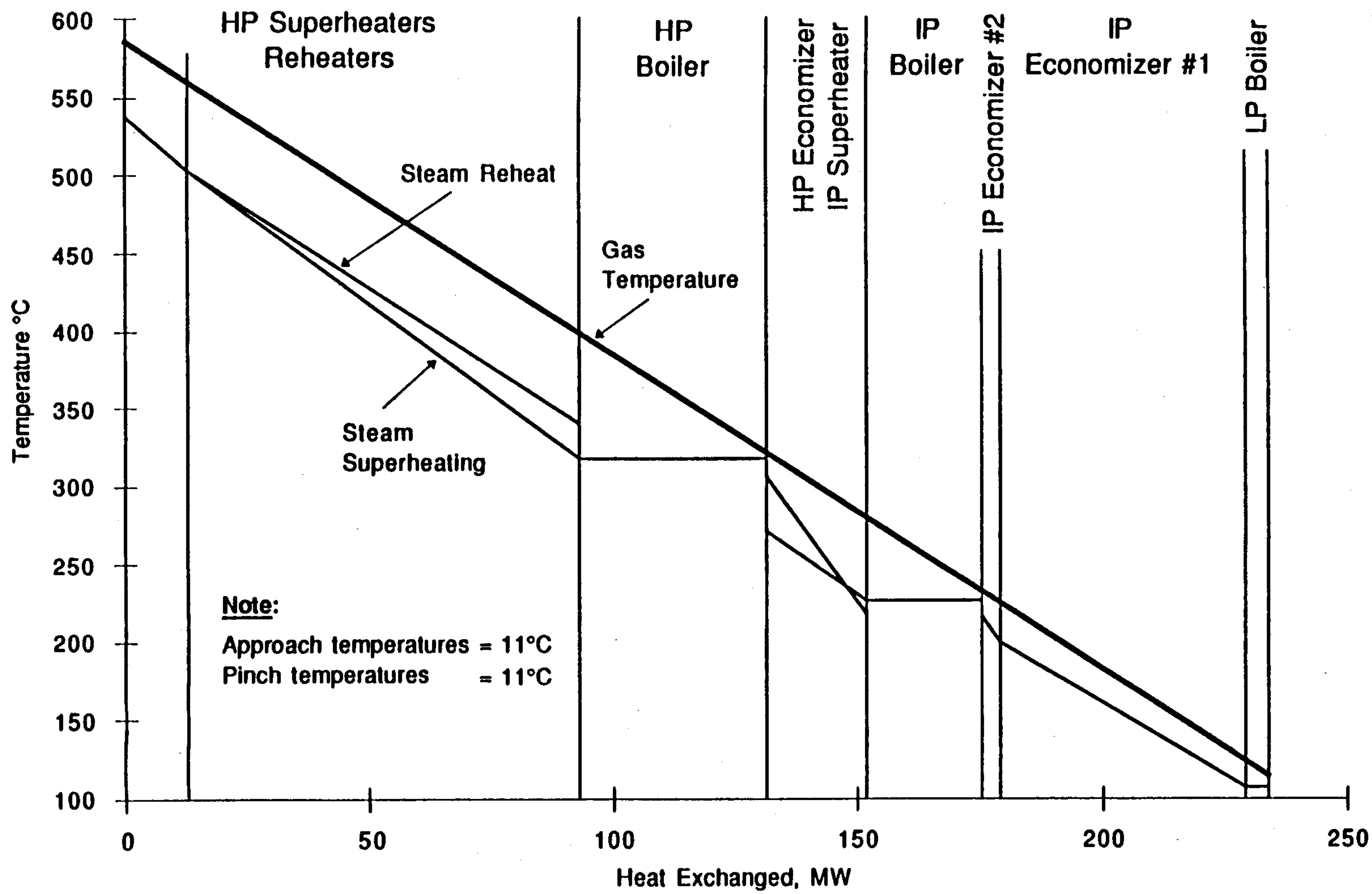


Figure 5 Base Case HRSG Cooling Curve

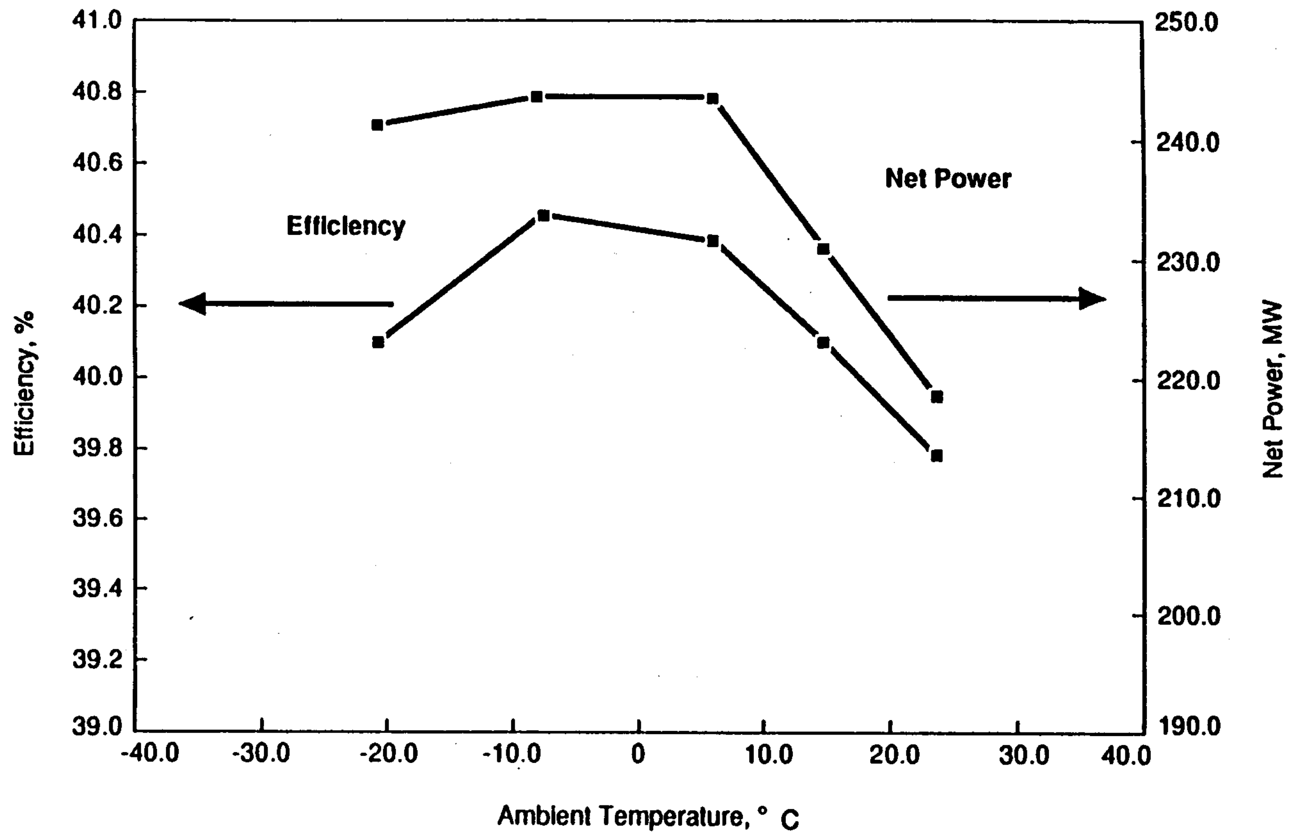


Figure 6 Base Case Design
Performance Versus Ambient Temperature

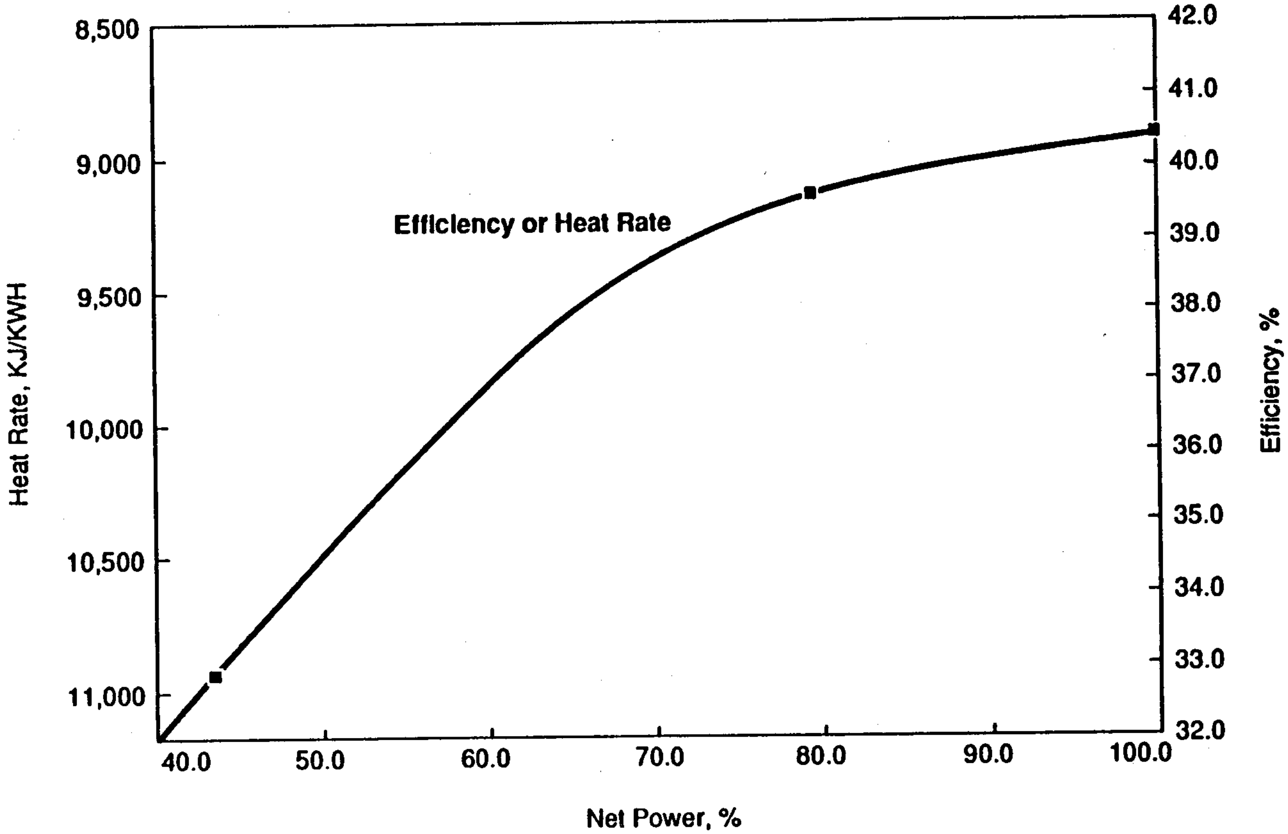


Figure 7 **Base Case Plant - Part-Load Operation IGCC Performance Versus Net Power Output**

4.0 BASE CASE PLANT EMISSIONS AND DISCHARGES

Important features of any IGCC plant design are low SO_x/NO_x emissions and high efficiency which results in lower CO_2 emissions. Estimated emissions and discharges for the base case design are shown in Table 7. Emissions for the alternate design cases are similar.

IGCC technology allows 99% recovery of potential SO_2 emissions as byproduct sulfur. Plant NO_x emission are limited to 51 ng/J, but developing designs using additional steam or nitrogen injection into the fuel gas, and SCR have the potential to reduce emissions to below 12 ng/J. High efficiency minimizes utilization of coal resources and inherently minimizes CO_2 emissions. The proposed CO_2 pilot plant would demonstrate the ability to recover CO_2 for enhanced oil recovery and give insight as to the economics of reducing CO_2 emissions below those of natural gas based power plants. Gasification at high temperatures eliminates VOC and gas purification limits particulates in the gas turbine exhaust to unburned carbon and condensables such as SO_3, which are formed during combustion. Carbon monoxide emissions are also estimated to be low.

Solid waste is limited to coal ash in the form of slag or fly slag, which leaves the process as a vitreous, non-leaching substance. All other waste material is recycled to extinction in the gasifier except for occasional maintenance-related substances which are recycled or sent to disposal. The base case plant is also designed for zero liquid waste discharge. All process water, boiler blow down, and effluent from makeup water treatment is treated and reused. This approach also minimizes the amount of water required by the process.

5.0 BASE CASE PLANT COSTS AND ECONOMICS

The capital cost of the base case plant was developed from detailed vendor input, from factoring cost estimates of similar scope and from Bechtel cost data. To achieve the desired confidence level, the plant was subdivided into 26 areas and detailed material and labor costs were estimated for each of these areas. A stochastic approach was used to define the percentage of contingency needed to achieve 50% probability of overrun. The total capital requirement (TCR) for the base case demonstration plant (exclusive of allowance for funds during construction - AFUDC) was estimated to be about \$595 million, or a specific cost of \$2440/kW. Components of the TCR are shown in Table 8. The capital costs are about 10 percent higher than those estimated in previous studies (Reference 3). The difference is largely attributed to higher estimated costs for the gasification plant. Operating and maintenance costs are summarized in Table 9. The plant economics and financial criteria are summarized in Table 10.

Table 7
Base Case Design
Summary of Estimated Air Emissions and Discharges

AIR EMISSIONS	UNITS	QUANTITIES
Stack Gas	kg/hr	1,606,573
	m^3/s	532
NO_x(as NO_2)	ppmvd	48
	tonnes/day	2.65
	ng/J	51
SO_2	ppmvd	1.3
	tonnes/day	0.10
	ng/J	2.0
CO	ppmvd	18
	tonnes/day	0.61
	ng/J	12
CO_2	Vol % (dry)	9.2
	tonnes/day	4,869
	ng/J	93,441
Particulates	tonnes/day	0.22
	ng/J	4.2
Unburned Hydrocarbons (UHC)	tonnes/day	0.05
	ng/J	0.89
Dryer and Lock Hopper Vents	kg/hr	91,930
	m^3/s	33
CO_2	Vol % (dry)	0.54
	tonnes/day	14.9
	ng/J	28.6
Particulates	tonnes/day	0.11
	ng/J	2.1
OTHER DISCHARGES		
Slag and Fly Slag	tonnes/day (dry)	387
	ng/J	7,428
Sulfur Byproduct	tonnes/day	5.1
	ng/J	98

Table 8
Total Capital Requirements
(April 1991, Canadian Dollars)

Description	$ Million	$/kW
Total plant cost	532.7	2186
Taxes and duties	8.4	34
Initial catalyst and chemicals	1.5	6
Spare equipment to increase availability	9.8	40
Owner's costs	42.2	173
Total capital requirements	594.6	2439

Notes:
1. Costs are in April 1991 Canadian dollars.
2. Capital cost estimates have a projected accuracy of ±20%.
3. Contingency of 14.1% was calculated from Monte Carlo analysis to achieve 50% probability of over/underrun.
4. Total capital requirements exclude AFUDC.

Table 9
Base Case Plant O&M Cost Summary
(April 1991, 1000's Canadian Dollars)

Description	Unit	Cost
Fixed Operating Costs	$1000/yr	8,510
	$/kW-yr	34.9
	mills/kWh **	4.98
Variable Operating Costs	$1000/yr *	14,060
	mills/kWh *	6.58
Sulfur By-Product Credit	$1000/yr *	−140
	mills/kWh *	−0.07
Net Variable O&M,	$1000/yr*	13,900
	$1000/yr**	11,100
	mills/kWh *	6.52
Total O&M (Excluding Fuel)	$1000/yr *	22,400
	$1000/yr**	19,600
	mills/kWh *	11.50
Fuel Cost ***	$1000/yr *	11,981
	$1000/yr**	9,585
	mills/kWh	5.61
Total O&M	$1000/yr *	34,400
	$1000/yr **	29,200
	mills/kWh **	17.10

Numbers are rounded off to three significant figures
* Based on 100% capacity factor
** Based on 80% capacity factor
*** Based on a coal price of $12/tonne

Table 10
Summary of Plant Economics

Total capital required (M$)	1991 $	594,600
Total capital expenditure (TCE)*(M$)	1991 $	642,852
	1996 $	848,176
Cumulative Present Value of Revenue Requirements (M$)	Discounted to 1991	819,499
	Discounted to 1996	1,444,238
Levelized Cost (Mills/kWh)(1)	Real 1991 (constant $)	59.8
	Real 1996 (constant $)	78.9
Levelized Cost (Mills/kWh)(1)	Nominal 1991 (current $)	90.1
	Nominal 1996 (current $)	118.8

Financial Criteria

Plant Life (book life)	20 years
Tax Life	20 years
Tax Rates	
Federal	28.84%
Provincial	15.00%
General Inflation	5.7% per year
Capital Cost Escalation Rate	5.7% per year
Fuel Escalation Rate	6.7% per year
Capital Cost Structure	
Equity	39%
Debt	42%
Preferred Stock	19%
	100%
Cost of Capital	
Return on Equity	14.0%
Interest on Debt	11.5%
Return on Stock	9.0%
Discount Rate for Levelizing	12.0%
Basis for Levelizing Cost of Electricity	All taxes paid, no income tax rebate, 80% plant operating factor, in service 1996.

* TCE includes as spent dollars and AFUDC

6.0 ALTERNATE DESIGN CASES

Similar performance and cost estimates were developed for the alternate design cases. Plant performance and cost data for these cases are summarized in Table 11. The requirements of all cases could be met without major changes in cost and performance compared to the base case design. Of all the cases, the highest net power output and best heat rate was achieved with the British Columbia coal. This case also had the lowest capital cost and cost of electricity. The remote gasification and 50% CO_2 recovery cases resulted in lower power output and less attractive heat rates. The remote gasification case required the highest capital investment and cost of electricity. However, potential savings through sale of byproduct CO_2 or integration of the remote combined cycle with a district heating and cooling system could offset higher capital costs.

7.0 IMPLEMENTATION PLANS

The project schedule, shown in Figure 8, along with construction labor and plant staffing requirements, were developed for the base case demonstration project. Approximately four years will be required to engineer, construct and commission the base case plant. Field work will require about 2.5 years. Potential sponsors of the project indicated a need for additional generating capacity by 1995/96.

8.0 FUTURE WORK

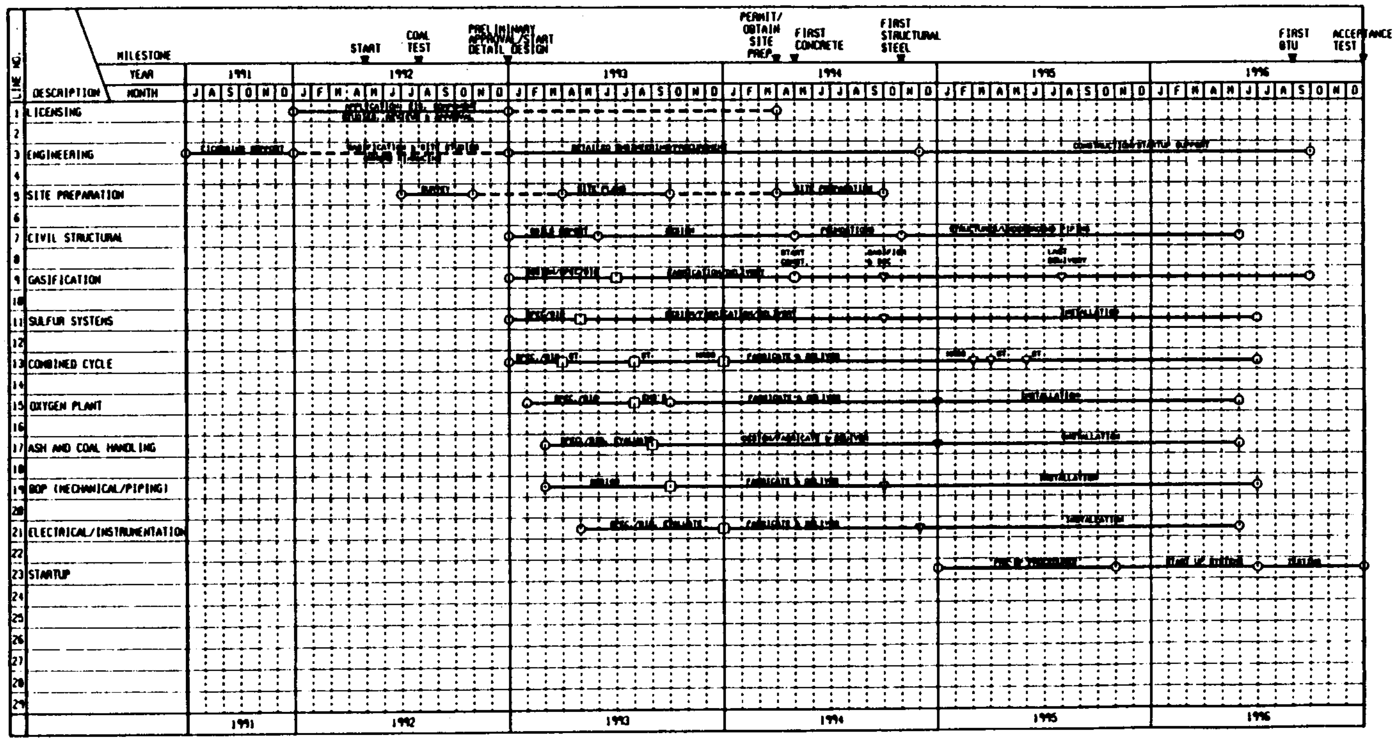
In order to facilitate the plant commissioning by the end of 1996 and to minimize technical and economic risks of the project, the study suggests that a potential project sponsor consider the following action items:

- Monitor the development of the Shell gasifier and Demkolec project with the objective of reducing technical, scaleup and economic risks
- Clearly define the properties of the design and any alternate coals
- Define the value of emission offsets
- Re-evaluate plant integration alternatives.
- Carry out gasification and coal drying tests
- Review CO_2 recovery options and markets
- Follow gas turbine developments
- Develop implementation plans for the Phase II efforts
- Develop detailed plans for demonstration testing

Table 11
Performance and Economic Summary

DESCRIPTION	Highvale (Base Case)	Highvale (Grassroots)	Highvale (CO2 Recovery)	Highvale (Remote)	Shand	Brooks	British Columbia
Net Plant Output, MW	243.7	243.2	225.2	229.1	231.4	242.6	244.3
Plant Efficiency, %	40.4	40.3	36.3	38.0	39.6	40.8	42.0
Total Plant Cost (M $), 1991 $	533,000	540,000	576,000	587,000	539,000	528,000	521,000
Total Plant Cost ($/kW), 1991 $	2,190	2,220	2,560	2,560	2,330	2,180	2,130
Total Capital Required (M $), 1991 $	595,000	602,000	644,000	652,000	603,000	590,000	583,000
Total Capital Expenditure (TCE)* (M $), 1991 $	643,000	651,000	696,000	705,000	651,000	638,000	630,000
1996 $	848,000	858,000	918,000	930,000	859,000	842,000	831,000
Total O&M, mills/kWh	17.11	17.43	10.85	18.91	18.17	17.00	16.89
Cumulative Present Value of Required Revenue (M $), Discounted to 1991	819,000	830,000	770,000	884,000	829,000	813,000	805,000
Discounted to 1996	1,444,000	1,463,000	1,357,000	1,558,000	1,461,000	1,432,000	1,419,000
Levelized Cost (mills/kWh), Real 1991	59.8	60.7	60.8	68.6	63.7	59.6	58.6
Real 1996	78.9	80.1	80.2	90.5	84.0	78.6	77.3
Levelized Cost (mills/kWh), Nominal 1991	90.1	91.4	91.6	103.4	95.9	89.7	88.3
Nominal 1996	118.8	120.6	120.9	136.4	126.6	118.4	116.5

* TCE includes as spent dollars and AFUDC



LEGEND

- ○ START OR FINISH ACTIVITY
- □ PURCHASE ORDER AWARD
- ▽ MAJOR MATERIAL/EQUIPMENT DELIVERY
- ▼ MILESTONE DESIGNATION
- --- NO ACTIVITY

SPEC. SPECIFICATION
EIS ENVIRONMENT IMPACT STATEMENT

ISSUED FOR DRAFT REPORT

BECHTEL
SAN FRANCISCO

IGCC FEASIBILITY STUDY

PROJECT SCHEDULE

JOB NO.	DRAWING NO.	REV
21185	Figure 8	A

9.0 PLANS FOR FURTHER DEVELOPMENT

The CAC involvement in this project is complete. The CAC has received proposals and requests for support of four projects. These proposal were received from:

- SaskPower
- TransAlta and Alberta Power
- Edmonton Power
- New Brunswick Power

The CAC decided not to support any single project. Individual sponsors could proceed with the development of their proposed project. Sponsors may solicit the financial support of federal and provincial governments, along with other interested investors. Hopefully this will lead to Phase II, (detailed design) and Phase III, (construction and operation) of an advanced IGCC demonstrations plant.

While it is clear that this technology can reduce emissions and improve efficiency, the initial investment and operating cost developed based on the design criteria assumed for this study, will be higher than that of a pulverized coal fired plant. The relative levelized cost of electricity will depend on utility specific economic factors and projections such as the cost of money, cost of fuel, escalation, etc.

Also, there are likely to be significant improvements in gasification and gas turbine design which can reduce the capital cost of future IGCC systems, thereby making IGCC an increasingly attractive economic option. A similar sized IGCC project using a Shell gasifier is being built in Holland. The industry should continue to monitor technology developments, review and evaluate the various aspects of this study in light of company-specific criteria, and evaluate potential cost saving design modifications and alternatives.

10.0 REFERENCES

1- Feasibility Study of an Integrated Gasification Combined Cycle Power Plant for Western Canada prepared for the Coal Association of Canada by Bechtel Canada Inc. - September 1991

2- Integrated Gasifier Combined Cycle Plant Demonstration for Western Canada by Brian W. Raymond and Dr. Giacomo Capobianco Presented at the Annual Meeting of the Canadian Institute of Mining and Metallurgical Engineers in Vancouver, April 1991.

3- Canadian Electrical Association, "Evaluation of Integrated Coal Gasification Combined Cycle for Canadian Utility Applications", R&D Project 811 G 659, December 1989.

TITLE: Use of Gypsum as Flocculent for Treatment of Mine Effluent Waters

AUTHOR: C. Bateman

AFFILIATION: TransAlta Utilities Corporation

ABSTRACT: The geological materials that overlie plains coal seams contain a high proportion of montmorillinite clays. As precipitation and groundwater come into contact with disturbed overburden materials at operating mines, these colloidal clays become suspended. This leads to mine effluent waters that are severely contaminated with suspended sediment. The dominance of sodium ions in the groundwater exacerbates the problem by causing the clay particles to disperse. This results in a stable colloidal emulsion that will not settle.

TransAlta Utilities Corporation and the Alberta Office of Coal Research and Technology developed a phased work program to investigate the use of gypsum as a low-cost, environmentally safe means of flocculating suspended sediment from mine effluent before it was released to surface watercourses.

The first phase of the current program was designed to determine key parameters for design of a field-scale pilot-testing facility. These are the dissolution rates of gypsum in the mine effluent, and the settling rates of clays in treated waters. These parameters were studied at the Alberta Research Council over a range of gypsum concentrations and expected seasonal temperatures. A preliminary design for a pilot test facility was also carried out by Monenco Consultants Limited as a part of Phase I. The pilot facility will supply critical data for development of an effluent-flocculation and sludge-management program for operational mines.

An overview of current coal-mine effluent management at Canada's largest coal mine will be presented. Changes in the mining procedures that are planned for the mid-90s, and their effect on the current mine-water handling system will be reviewed. The importance of the development of a new approach to mine-effluent handling and management, and the effect on future operations will be discussed.

TITLE: LNS-CAP Project

AUTHOR: B. Simonson

AFFILIATION: TransAlta Resources Investment Corporation
110 - 12 Avenue S.W.
Calgary, Alberta
T2P 2M1

ABSTRACT: The Low NO_x/SO_x Burner (LNSB) uses a simple, but innovative combustion process to burn pulverized coal at high temperatures to achieve effective, low-cost control of sulphur dioxide (SO_2) and nitrous oxides (NO_x) emissions. Whereas the sulphur present in coal is captured with calcium and retained in the fly ash or molten slag, which are removed from the burner, the nitrous oxides are converted to harmless elemental nitrogen in the burner.

The LNSB evolved from theoretical concepts developed at Rockwell International in California in 1979. Bench-scale tests confirmed the concepts, and development at Rockwell continued from 1980 to 1986, using a 25 MBtu/h pilot-scale facility. This work was supported by a consortium of interested utility companies, including TransAlta.
In 1986, TransAlta acquired the technology from Rockwell and continued testing high-sulphur coal at the Rockwell facility from 1987 to 1988.

A feasibility study sponsored by TransAlta Resources Investment Corporation, Esso Resources Canada Limited, Shell Canada Limited and the Alberta Office of Coal Research and Technology was carried out in 1988. It confirmed the applicability of the technology to steam generation for heavy oil recovery, and recommended the construction of a 50 MBtu/h plant for demonstration testing of the LNSB.

Project engineering for the subsequent Low NO_x/SO_x Coal Application Pilot (LNS-CAP) project at Cold Lake started in October 1988.

The LNS-CAP is a stand-alone facility at Esso's Mahihkan plant site at Cold Lake. It uses boiler feedwater, gas and other utilities from the plant and fires a steam generator that was specially built for the project. High-pressure steam is returned to the Mahihkan plant. Detailed engineering of the project was completed in 1989 by Monenco Consultants Ltd. Construction and commissioning were completed in June 1990.

The objectives of the test program were:

- to demonstrate the ability of the LNSB to burn coal in a heavy oil recovery steam generator;
- to demonstrate the ability of the LNSB to control SO_x and NO_x emissions to federal and provincial government emission guidelines when firing Alberta subbituminous coals;
- to demonstrate the reliability and durability of auxiliary systems using conventional equipment; and
- to evaluate the ability of the LNSB to fire natural gas for short periods.

The LNS-CAP project was completed in September 1991. Test data are being analysed and reports prepared. NO_x control was demonstrated clearly, but sulphur capture has been more difficult to confirm.

Valuable experience was gained with regard to slag tapping, refractory design, fouling of the steam generator and the importance of maintaining accurate coal and air flow.

A Coal Fired Steam Generator Demonstration For Heavy Oil Recovery Operations

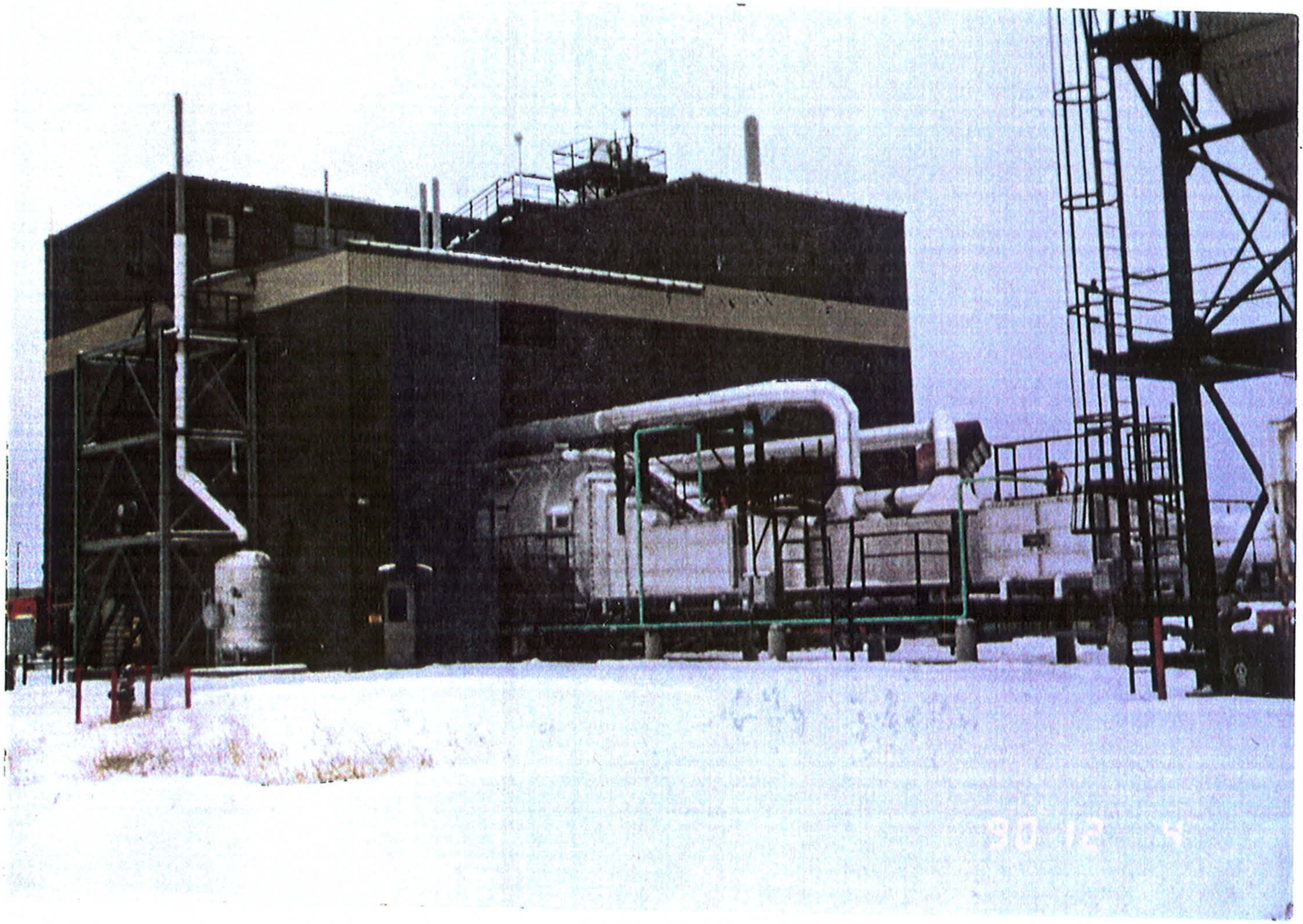
History

- Initially Developed by Rockwell - 1979
- TransAlta - 1986
- Feasibility Study
- LNS-CAP Project

Objectives

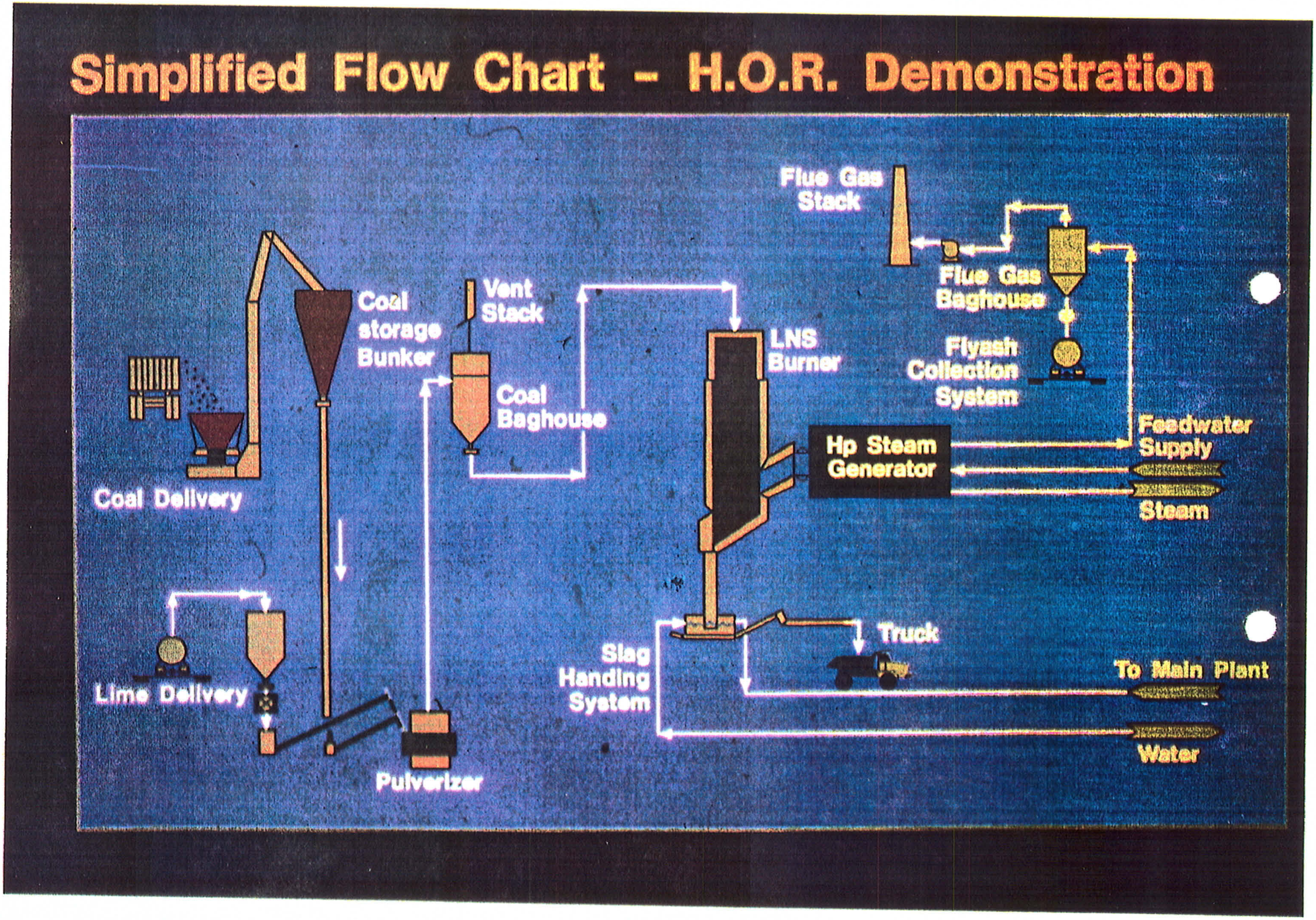
- Fire Coal in HOR Steam Generator Using LNS Burner
- Control NOx and SOx Emissions
- Prove Auxilliary Systems Performance

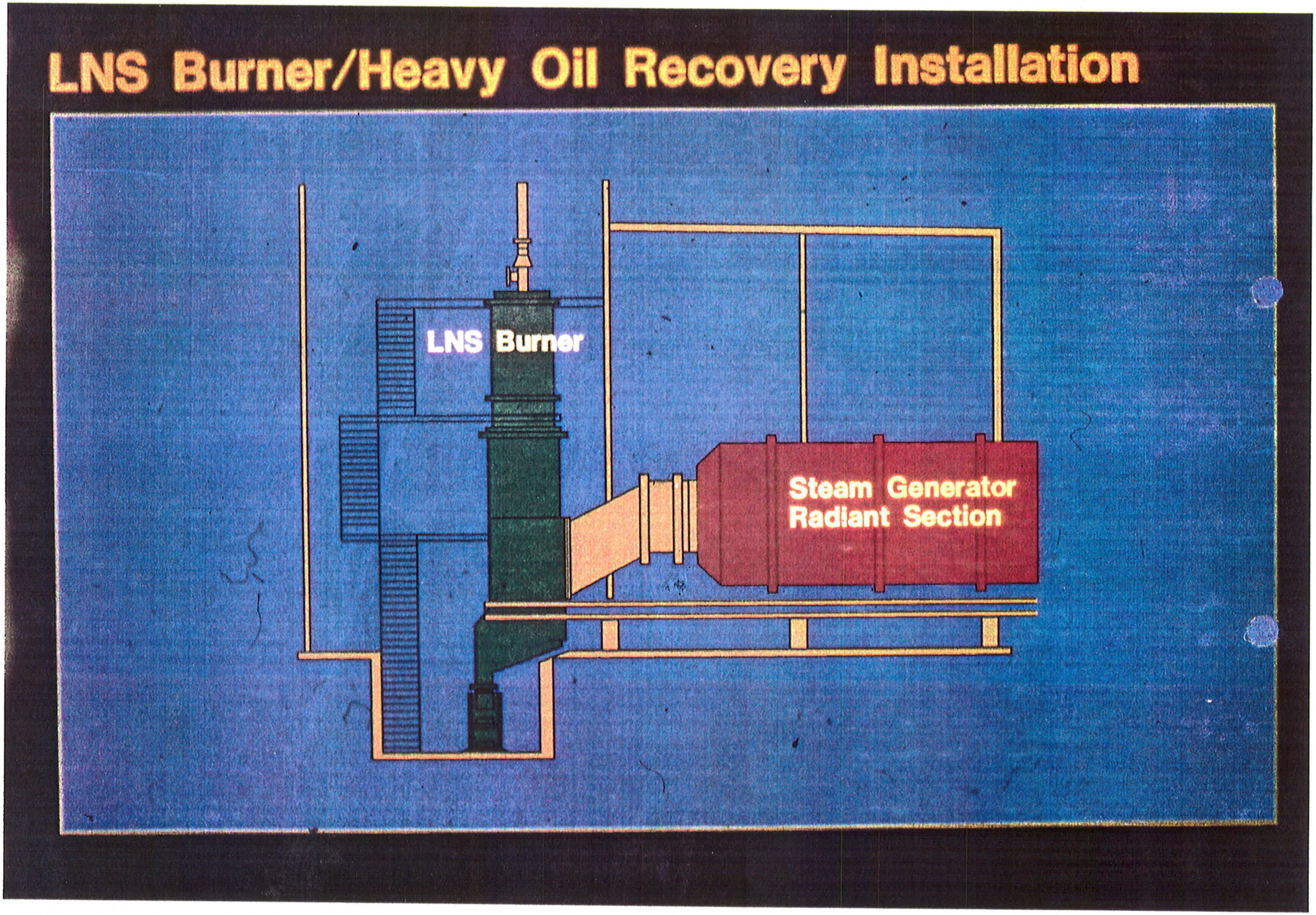
LNS-CAP



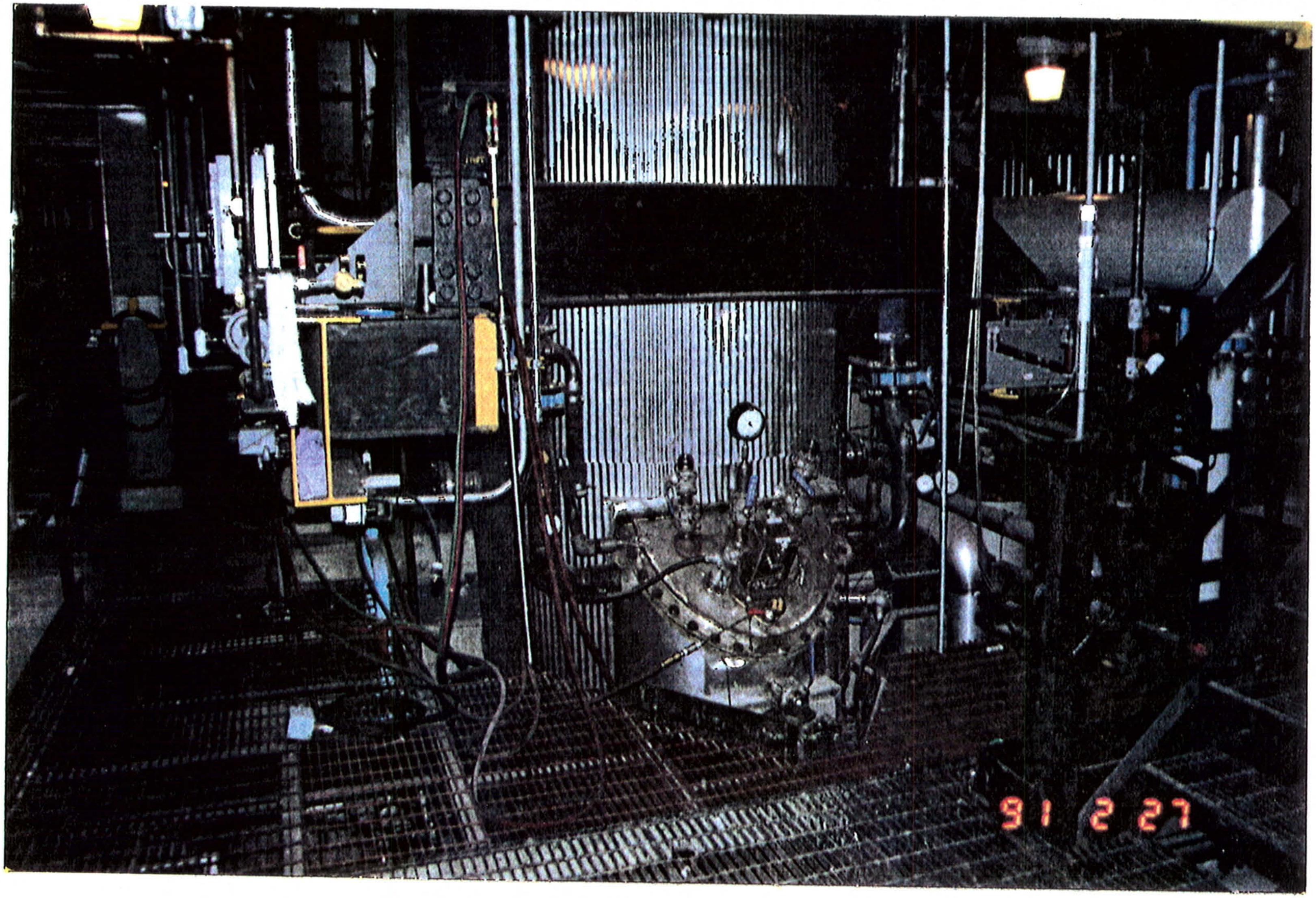
LNS-CAP Facility

TransAlta Resources Investment Corporation


Simplified Flow Chart - H.O.R. Demonstration
Flue Gas Stack
Flue Gas Baghouse
Flyash Collection System
Vent Stack
Coal storage Bunker
Coal Baghouse
LNS Burner
Hp Steam Generator
Feedwater Supply
Steam
Coal Delivery
Lime Delivery
Pulverizer
Slag Handing System
Truck
To Main Plant
Water

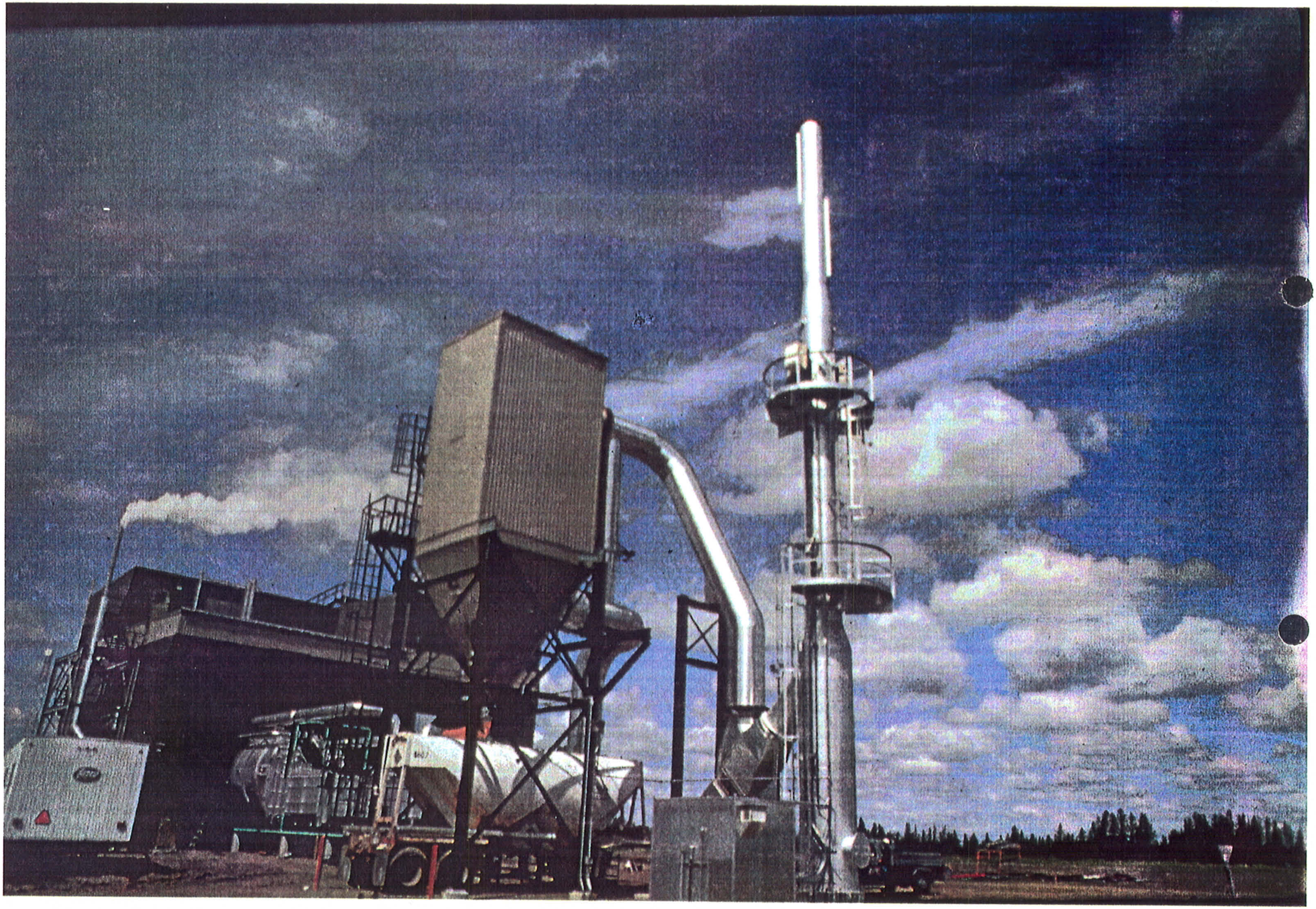

LNS Burner/Heavy Oil Recovery Installation
LNS Burner
Steam Generator
Radiant Section

LNS-CAP

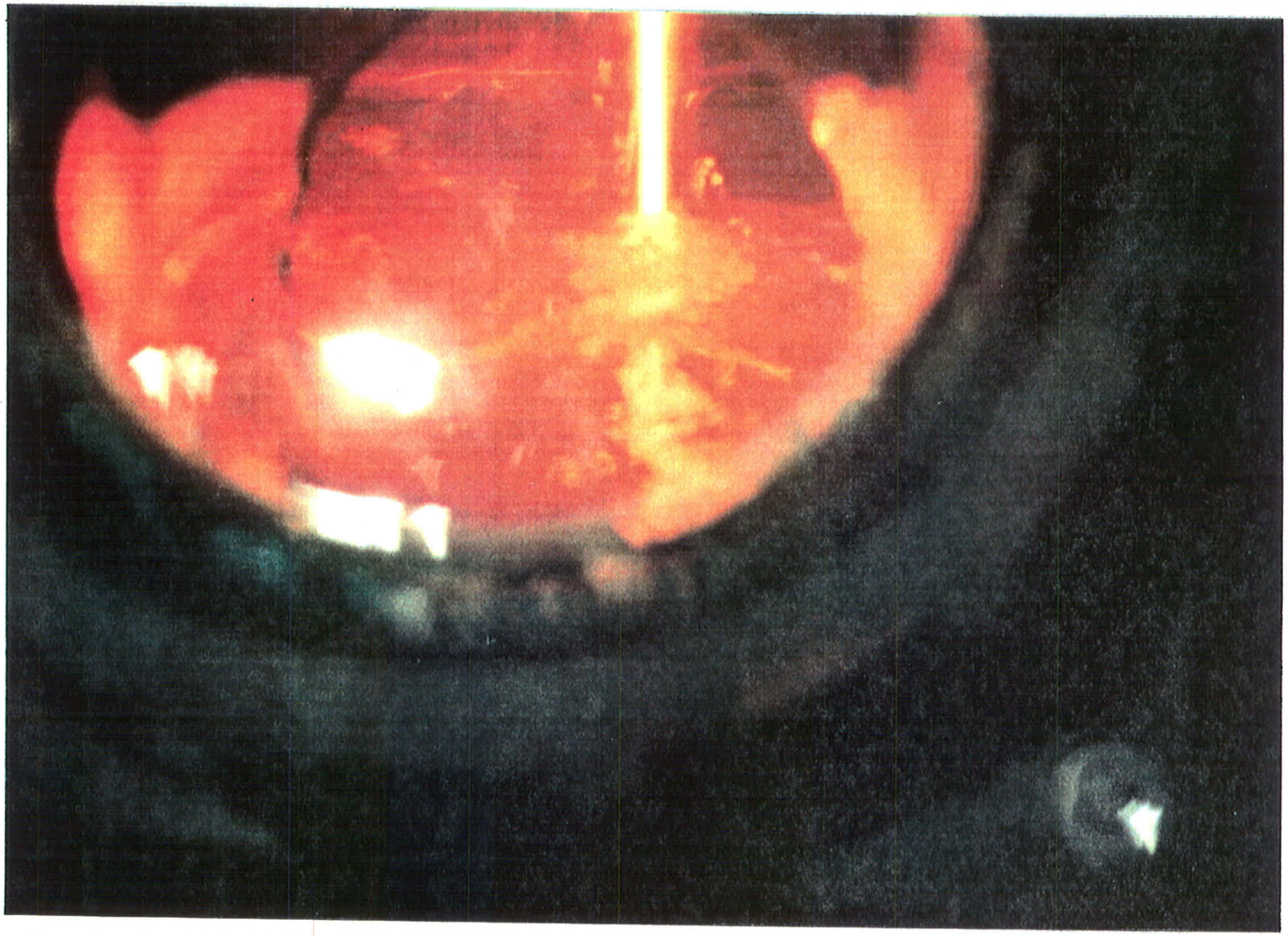


LNS Burner Slag Tap

TransAlta Resources Investment Corporation



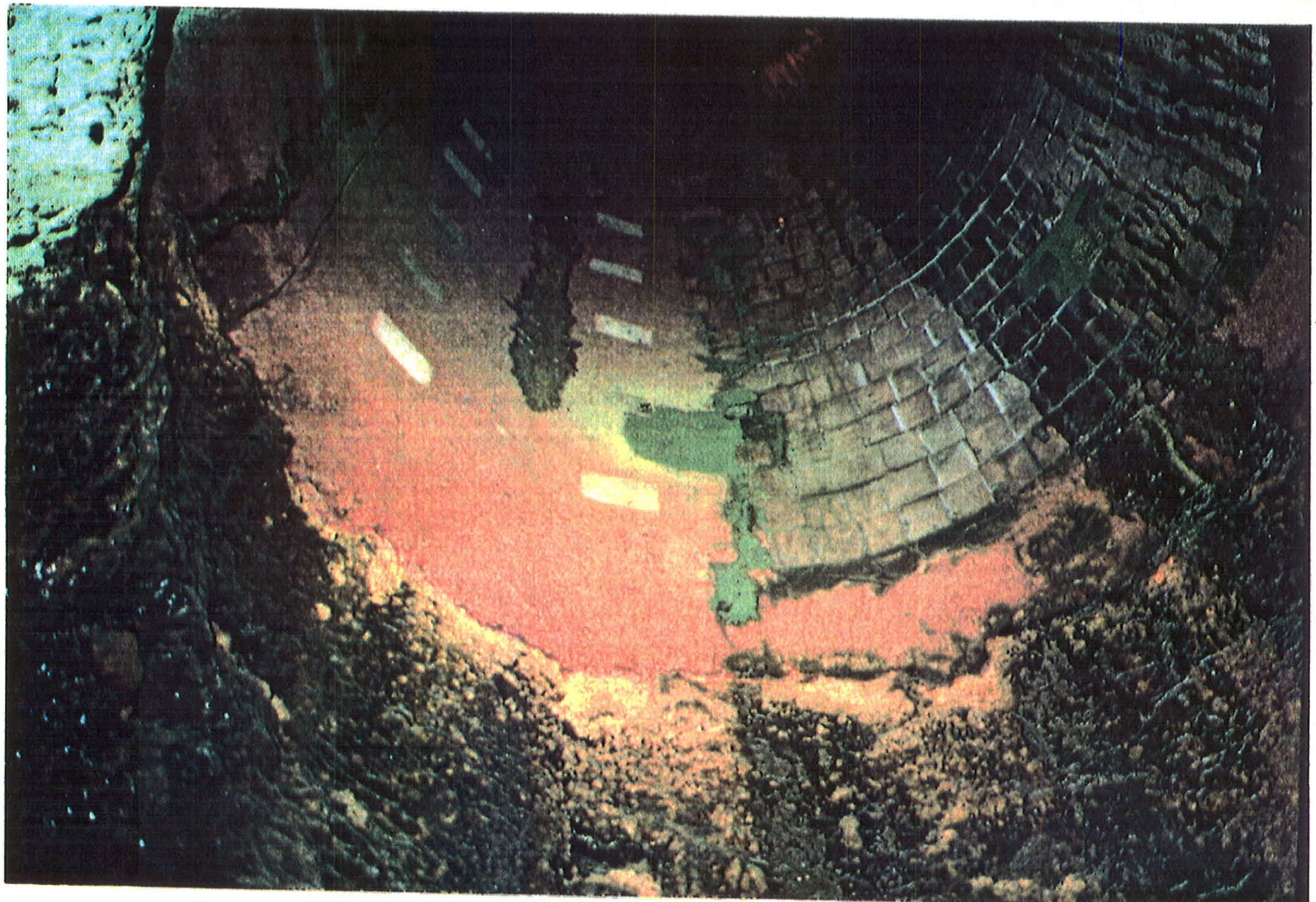
LNS-CAP



Slag Tap View Port

TransAlta Resources Investment Corporation

LNS-CAP



LNS Burner - Refractory Panels - Before Test

TransAlta Resources Investment Corporation

LNS-CAP

LNS Burner - Refractory Panels - After Test

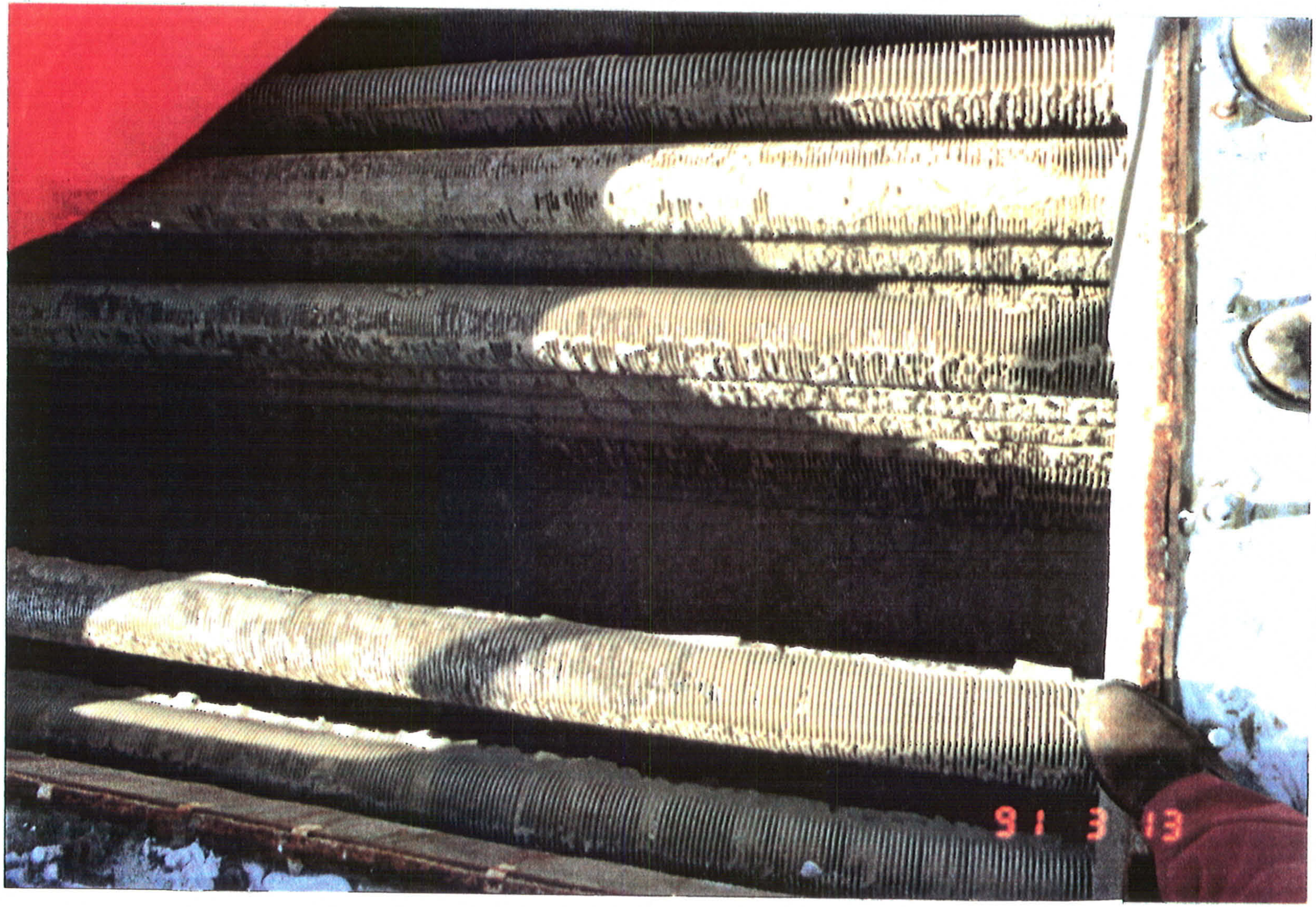
TransAlta Resources Investment Corporation

LNS-CAP

Test 12 - Boiler Deposits

TransAlta Resources Investment Corporation

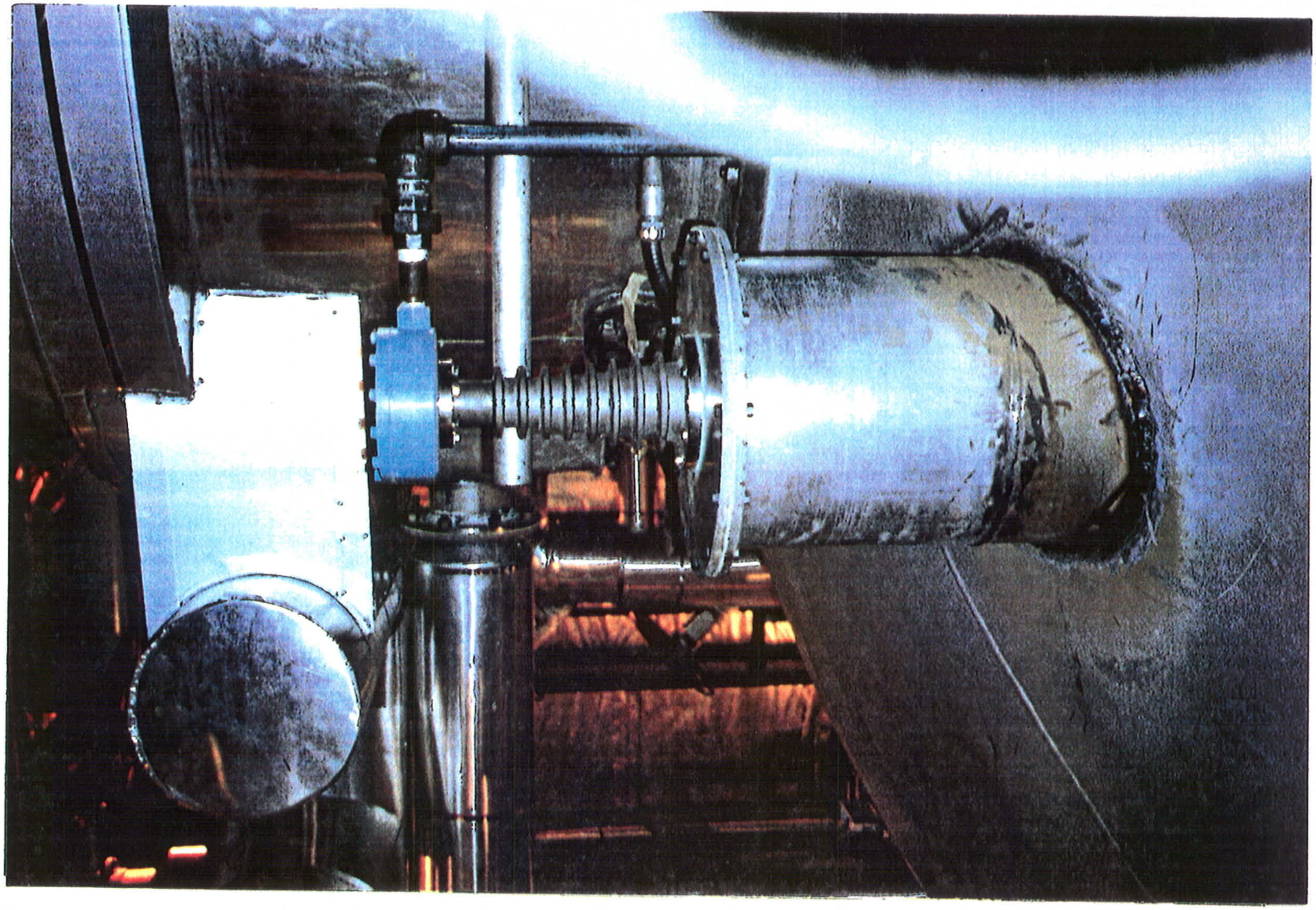
LNS-CAP



Test 12 - Economizer Deposits

TransAlta Resources Investment Corporation

LNS-CAP



Test 13 - Sonic Soot Blower

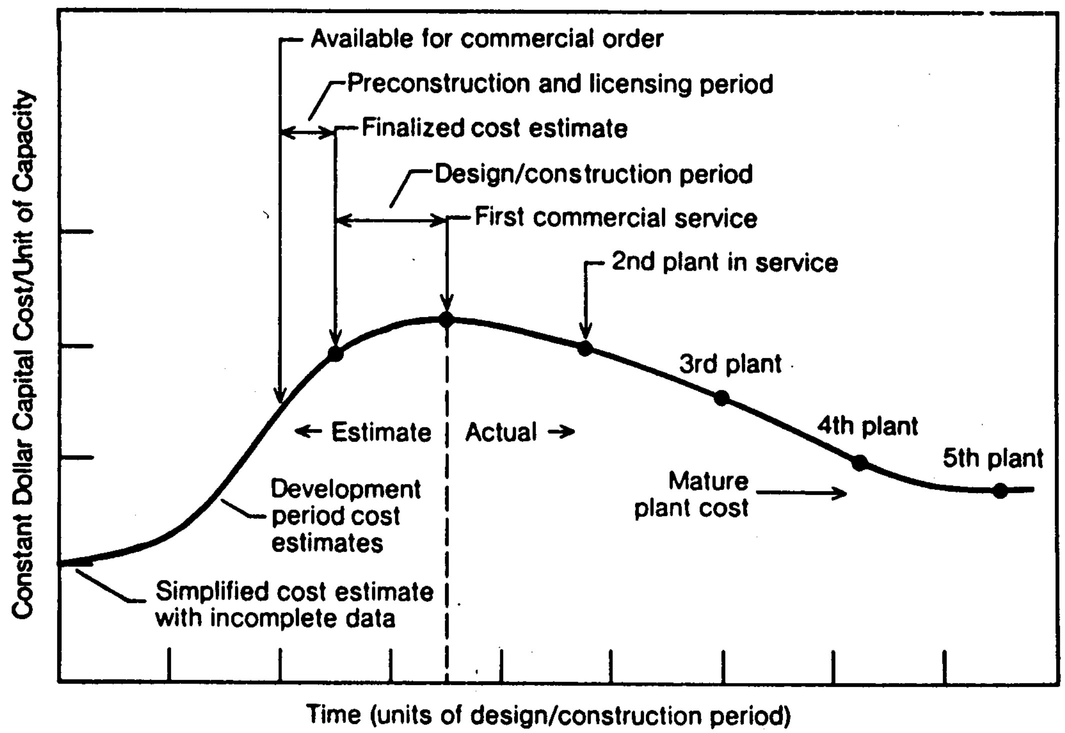
TransAlta Resources Investment Corporation

LNS-CAP

Control Room

TransAlta Resources Investment Corporation

Exhibit 1
EPRI CAPITAL COST LEARNING CURVE



TITLE: R&D Status of Carbon Dioxide Separation, Disposal and Utilization Technologies

AUTHOR: M.M. McDonald

AFFILIATION: TransAlta Utilities Corporation
110 - 12 Avenue S.W.
Calgary, Alberta
T2P 2M1

ABSTRACT: A study was carried out to provide a consortium of organizations with a basic understanding of the status of carbon dioxide separation, disposal and utilization technologies.

The key questions that were addressed were:

- what has been accomplished in technologies that relate to capture, disposal and use of carbon dioxide?
- how strong are the current efforts by others to solve the technology problems?
- which areas should be the subject of active research and development in western Canada, given the region's geographical location and natural resources?

COPROCESSING OF BITUMEN AND COAL WITH MOLTEN HALIDE CATALYSTS

By:

Dr. A. Chakma
Dept. of Chemical and Petroleum Engineering
The University of Calgary
Calgary, Alberta, Canada T2N 1N4

Presented at the Coal Research Contractor's Conference organized by the Alberta Office of Coal Research and Technology, Calgary, Alberta, October 30-31, 1991

ABSTRACT

Molten halide catalysts have been found to have excellent cracking activity towards coal and bitumen related structures as has been reported in the literature.

Coprocessing of coal with bitumen produces better solvolytic and hydrogenation effects. Therefore, coprocessing of coal and bitumen in the presence of a catalyst, which has excellent cracking activity towards both, should produce better results in terms of product yield. No work in this regard has been reported in the open literature.

Upgrading experiments have been carried out in a batch autoclave to study the effectiveness of molten halide catalysts consisting of $ZnCl_2$ and its mixtures with $MoCl_5$ and KCl, on simultaneous liquefaction of a sub-bituminous coal and upgrading of Athabasca bitumen. Two methods of catalyst addition to the reaction slurry were compared. Impregnation of the coal with a catalysts saturated methanol solution was found to be more effective than direct addition of the pulverized catalyst. The effect of different variables such as temperature, reaction time, catalyst quantity and, more importantly, catalyst formulation were studied. The optimum temperature and reaction time were found to be 400^{o}C and 2 hours, respectively. Under these conditions, a net coal conversion of 75% coal was achieved on daf (dry ash free) basis. A combination of $ZnCl_2$, $MoCl_5$ and KCl was found to be the best catalyst for the coprocessing of Athabasca bitumen and a subbituminous coal. For this catalyst combination the liquid yield was 75.9 wt% of the daf feed. Coke and gas yields were 16.75% and 7.35% respectively. For separate processing of bitumen and coal under similar operating conditions liquid, coke and gas yields would be 56.13%, 28.70% and 15.17%, respectively. Thus the expected synergistic effect on product yields during coprocessing compared to processing of coal and bitumen separately, has been confirmed.

INTRODUCTION

Coprocessing involves simultaneous upgrading of coal and bitumen/heavy oil in the presence of hydrogen into synthetic crudes. In coprocessing, coal is liquefied while bitumen/heavy oil are converted into lighter products. Coprocessing is more advantageous than separate liquefaction of coal and upgrading of bitumen/heavy oil for many reasons. First of all, it literally eliminates or reduces the need for solvent recycle for coal liquefaction. Secondly, coprocessing provides higher liquid yields than what would be expected from linear addition of liquid yields obtained from separate coal liquefaction and bitumen/heavy oil upgrading operations. This is due to the fact that the bitumen/heavy oil fractions can act as hydrogen donors for the coal to some limited extent and also the inorganic matters of the coal may provide some beneficial catalytic activity for the upgrading of bitumen/heavy oil. In addition, coal is also known to be capable of enhancing demetallation of the bitumen/heavy oil (Lett and Cugini, 1986).

Alberta has huge reserves of subbituminous coals, bitumen and heavy oil and hence coprocessing of these low value fossil fuels is very attractive for the above mentioned reasons. Boehm et al. (1989) reviewed the economics of coprocessing of vacuum residue of Cold Lake bitumen and subbituminous coal and found coprocessing to have significant economic advantage over conventional heavy oil upgrading. Lenz et al. (1988) also found that the liquefaction of lignite via coprocessing with residual oils would have economic advantage over liquefaction of the lignite alone.

In this project, we have examined coprocessing of Athabasca bitumen and a subbituminous coal using molten halide catalysts.

LITERATURE REVIEW

THERMAL COPROCESSING:

Extensive research work has been carried out on coprocessing since the late seventies. High coal-oil conversion rates and a synergistic effect on the conversion have been reported by several authors (Boehm et al., 1989; Lenz et al., 1988). The synergism is believed to be due to the solvolytic effects provided by the bitumen in which bitumen solubilizes the coal particles and therefore makes it easier to break its physical bonds. Bitumen may also act as a hydrogen donor solvent. The aliphatic hydrogen of the bitumen may help in the cleavage of coal structures. On its part, coal may enhance the upgrading of bitumen due to the presence of inorganics such as pyrite, which may provide beneficial catalytic effects for the upgrading of bitumen. An extensive demetallation, deoxygenation and desulphurization of the liquid products have also been reported (Cugini et al., 1989) Fouda et al. (1989) investigated the effect of coal concentration on the coprocessing performance and observed a marked improvement in the process operability with the addition of coal, as compared to the processing of Cold Lake vacuum bottoms only. They found the extent of oxygen, vanadium and nickel removal to increase while coke formation was found to decrease. It is believed that metals such as Vanadium are adsorbed onto the surface of the unconverted coal.

CATALYTIC COPROCESSING:

While thermal coprocessing provides all the previously mentioned advantages of coprocessing, the liquid yields are rather low. As such catalytic co-processing is the preferred choice. The use of a catalyst may enhance coal conversion and liquid yields. Several different catalysts have been investigated for coprocessing. Mosphedis et al. (1980) have investigated the use of $CoMo/Al_2O_3$ while Mosphedis and Ozum (1984) have examined Fe_2O_3 and K_2CO_3 , as coprocessing catalysts. They found that the addition of $CoMo/Al_2O_3$ increased the conversion of coal to toluene soluble products. However, the quality of the products was found to depend on the type of coal and solvent (bitumen, coker gas oil, heavy oil etc.) used. Curtis et al. (1987) as well as Curtis and Pallegrino (1989) have conducted extensive research on different catalysts types. A comparison between a commercially available $NiMo/Al_2O_3$, bulk FeS_2 and MoS_2, and oil-soluble metal salts of organic acids showed that the oil-soluble catalyst precursors gave higher coal conversion and oil production in the coprocessing reactions than did the other catalysts. It was also found that more accessible pulverized $NiMo/Al_2O_3$ promoted higher conversion and better upgrading than the less accessible extrudates. Cugini et al. (1988) have examined ammoniam molybdate, a water soluble salt, as a dispersed phase catalyst for use in single stage coprocessing.

Molten halide catalysts have long been used in the liquefaction of coal. Zielke aet al. (1966) found molten $ZnCl_2$ to be superior over conventional catalysts for hydrocracking of pyrene, coal and coal extracts. They also carried out continuous hydroliquefaction of subbituminous coal over molten $ZnCl_2$ and obtained octane rich gasoline-range products. The catalytic action is suggested to proceed via an ionic mechanism which cleaves bonds in polyaromatic compounds, but is unable to open mono-aromatic rings (Zielke et al., 1966 a).

Nomura and coworkers (1981, 1982a; 1982b; 1983; 1986) through a series of studies have shown good conversion rates as well as high liquid yields from different coals, when combinations of several different metal chlorides were used as catalysts. Combination of $ZnCl_2$ and one or several other metal chlorides ($MoCl_5$, KCl, CuCl, $CrCl_3$) were shown to be more active than $ZnCl_2$ alone, the $ZnCl_2$ - $MoCl_5$ mixture being the most effective. Also, the amount of consumed hydrogen, was lowest when this mixture was used (Nomura et al., 1982a). $SnCl_2$ containing salts were shown to give higher yields of both hexane-soluble and benzene-soluble fraction, and lower yields of gases, than $ZnCl_2$, and $ZnCl_2$-KCl-NaCl salts (Nomura et al., 1983).

$ZnCl_2$ has also been found to be an excellent catalyst for converting asphaltenes into maltenes . Chakma et al. (1989) treated Athabasca coker feed bitumen with $ZnCl_2$, CuCl, and mixtures of the two, under a continuous flow of hydrogen and found that $ZnCl_2$ was effective for the conversion of asphaltenes to maltenes and for lowering the coke formation. CuCl was not as effective, and decreased the activity of $ZnCl_2$ when added to it. Nomura et al. (1981) found that an addition of $MoCl_5$ to a mixture of $ZnCl_2$-KCl-NaCl gave higher conversion of asphaltenes to maltenes and a markedly reduced coke formation (3.7% compared to 15.6 %), suggesting very good hydrocracking abilities of the $MoCl_5$ on asphaltenes. Also the sulphur contents of the pentane-soluble and benzene-soluble fractions were lower when $MoCl_5$ containing catalysts were used.

Since the molten halide catalysts have been shown to be effective in hydrocracking of both coal and asphaltenes/bitumen, the use of these catalysts in coprocessing should also be effective.

COPROCESSING SCHEMES:

Coprocessing essentially involves two major steps, namely (i) solubilization or solvolysis and (ii) hydrogenation. Coprocessing reactions can be carried out either in a single stage process where solvolysis and hydrogenation steps are carried out in a single stage or in a two stage process, where these two steps are carried out separately. The CCLC process currently under development by Canadian Energy Development, Inc., as well as Alberta Research Council's coprocessing scheme are examples of two stage processes. In the CCLC process, the first stage solubilization is followed by second stage hydrogenation at temperatures ranging from 440 - 460 ^{o}C and H_2 pressures ranging from 14-18 MPa. Hydrogen consumption is reported to be about 2-3 wt%. In the ARC two stage coprocessing scheme, coal solubilization is enhanced by a mixture of CO and steam in the presence of an alkali metal catalyst during the first stage of operation. The second stage involves catalytic hydrogenation. Operating temperatures ranged from 416 to 430 ^{o}C while H_2 pressure ranged from 14.2 to 20.7 MPa. Hydrogen consumption was in the range of 1.6 to 2.85 wt%.

OPERATING CONDITIONS:

Most of the coprocessing schemes utilize high severity temperature and pressure conditions, in an effort to maximize liquid yields. Typical coprocessing conditions are; temperature: 420 to 460 ^{o}C; H_2 pressure: 15 - 25 MPa. High temperature operation means greater energy requirement while higher pressure means not only recompression cost for the hydrogen but also implies higher capital cost for the high pressure reactor vessel. Since there is considerable capital and energy savings potential for low severity processing, some efforts have been directed at achieving desirable liquid yields under low severity conditions. The use of CO and water as reducing agents for the liquefaction of coal under mild conditions (350 ^{o}C) had been investigated in 1921 by Fischer and Schrader. The other option is of course to use catalysts which would allow low severeity operation and yet provide acceptable yields. The use of a catalyst allows to achieve high coal conversion and high liquid yield, simultaneously. In addition, with the proper choice of catalyst, one can also upgrade the quality of the liquid product by increasing its H/C ratio. Several catalysts have been investigated in these respects. Moschopedis et al. (1984) have investigated the effects of $CoMo/Al_2O_3$ and Fe_2O_3 and K_2CO_3 catalyst combinations on coprocessing. They found that the addition of $CoMo/Al_2O_3$ increased the conversion of coal to toluene soluble products and found the quality of the products to be dependent on the type of coal and solvent (i.e. bitumen, heavy oil etc.) that were being used. Curtiss and coworkers have conducted extensive research on different catalysts types (1987 and 1989). A comparison between a commercially available $NiMo/Al_2O_3$ catalyst, bulk FeS_2 and MoS_2, and oil soluble metal salts of organic acids showed that the oil soluble catalysts precursors gave higher coal conversion and liquid yield than did the other catalysts. It was also found that more accessible pulverized $NiMo/Al_2O_3$ promoted higher conversion and more upgrading than the less accessible extrudates.

OBJECTIVES

The primary objectives of the present project are:

1. To develop a coprocessing process for the simultaneous liquefaction of coal and upgrading of Athabasca bitumen which yields high quality liquid products.
2. To develop a suitable molten halide catalysts for this purpose.

To achieve the above stated objectives an experimental program was initiated. The experimental program mainly consisted of reacting coal and bitumen mixtures in the presence of catalysts under H_2 pressure at desired temperatures for 1 to 2 hr periods and analyzing of the reaction products.

EXPERIMENTAL PROCEDURES

Reagents

Athabasca coke feed bitumen, produced by the hot water process, was supplied by the Alberta Research Council sample bank. It was used without any further preparations. Select properties are shown in Table 1.

Table 1: Select properties of Athabasca Coker Feed Bitumen

API gravity:	9.0
Density (kg/m^3)	1006
Viscosity 25^{o}C (mPa.s)	197,000
Ash (wt%)	0.70
Carbon (wt%)	83.16
Hydrogen (wt%)	10.27
Nitrogen (wt%)	0.47
Sulpher (wt%)	4.95
Oxygen (wt%)	0.76
Asphaltenes (wt%)	15.4
H/C-ratio	1.47

Subbituminous coal was supplied by Forestburg Colliers Ltd. It was pulverized and sieved to pass through <75 µm (200 Tyler Mesh) screen. Select properties of the coal used are shown in Table 2.

Table 2: Select Properties of Subbituminous coal

Ash (wt%)	19.9
Carbon (wt%)	67.59
Hydrogen (wt%)	4.28
Nitrogen (wt%)	1.48
Residue (wt%)	26.35
H/C-ratio	0.76

$ZnCl_2$, $MoCl_5$, KCl, CuCl, NaCl, $SnCl_2$ (certified ACS grade), were supplied by Fischer Scientific Co. Reagent grade n-pentane and toluene were provided by Anachemia Science. Omnisolv methanol with a minimum purity of 99.9% was supplied by BDH Inc. Hydrogen (g) and nitrogen (g) with purities exceeding 99.95% were supplied by Medigas Canada Inc.

Apparatus

The experiments were carried out in a stainless steel, bolted closure, stirred autoclave with an internal volume of 300 mL. Agitation was achieved by a magnetically actuated, packless and non-contaminating rotary impeller system. The impeller was designed to draw the process gases (hydrogen) through the tubular drive shaft and disperse it through the liquid phase at high velocity. The agitator was driven by a DC motor at 1800 rpm.

Due to the high corrosive impact of molten halide catalysts on stainless steel, certain changes had to be made inside the autoclave. The most severe type of corrosion found when using $ZnCl_2$ is stress corrosion. This appears as cracks in tube bends or in places where stresses, e.g. through torque, are applied. To eliminate cracks, the stainless steel tubing was replaced with one made of Inconel 600 tubing, which is more resistant to chloride corrosion.

The autoclave was heated by an electric, external jacket type heater, which was controlled with a PID temperature controller.

Impregnation of Coal with $ZnCl_2$

In the first few experiments, the catalyst was added in bulk directly into the autoclave to the coal and bitumen. This caused very little catalytic effect when small concentrations were used (10 wt.% of the dry, ash free coal), and resulted in severe corrosion when larger amounts were added (100 wt.% of the daf coal).

To increase catalytic activity and decrease the impact of corrosion, we decided to try to trap the catalyst in the pores of on the surface of the coal particles. This was done by impregnating the coal with a $ZnCl_2$ saturated methanol solution . $ZnCl_2$ and coal were dried for 2 hr. at 105^{o}C. 100 g of dry $ZnCl_2$ was then dissolved in 200 mL of methanol. To this solution was added 124 g of dry coal, corresponding to 100 g of dry, ash free (daf) coal. The mixture was continuously stirred with a magnetic stirrer. After soaking the coal overnight, the mixture was filtered through a coarse fritted disc funnel and a Whatman # 2 filter paper. The funnel with the methanol/$ZnCl_2$ soaked coal was then dried for up to 48 hr. or to constant weight, in a vacuum oven at 75^{o}C. The result was a coal powder impregnated with approximately 40 wt.% $ZnCl_2$, based on the daf coal.

Reactor Operation

The autoclave was cleaned before each run with acetone followed by dichloromethane and tetrahydrofuran to remove any $ZnCl_2$ and organics that might have been left from the previous experiment.

40-45 g of the $ZnCl_2$ impregnated coal was dried for 2 hr. at 105^{o}C. After cooling to room temperature in an evacuated desiccator, the total weight of the dry coal, catalyst and the container (beaker) was recorded. The coal was then transferred into the autoclave, and the total amount of coal transferred was found from material balance. Bitumen was then added to the autoclave. The ratio of bitumen to coal was maintained at 70:30.

The autoclave was then sealed and pressure tested with nitrogen. After it had been determined that no leaks were present, the pressure was released very slowly to prevent bitumen and/or coal from being trapped and blown out of the reactor. To remove as much atmospheric oxygen and moisture as possible, the autoclave was purged once more.

The autoclave was then pressurized with hydrogen to an initial pressure of 6.9 MPa at ambient temperature, and heating of the reactor was started. After heating to the desired temperature, the temperature was held constant for either 1 or 2 hr. At the end of the experiment, cold water was circulated through an internal cooling coil. The temperature could be brought down to 100 ^{o}C in about 30 min., and to room temperature in about an hour. The heating and cooling rates as well as the temperature fluctuations were monitored and recorded.

After cooling the autoclave to room temperature, the pressure was carefully released. When the autoclave had reached atmospheric pressure, all connections were removed, and the reactor was cleaned for product recovery.

Product Workup

The reaction products were separated into different fractions according to Figure 1. The total product was removed from the autoclave with toluene. The walls of the reactor as well as all tubing and other parts were carefully scraped to remove as much coke and catalyst as possible. Larger aggregates or lumps of coke and $ZnCl_2$ were crushed with a mortar and pestle to prevent toluene soluble material from being trapped within such lumps.

To ensure that all toluene soluble material was dissolved, the products together with 0.8-1.0 L toluene were placed in the ultrasonic bath for 5 min. at room temperature. The mixture was then filtered through a weighed Whatman # 2 filter and a weighed coarse (ASTM 40-60) fritted disc funnel. This separated the toluene insolubles (TI). The filter cake was washed repeatedly with a total of approximately 0.8 L toluene, so that the total volume of toluene used was approximately 1.6 L.

The fritted disc funnel, with the filter cake, was dried to constant weight in a vacuum oven at 75^{o}C. The weight of the separated solids was the yield of the TI fraction, which contained coke, unreacted coal, catalyst and ash. All of the $ZnCl_2$ and ash included in the feed was considered to be recovered in the TI fraction.

The filtrate was transferred to a tarred round bottom flask. The toluene was evaporated in the Rotavapor at 65^{o}C and 90 rpm. The pressure of the rotavapor was gradually reduced to 50 mbar, and the end point of the evaporation was chosen to be when no more condensation could be observed at this pressure. The weight of the remaining liquid product was the yield of the toluene soluble fraction (TS).

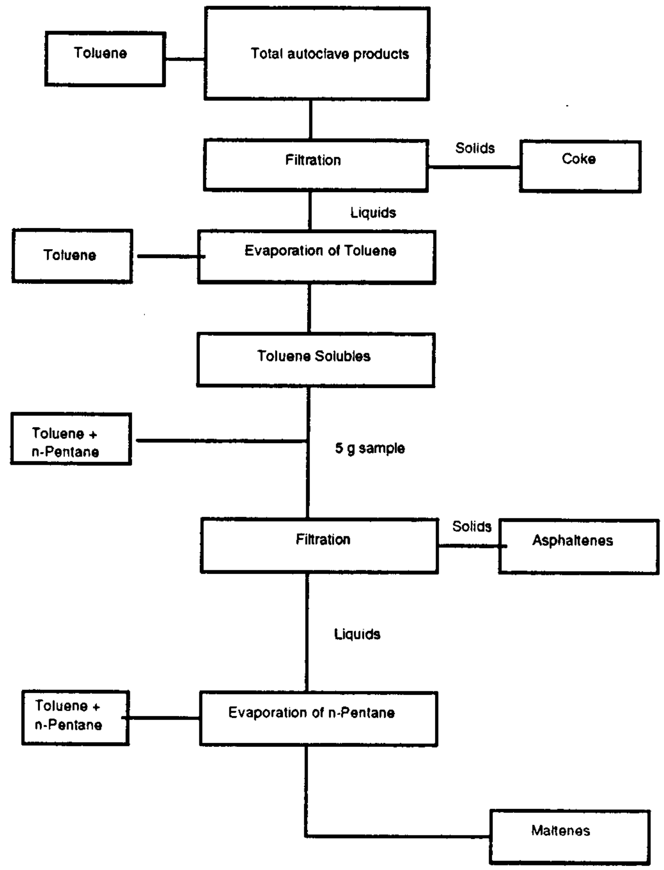


Figure 1: Basic product workup procedure.

The toluene solubles (TS), were further fractionated into asphaltenes and maltenes (distillables, pentane soluble oils), according to a procedure proposed by Moschopedis and Moschopedis and Hawkins (1981), with some minor modifications. 5 g of the TS was dissolved in 5 g toluene, in an Erlenmeyer flask in an ultrasonic bath, the asphaltenes were filtered off through a Whatman # 2 filter paper in a tarred medium porosity (ASTM 10-15) fritted disc funnel. The filter cake was crushed and rinsed with an additional 200 mL pentane. Any asphaltenes left on the walls of the Erlenmeyer flask were dissolved in a small amount of toluene and transferred to the fritted disc funnel. The funnel was dried in the vacuum oven at 75^{o}C until constant weight, and then cooled to room temperature in a desiccator. The weight of the filter cake was the weight of the pentane insoluble fraction (PI), or asphaltenes.

The pentane was evaporated in the Rotavapor at 65^{o}C and 830 mbar. The pressure was then lowered to 40 mbar to evaporate the remaining toluene. The evaporation was stopped when the weight of the remaining pentane solubles (PS), also called maltenes, and the weight of the PI, or the asphaltenes, approximately equalled the weight of the original sample taken from the TS fraction.

Analyses of Products

The viscosity of the total toluene soluble fraction was measured, when sufficient volume of this fraction was recovered. This was done using a Contraves Rheomat 115 rotational viscometer with a MS-DIN 125 measuring system.

The coke and the maltene fractions were analysed for C,H and N content using a Control Equipment Corporation Model 240-MA elemental analyser.

RESULTS AND DISCUSSION

Effects of Different Means of Charging the Catalyst

We have looked at two different ways of charging the catalyst. First the catalyst was added in powder form, straight into the autoclave together with the feed coal and bitumen. When low concentrations of $ZnCl_2$ (less than 10 %) were added in this way, negligible changes in the products yields were noticed when compared with the non catalytic experiment as can be seen in Figure 2.

A higher load of catalyst (30%) added in the same way resulted in relatively high yields of gases and coke, but lower yield of maltene products. It also resulted in severe corrosion of the autoclave. In addition, the H/C ratios of the maltene fraction was also low (1.3), suggesting a low degree of hydrogenation of the liquid products.

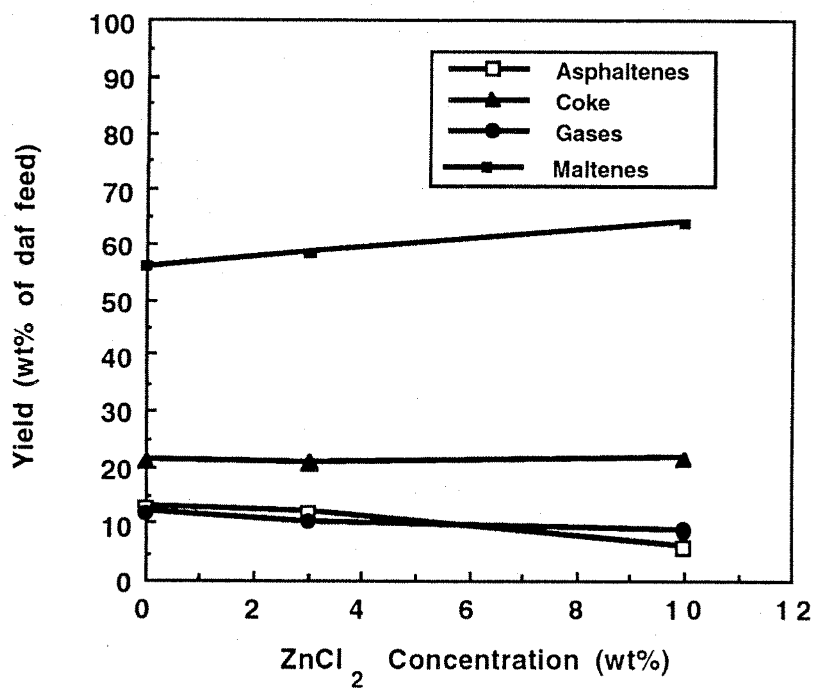


Figure 2: Effect of $ZnCl_2$ concentration on product yields (400 C, 60 min.)

The second method used was impregnation of the catalyst on the surface and on the pores of the coal as outlined in the sample preparation section. With an impregnation of $ZnCl_2$ amounting to 36-40 wt.% of the daf coal, the total charge of catalyst, when the feed contained 30 wt.% daf coal, was approximately 10% of the total daf feed. Even with this smaller load of $ZnCl_2$, the results were better than those from the run with 30 wt.% $ZnCl_2$ introduced directly into the autoclave under similar operating conditions. Figure 3 shows a comparison of results obtained at 420°C and 2 hr. of reaction time.

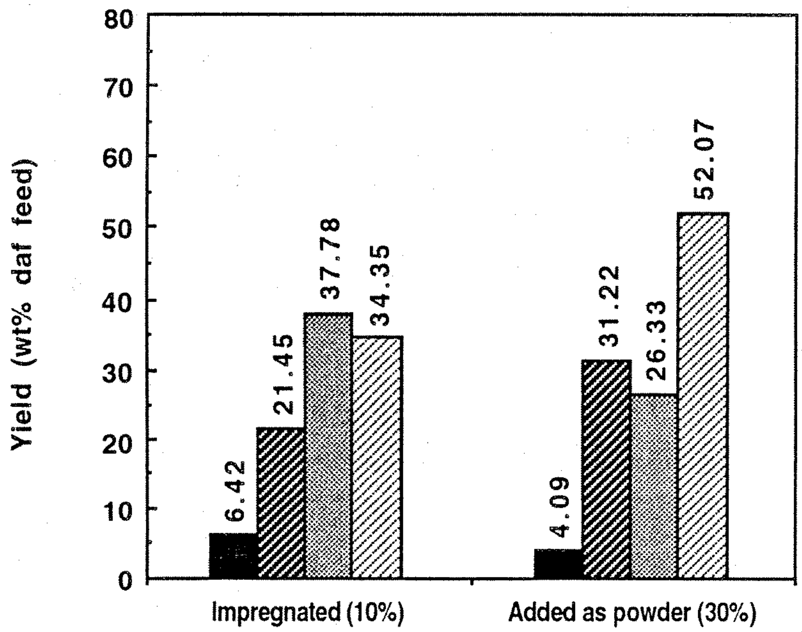


Figure 3: Effect of catalysts addition on product yield (420^{o}C, 2hr).

The maltene yield decreased a little, from 38.36% to 34.35% of the total daf feed. While a significant decrease in the TI fraction was noticed, from 31.22% to 21.45%, the asphaltene yield actually increased somewhat, from 4.09 to 6.42%. The gas yield also increased from 26.33% to 37.78% of the total daf feed when the impregnated coal was used. It is evident that the catalyst impregnated coal provided higher catalytic activity than directly added catalyst. Corrosion problem was also minimal when impregnated catalyst was used. Subsequent experiments were carried out with impregnated catalysts only.

COPROCESSING WITH MIXED CATALYSTS

Having proven the effectiveness of $ZnCl_2$ catalysts in the coprocessing of bitumen and coal, attention was focussed on improving the quality of the liquid product with mixed catalysts. Since $ZnCl_2$ is a strong cracking catalysts, it was clear from the beginning that a hydrogenation catalyst need to be added to improve hydrogenation reactions and thereby increase the H/C ratio of the product. Our obvious choice was $MoCl_5$ for the hydrogenation purpose. Figure 4 shows a comparison of product yields obtained with $ZnCl_2$ and $MoCl_5$ catalysts.

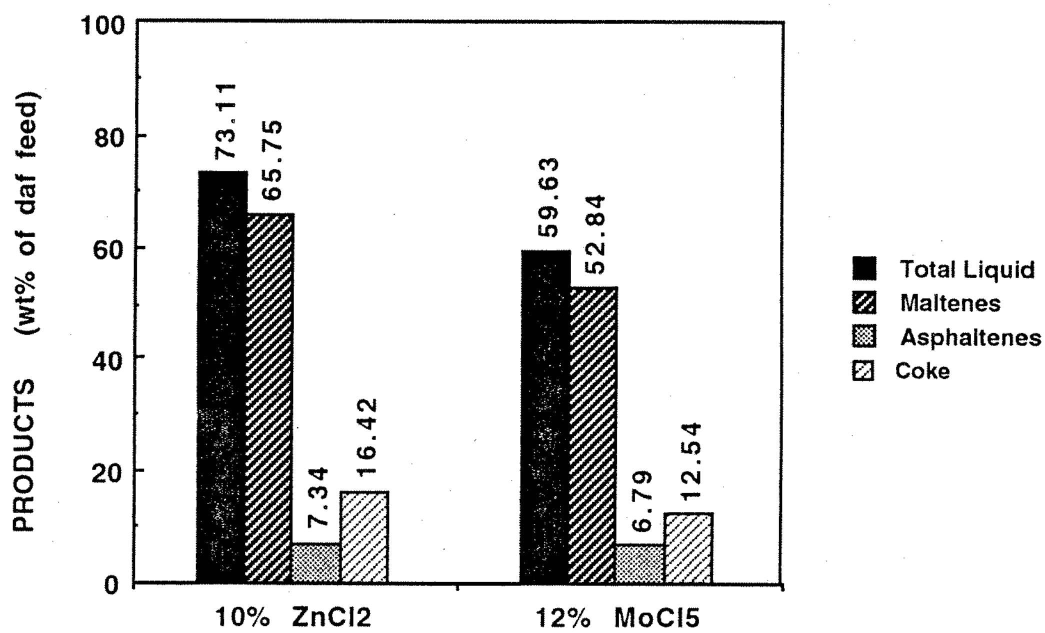


Figure 4: Comparison of product yields with $ZnCl_2$ and $MoCl_5$ catalysts.

The following observations can be made from Figure 4. First of all, the total liquid yield is lower with $MoCl_5$ (59.63%) than with $ZnCl_2$ (73.11%). In addition the maltene content of the liquid product is also lower with $MoCl_5$. This suggests that $MoCl_5$ alone is not a very ideal catalyst for co-processing. However, the asphaltene and coke fractions are lower with $MoCl_5$ than with $ZnCl_2$. This is an indirect indication of the hydrogenation capability of the free radicals by $MoCl_5$ which would otherwise recombine and form asphaltenes and coke. Based on these observations, it becomes clear that $MoCl_5$ is a very strong catalyst and also that these catalysts need not be used in high quantities. Hence, a mixed catalyst consisting of $ZnCl_2$ (5.25 wt% of daf feed) and $MoCl_5$ (1.75 wt% of daf feed) was prepared and used in a coprocessing experiments under operating conditions similar to separate $ZnCl_2$ and $MoCl_5$ runs. The product yields obtained are compared with the separate runs using higher $ZnCl_2$ and $MoCl_5$ concentrations in Figure 5.

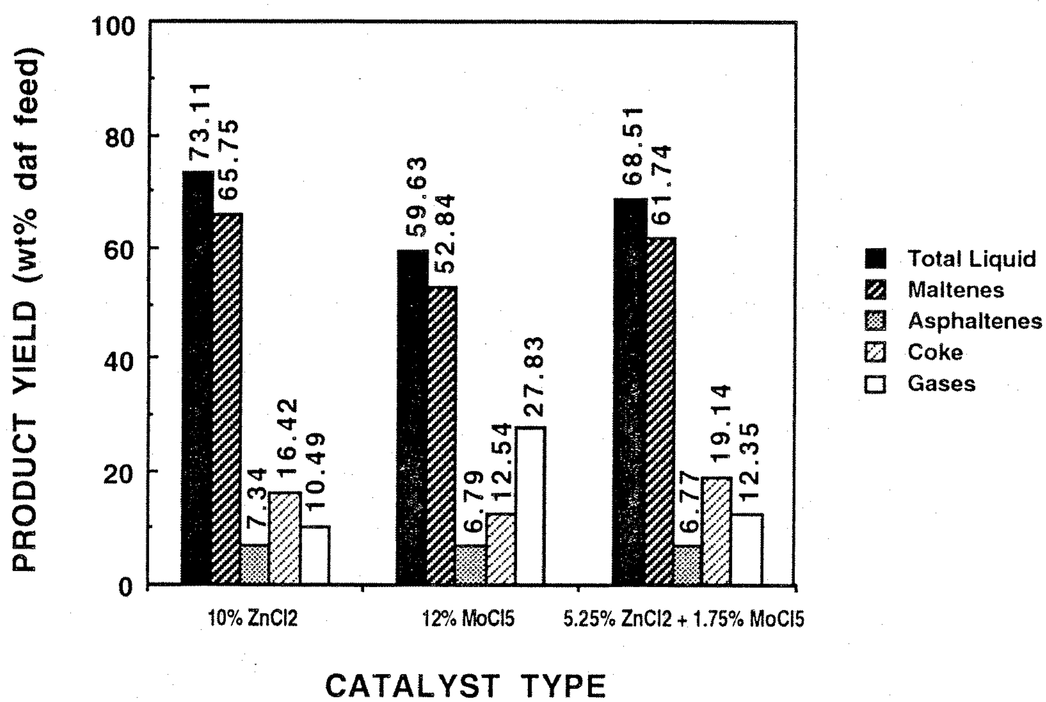


Figure 5: Comparison of product yields using $ZnCl_2$ and $MoCl_5$ catalysts.

A comparison of the H/C ratio of the products obtained with $MoCl_5$ and $ZnCl_2$ provides a better evidence of the hydrogenation activity of the $MoCl_5$ catalyst as shown in Figure 6. As can be seen, the H/C ratio of the liquid product is higher for $MoCl_5$ (1.55) compared with that of $ZnCl_2$ (1.46). The H/C ratio of the coke and the asphaltene fractions are somewhat higher for the $MoCl_5$ case, since $MoCl_5$ had hydrogenated the lighter free radicals to form other lighter products (maltenes or gases) thus contributing to the hydrogen deficiency of the coke and the asphaltene fractions.

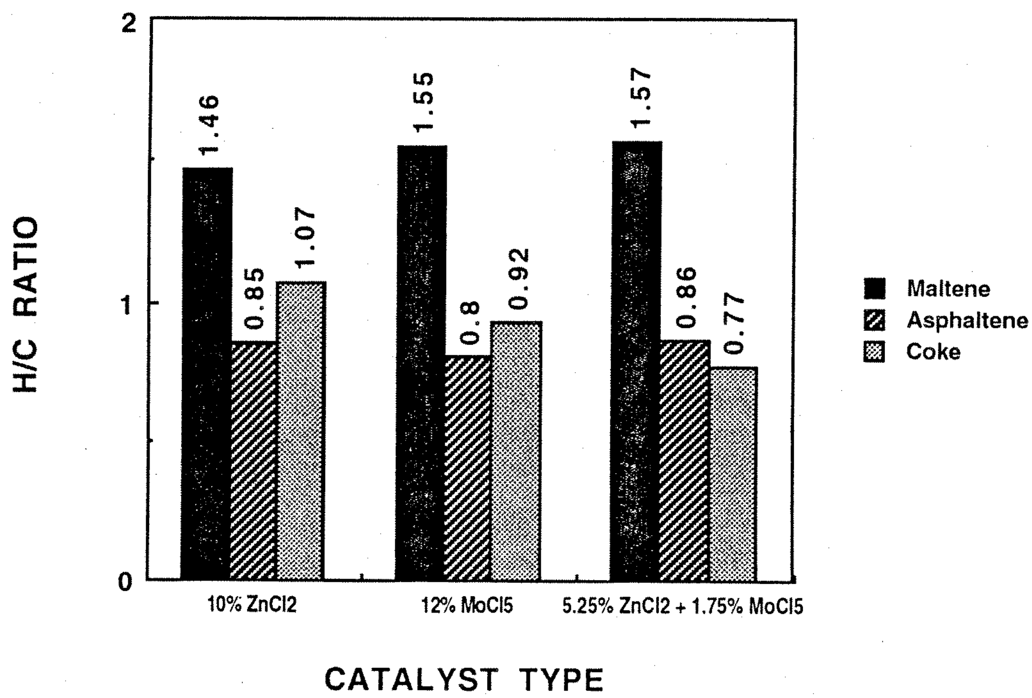


Figure 6: Comparison of H/C ratios of various fractions with $MoCl_5$ and $ZnCl_2$.

COMPARISON OF KCl AND CuCl ADDITIVES TO $ZnCl_2$ - $MoCl_5$ MIXTURE

From previous work as well as literature survey, it was decided to investigate the effects of two additives, namely KCl and CuCl, on the performance of the $ZnCl_2$ and $MoCl_5$ catalyst mixture. Figure 7 shows the product yields obtained with these two additives. KCl was found to provide higher liquid yields compared with CuCl. While CuCl was found to have more reactivity towards asphaltene cracking, thereby lowering its concentration in the product stream, it was equally reactive towards other fractions and this resulted in the production of gaseous products at the expense of other fractions. More importantly, higher catalyst activity lead to the formation of more coke. Figure 8 shows liquid product viscosity and asphaltene content of the liquid product as a function of additive type. CuCl provides a liquid product of lower viscosity due to its ability to reduce the asphaltene concentration by converting it to mostly gaseous products.

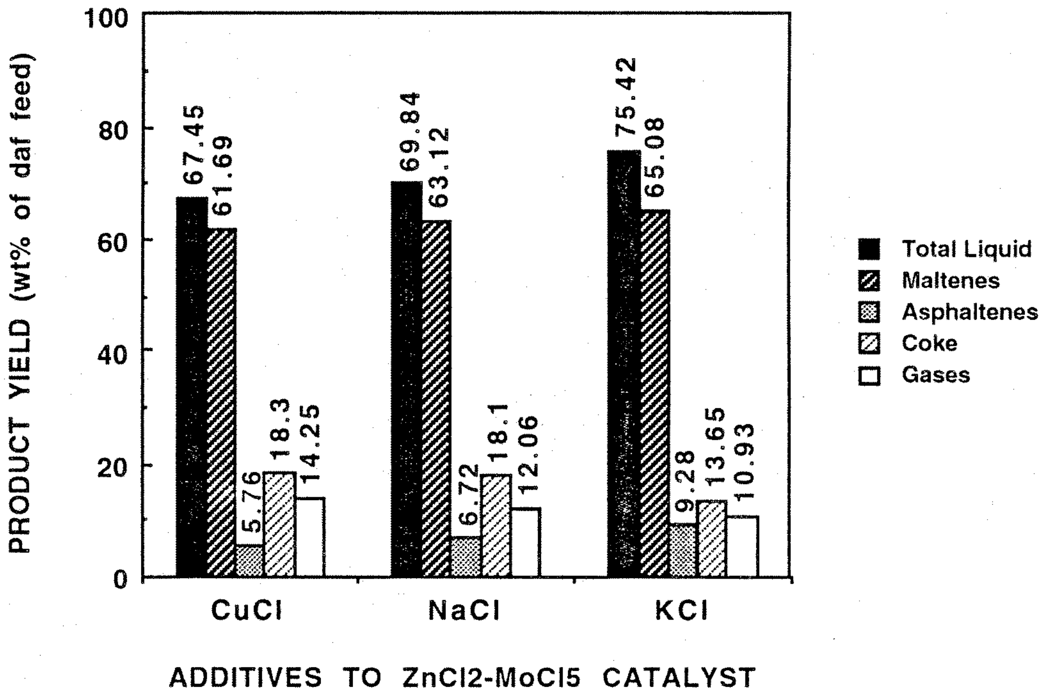


Figure 7: Product yield as a function of catalysts additive.

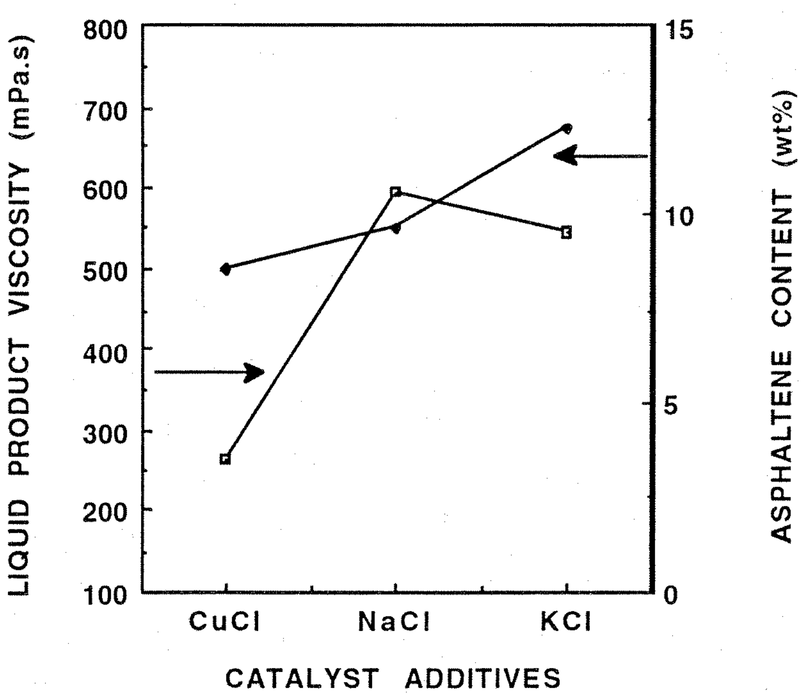


Figure 8: Liquid viscosity and asphaltene content as a function of t additive type.

Figure 9 shows the H/C ratio of different fractions. KCl additive gave better product quality as can be seen from the higher H/C ratio of the maltene fraction. As such CuCl was not considered to be a very good additive at the concentration levels examined.

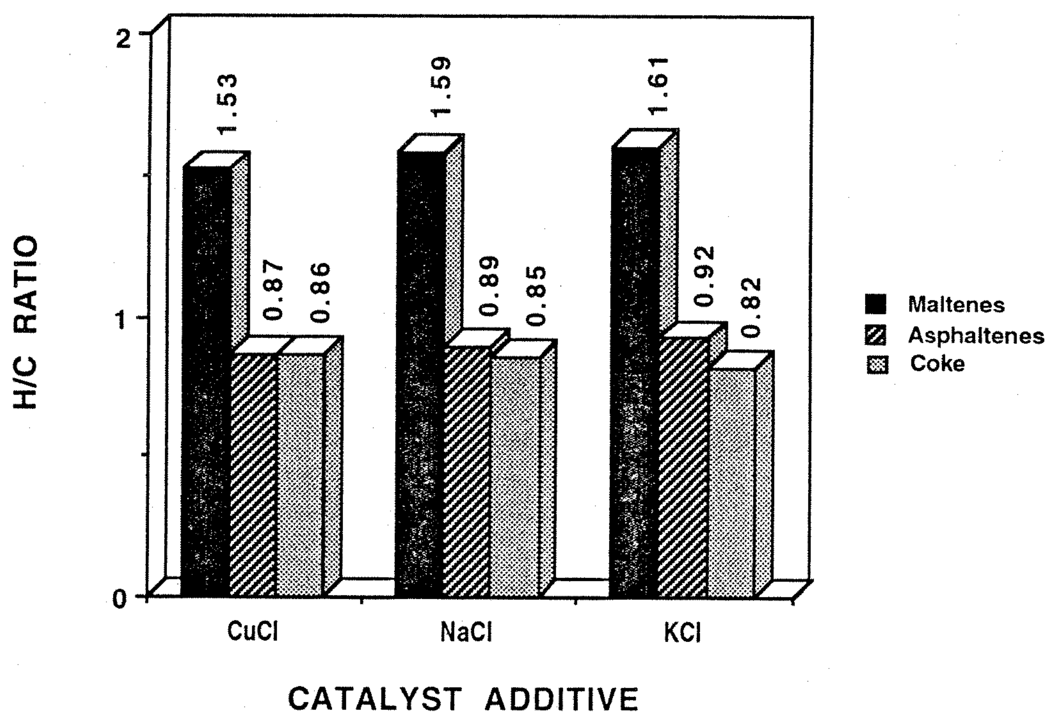


Figure 9: H/C Ratio of different product fractions as a function of catalysts additive type.

EXPERIMENTS WITH $ZnCl_2$-$MoCl_5$-KCl CATALYSTS

EFFECT OF REACTION TEMPERATURE

One of the other main objective of this project was to be able to conduct the coprocessing reactions under low severity conditions. Hence our target reaction temperature has been around 400^{o}C. Our experience with $ZnCl_2$ catalyst also convinced us that 400^{o}C reaction temperature is indeed feasible. However, due to the introduction of the mixed catalyst, we wanted to be sure that 400^{o}C was still the right temperature to work with. Hence two experiments were conducted at 400°C and 420°C in order to determine the effects of reaction temperature in this temperature range. Figure 10 shows the product yields obtained from these runs.The results are more or less the same for all practical purposes. Higher temperature results in the formation of more coke and gases and in the reduction of asphaltenes and maltenes. Figure 11 shows the H/C ratios of the different fractions. Higher operating temperature lowers the H/C ratio of all the non gaseous fractions. However, the effects in the temperature range of our interest are minimal. Therefore, it was decided to continue the remaining experiments at a temperature of 400°C.

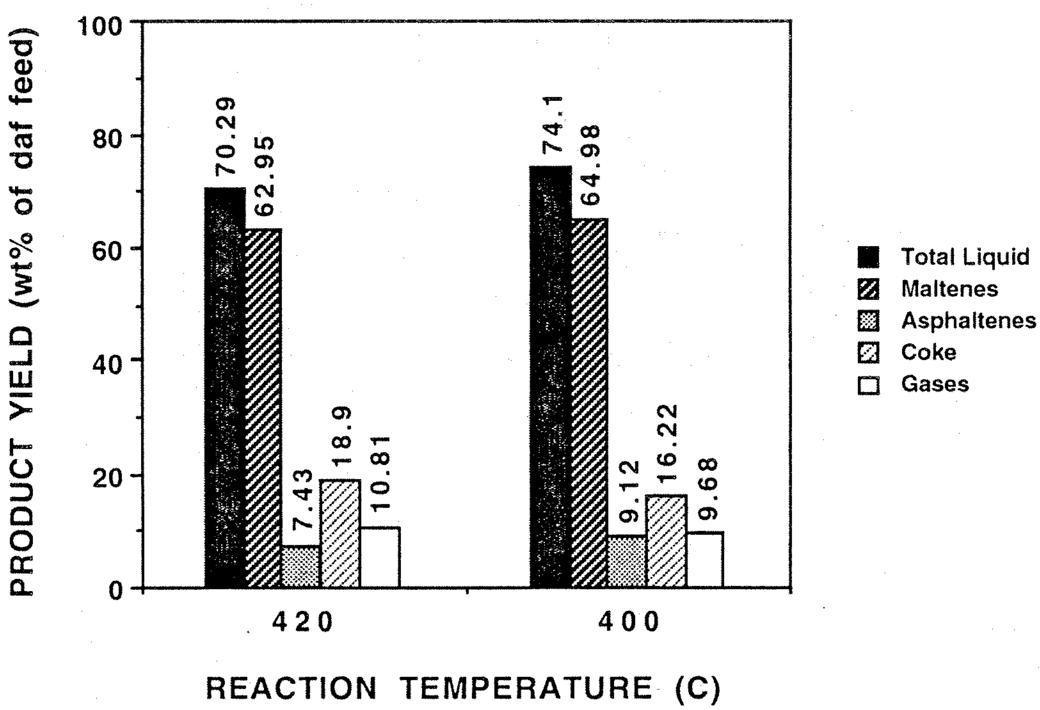


Figure 10: Effect of reaction temperature on product yield.

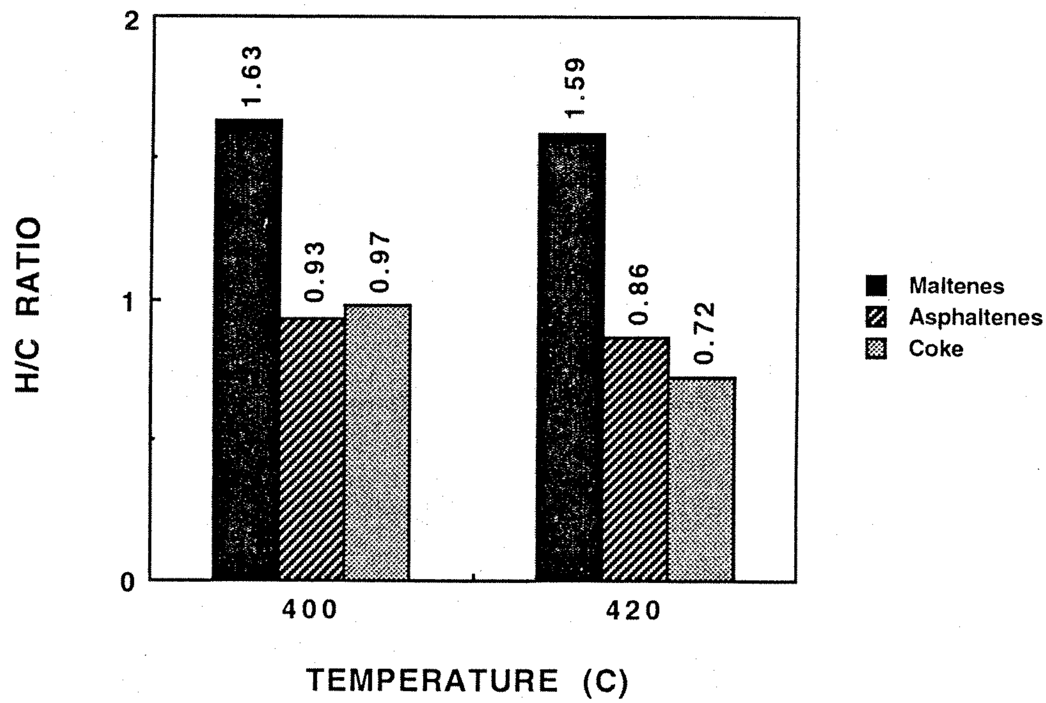


Figure 11: Effect of reaction temperature on the H/C ratio.

EFFECT OF REACTION TIME

A number of experiments were carried out to determine the effect of reaction time on coprocessing with the optimum mixed catalyst composition. The reaction temperatures for these runs were fixed at 400^{o}C. The catalyst was 7 wt% $ZnCl_2$-$MoCl_5$-KCl (3:1:1). The reactor was pressurized with H_2 to a pressure of 6.89 MPa.

Figure 12 shows product yields from as a function of reaction time. The effect of time on the liquid yield under these conditions is minimal. Only pronounced effect is on coke formation. It decreases with higher reaction time. This is perhaps better interpreted as, the conversion of coke formed during the early phase of the reaction to other products, such as gas.

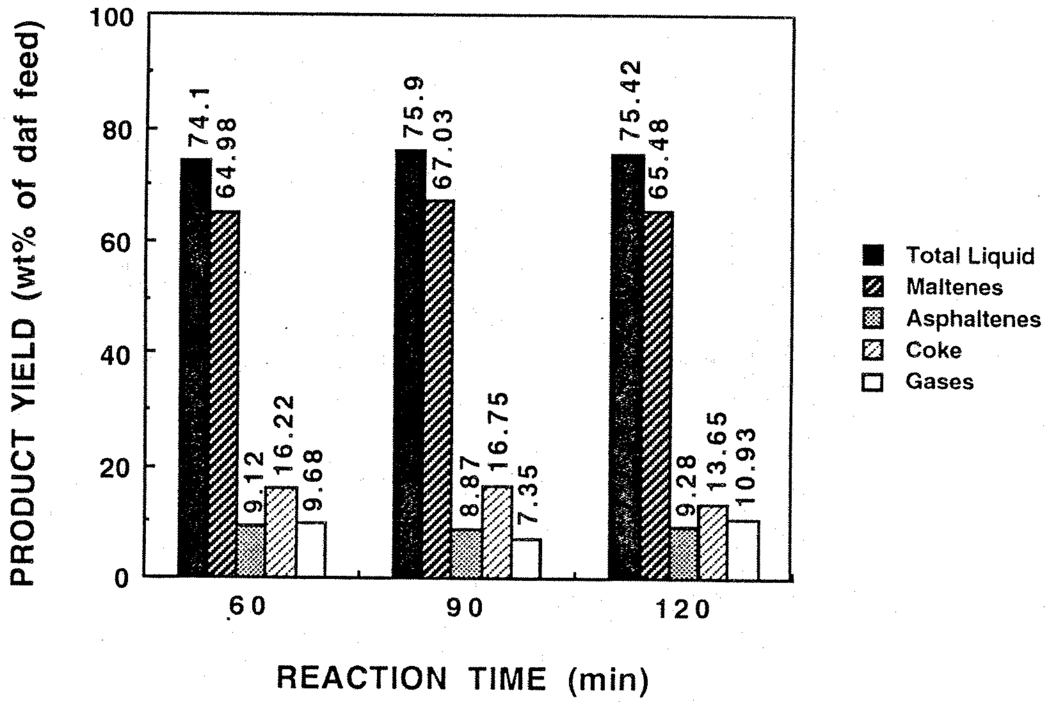


Figure 12: Effect of reaction time on product yields

Since more and more gas is formed as the reaction time increases, the H/C ratio of the other product fractions decrease accordingly. This can be seen from Figure 13, very clearly.

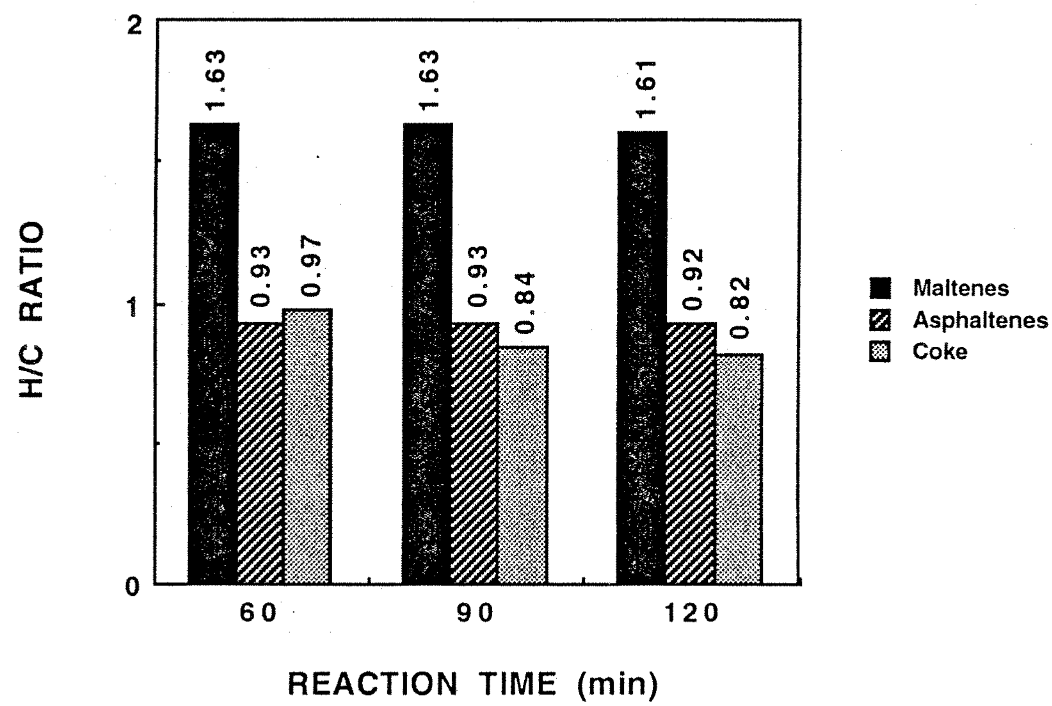


Figure 13: Effect of reaction time on the H/C ratio of different product fractions.

EFFECT OF CATALYST QUANTITY

In order to determine the optimum catalysts quantity required, a series of coprocessing experiments were carried out at 400 ^{o}C of reaction temperature and 120 minutes of reaction time. Figure 14, shows the product yields obtained with varying amounts of catalysts. An optimum was found which gave the highest liquid and maltene yields. When this optimum amount is exceeded, then the higher reactivity results in further conversion of the liquid into gaseous products.

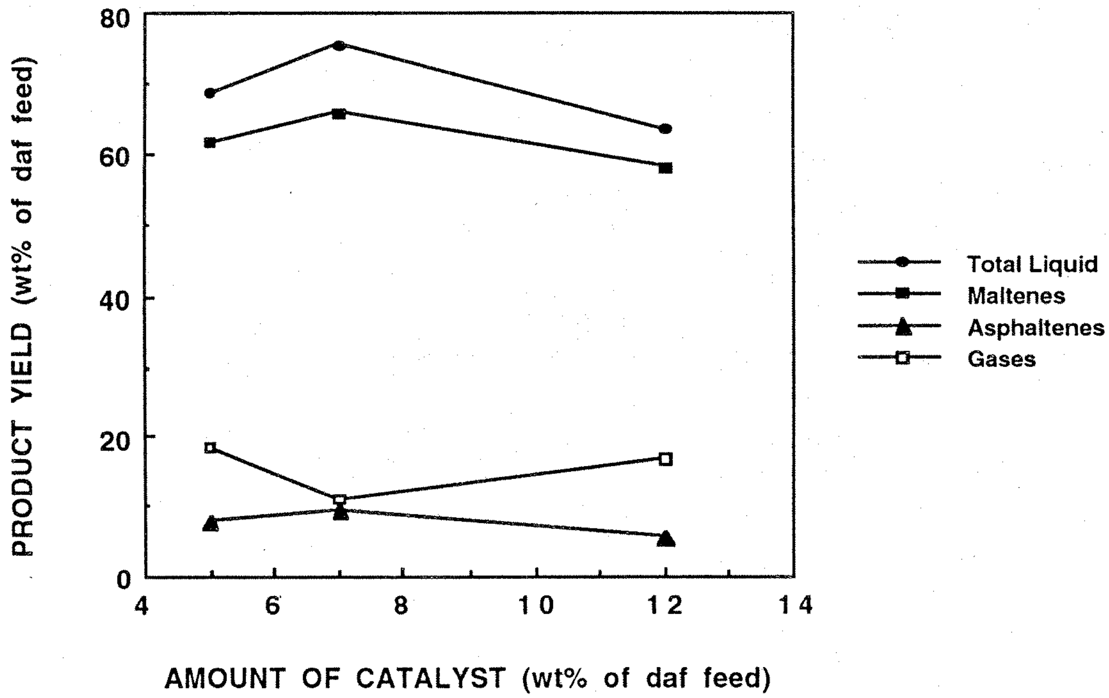


Figure 14: Effect of catalysts quantity on product yield.

Figure 15 compares the H/C ratio of different products as a function of the amount of catalysts added. This figure shows that the H/C ratios of the maltene and the asphaltene fractions decrease with the amount of catalyst added, while that of coke increases. The decline in H/C ratio of the maltenes and the asphaltenes can be attributed to the formation of gases from these two fractions and consequent loss of hydrogen rich components. Figure 16 presents liquid viscosity and asphaltene content of the liquid product as a function of catalysts quantity. In general viscosity of the liquid product tends decrease with catalysts concentration due to increased reactivity. Asphaltene content of the liquid product seems to affect its viscosity, higher asphaltene content leading to higher product viscosity.

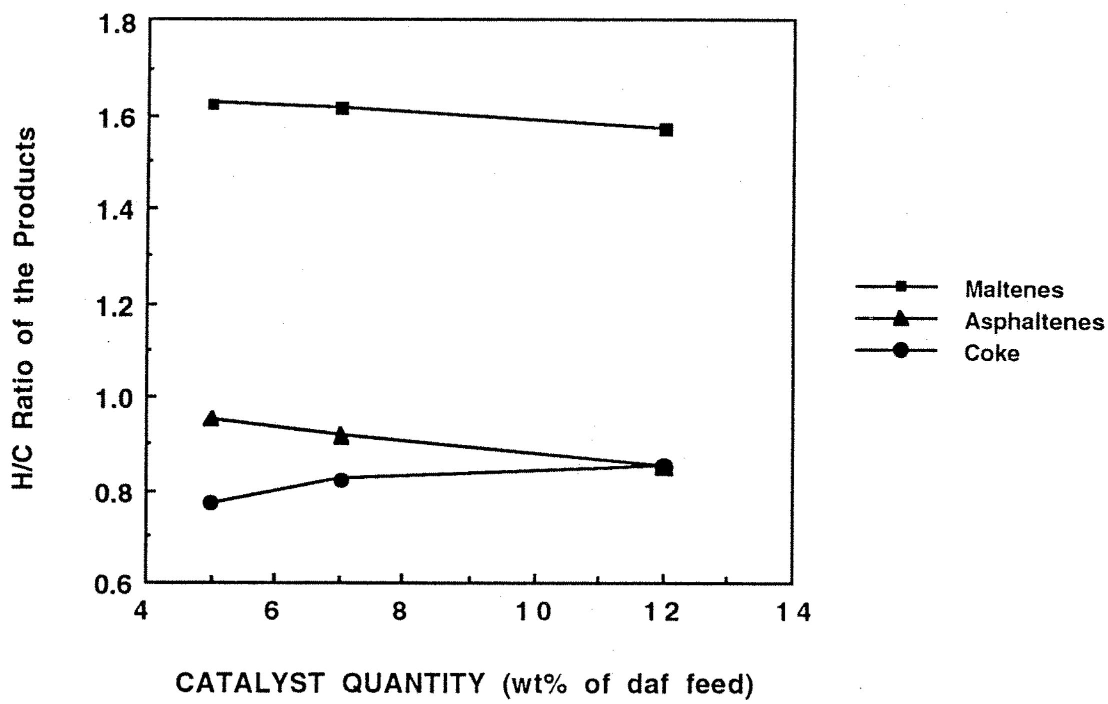


Figure 15: Effect of catalyst quantity on the H/C ratio of the products.

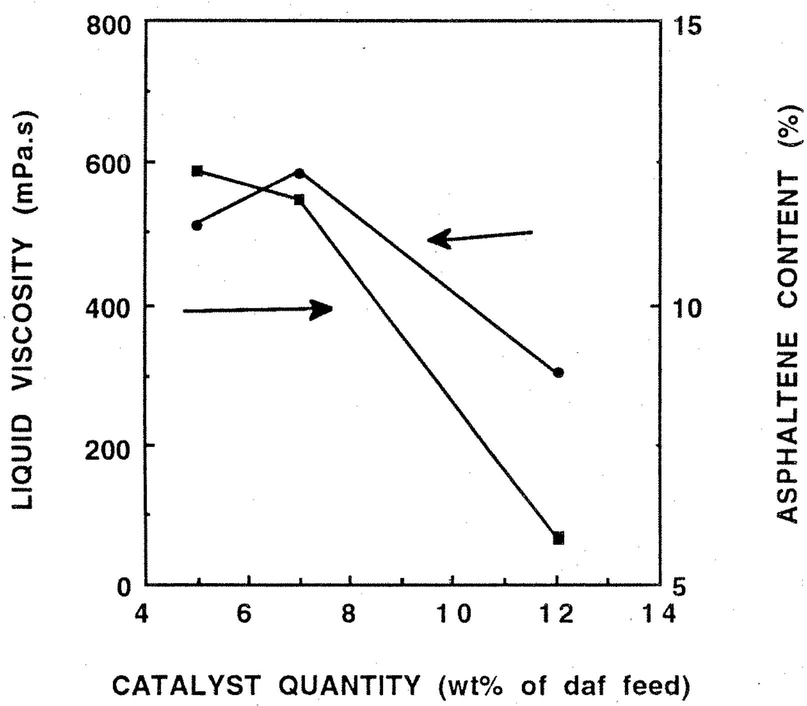


Figure 16: Liquid product viscosity and asphaltene content as a function of catalyst quantity.

SUMMARY OF THE RESULTS:

OVERALL COMPARISON

The results of the present study are summarized in the following section. Figure 17 shows a prorated breakdown of the products, if bitumen and coal were to be processed separately, without any catalysts. This would yield 56.14% liquid, 28.70% coke and 15.17% gas. Figure 18 shows similar breakdown for the case where 70% bitumen would be coprocessed with 30% of coal. This time the liquid yield increases to 66.88%. Figure 19 shows product breakdown for coprocessing of the same feedstock combination with $ZnCl_2$ and as can be seen the liquid yield now goes up to 73.09%. Coprocessing with a mixed catalyst combination ($MoCl_5$ and $ZnCl_2$ and KCl) increases the liquid yield to 75.9%, as can be seen from Figure 20.

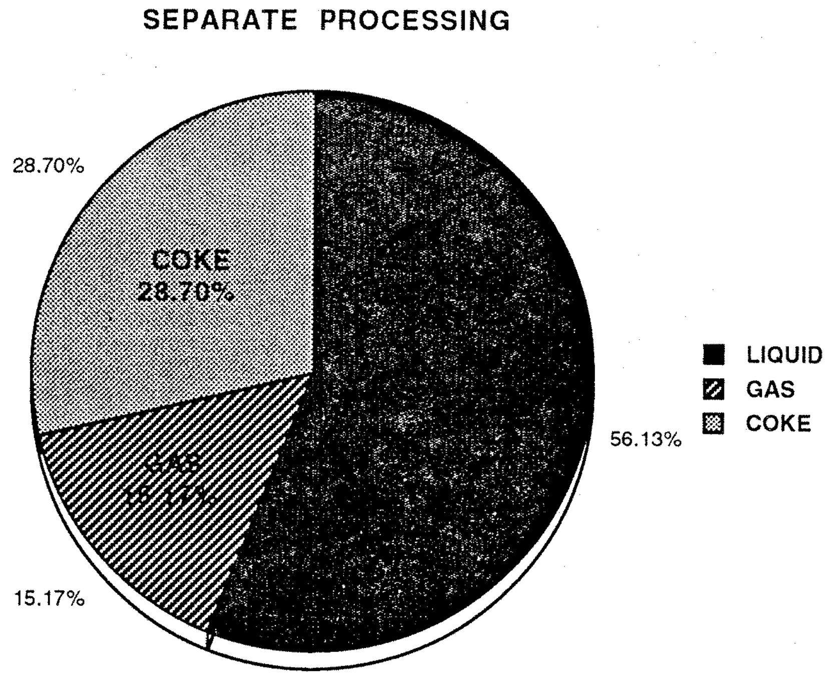


Figure 17: Breakdown of products when bitumen and coal are processed separately .

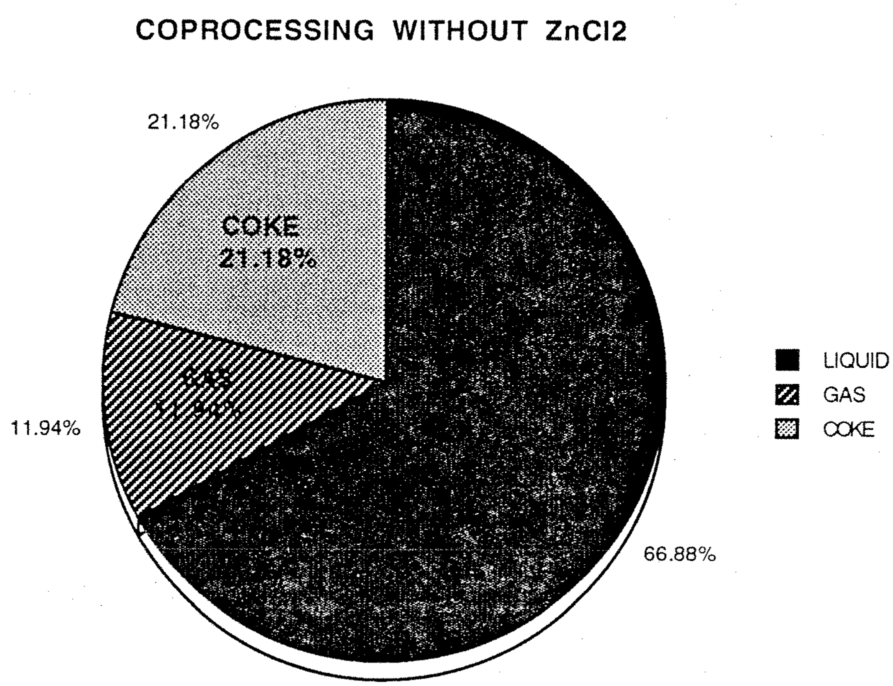


Figure 18: Breakdown of products when bitumen and coal processed simultaneously.

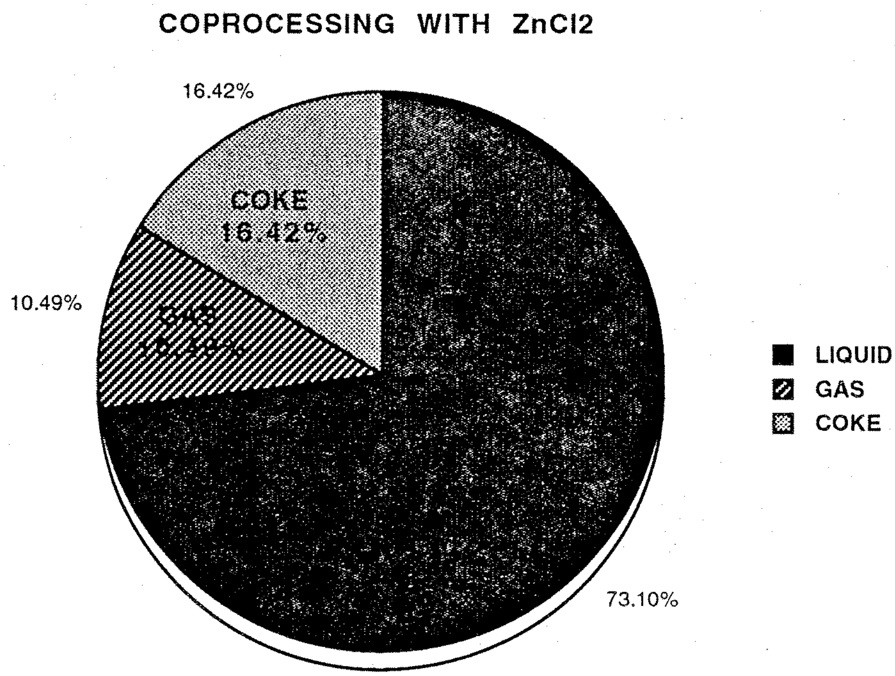


Figure 19: Breakdown of coprocessed products with ZnCl2 catalyst.

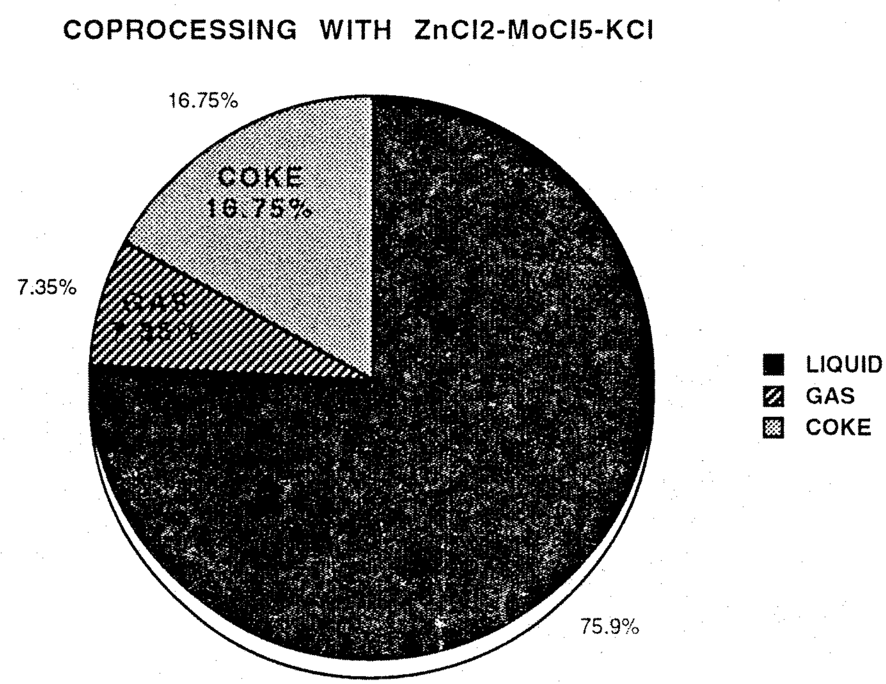


Figure 20: Breakdown of coprocessed products with $ZnCl_2$ - $MoCl_5$- KCl catalyst.

LIQUID YIELDS:

Figure 21 shows liquid yields as a function of different processing schemes. As can be seen liquid yield is the highest with the optimized mixed catalyst formulation.

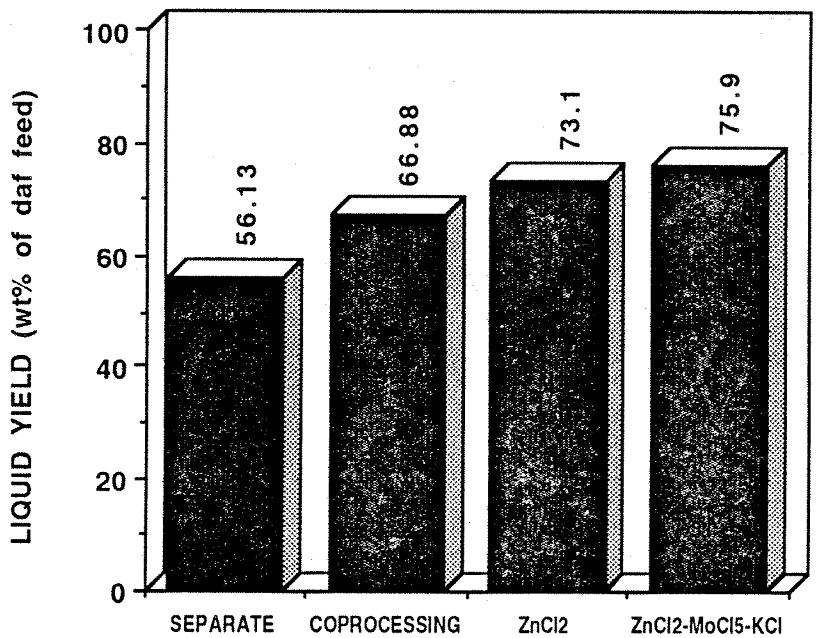


Figure 21: Liquid yields for various processing schemes.

COKE YIELDS:

Figure 22 shows coke yields for different processing schemes. As can be seen coke content can be lowered significantly by coprocessing with mixed catalyst formulation.

GAS YIELDS:

Figure 23 shows gas yields for various processing schemes. Like the coke yield, the gas yield obtained is lowest for the mixed catalyst formulation.

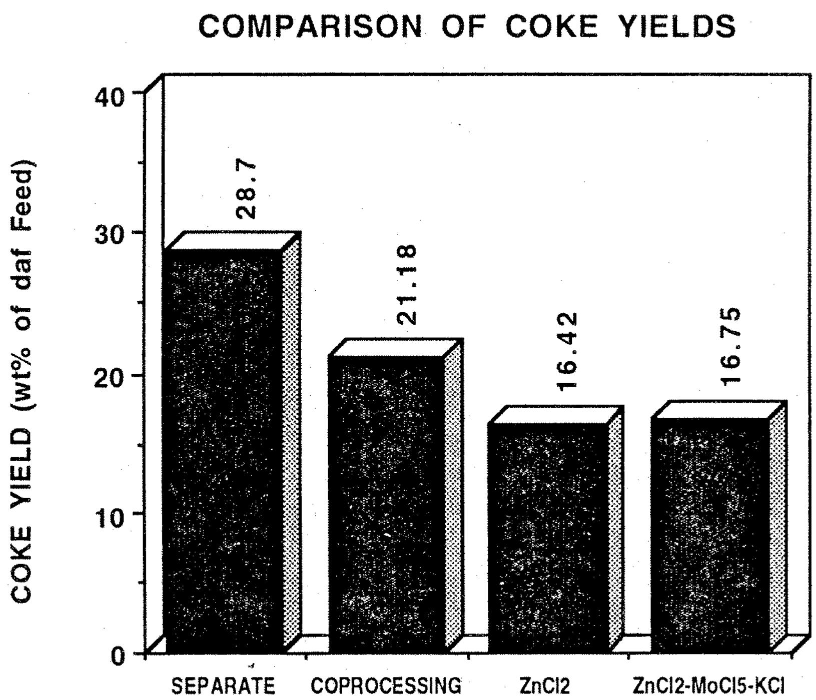


Figure 22: Coke yields for various processing schemes.

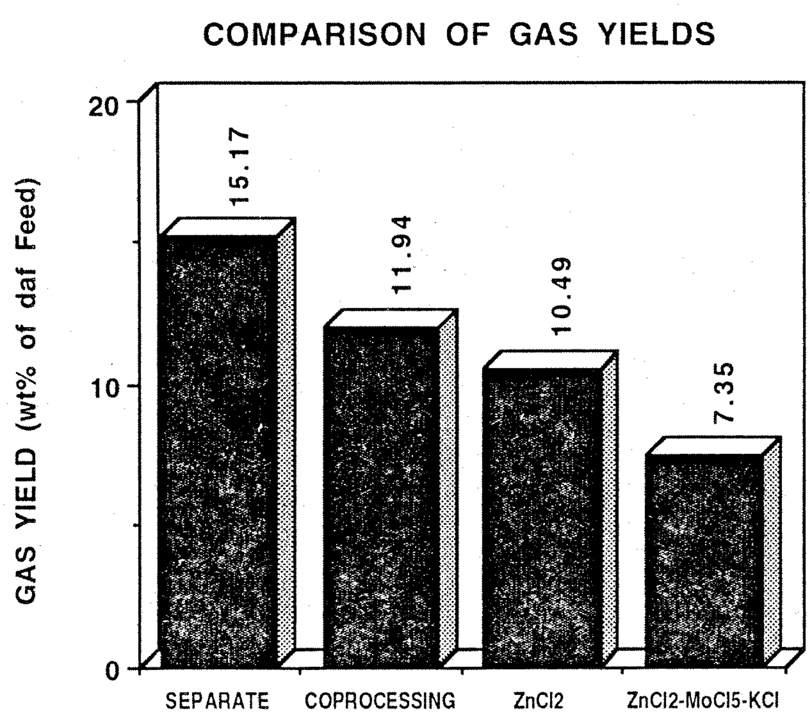


Figure 23: Gas yields for various processing schemes.

LIQUID PRODUCT QUALITY:

The H/C atomic ratios of maltene fractions obtained from various processing schemes are shown in Figure 24. Again, the mixed catalyst provides a superior performance over other processing schemes by providing maltenes with higher H/C ratio.

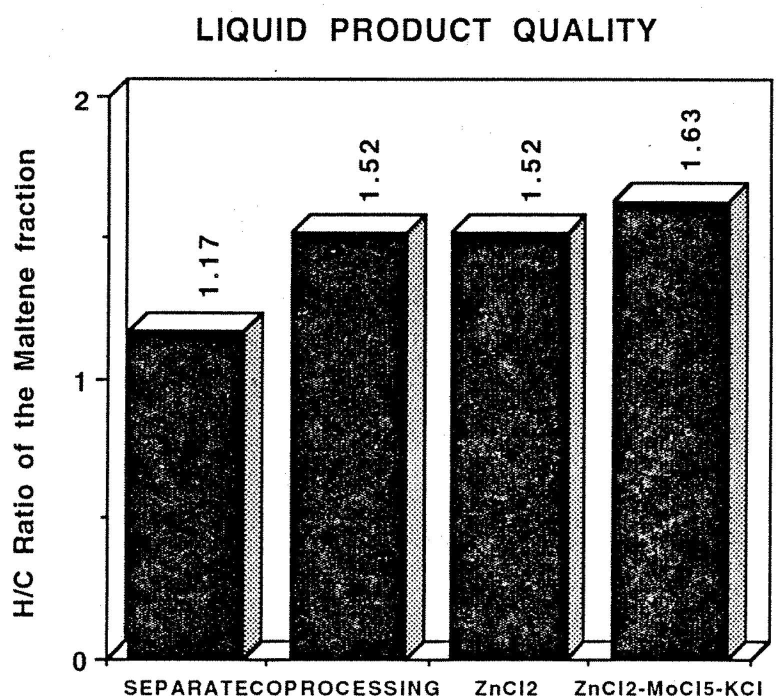


Figure 24: H/C Ratio of the maltenes for various processing schemes.

COMPARISON OF THE COPROCESSED LIQUID WITH OTHER CRUDES

The following figure (Figure 25) compares the quality of the coprocessed liquid obtained with our mixed catalyst formulation with the H/C ratio of other crudes to give the author some idea as to the extent of upgrading which took place during the coprocessing reactions. As can be seen, the coprocessed liquid falls somewhere between light crude and heavy crude in terms of H/C ratio. Therefore, the coprocessed liquid can be further upgraded in existing refineries with slight or no modifications.

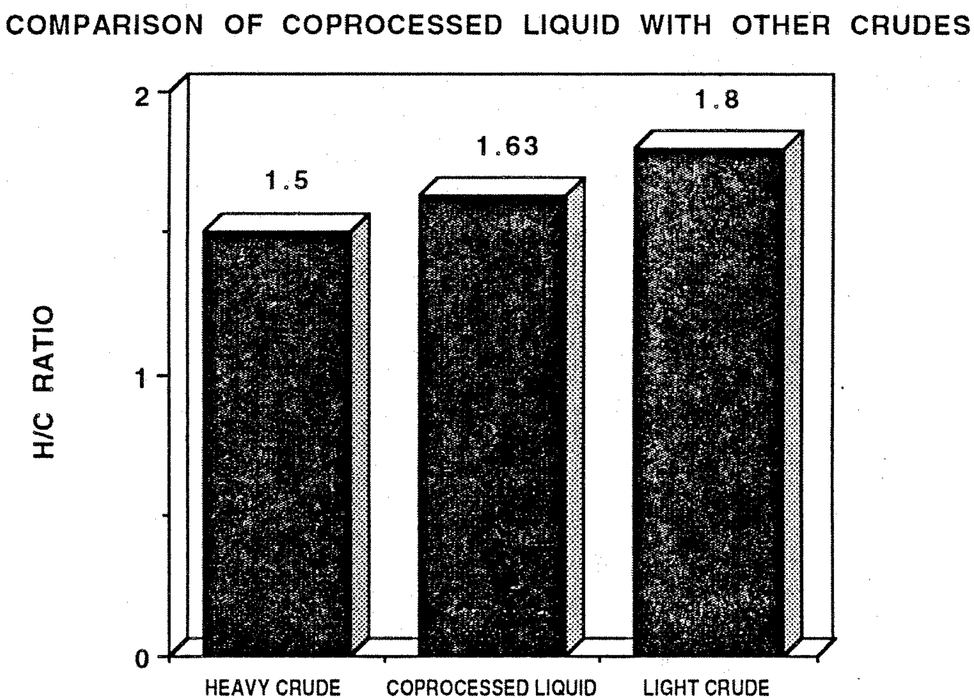


Figure 25: Comparison of H/C ratio of the coprocessed liquid product with other crudes.

CONCLUSIONS

Coprocessing of Athabasca bitumen and a subbituminous Alberta coal in the presence of hydrogen and mixed molten halied catalysts consisting of $ZnCl_2$, $SnCl_2$, $MoCl_5$, CuCl, KCl, NaCl etc. catalyst has been studied under moderate reaction temperatures (typically 400^oC) and moderate H_2 partial pressures (typically 6.89 MPa) in a laboratory reactor. Coprocessing was found to promote coal conversion and maltene production as compared to separate processing schemes. A combination of $ZnCl_2$-$MoCl_5$-KCl catalyst was found to be the best combination, which provided the highest liquid yields, and the lowest gas and coke yields. The quality of the coprocessed liquid obtained is quite high with a H/C ratio of 1.63.

FUTURE WORK

Future work will concentrate on catalyst recover and implementation of the processing scheme in a continuous flow reactor.

ACKNOWLEDGEMENTS

The financial assistance provided by the Office of Coal Research and Technology of Alberta Energy of the Province of Alberta and the Natural Sciences and Engineering Research Council of Canada (NSERC) in support of this work are gratefully acknowledged. Mr. Y. Hu and C.J. Franzen assisted in the laboratory work.

REFERENCES

Boehm, F.G., Caron, R.D., Banarjee, D.K., 1989, “Coprocessing Technology development in Canada”, Energy and Fuels, 3, 116-119.

Chakma, A., Chornet, E., Overend, R.P. and Dawson, W.H., 1989, “Hydrocracking of Athabasca Coker Feed Bitumen Using Molten Halide Catalysts in a Bubbling Microautoclave”, Energy and Fuels, 3, 144-147.

Cugini, A.V., Lett, R.G. and Wender, I., 1989, “Coal/Oil Coprocessing Mechanism Studies”, Energy and Fuels, 3, 12-126.

Curtis, C.W., Tsai, K.J., Guin, J.A., 1987, “Catalytic Coprocessing: Effect of Catalyst Type and Sequencing”, Ind. Eng. Chem. Res., 26, 12-18.

Curtis, C.W. and Pellegrino, J.L., 1989, “Activity and Selectivity of Three Molybdenum Catalysts For Coal Liquefaction Reactions”, Energy and Fuels, 3, 16-168.

Derbyshire, F., 1989, “Review: Role of Catalysis in Coal Liquefaction Research and Development”, Energy and Fuels, 3, 273-277.

Fouda, S.A., Kelly, J.F. and Rahimi, P.M., 1989, “Effects of Coal Concentration on Coprocessing Performance”, Energy and Fuels, 3, 154-160.

Lenz, U. , Wawrzinek, J. and Giehr, A., 1988, "Coprocessing - The Hydrogenation of Lignite Together with Residual Oil", Preprints of ACS Division of Fuel Chemistry, 33(1), 27-41.

Lett, R.G. and Cugini, A.V., 1986, "Coprocessing Studies", Proceedings of the DOE Direct Liquefaction Contractors' Review Meeting, October 22.

Mosphedis, S.E., Hawkins, R.W., Fryer, J.F. and Speight, J.G., 1980, "Use of Heavy Oils (and Derivatives) to Process Coal", Fuel, 59, 647-653.

Mosphedis, S.E., Ozum, B., 1984, "Coprocessing of Alberta Subbituminous Coal with Oil Sand Bitumen",Liq. Fuels Technology, 2(2), 193-209.

Nomura, M., Terao, K., and Kikkawa, S.,1981, "Hydrocracking of Athabasca Asphaltene Over $ZnCl_2$-KCl-NaCl Melts With or Without an Additive of Some Transition Metal Chlorides", Fuel, 60, 699-702.

Nomura, M., Miyake, K., Kikkawa, S.,1982a, "Hydrogenation of Yubari Coal Coated with $ZnCl_2$-MCl_n (CuCl, $CrCl_3$, and $MoCl_5$) Catalyst Melts", Fuel, 61, 18-20.

Nomura, M., Kimura, K., Kikkawa, S.,1982b, "Effectiveness of Both $ZnCl_2$- and $SnCl_2$ -Containing Molten Salt Catalysts for Hydrogenation of Big Ben Coal in the Absence of a Solvent", Fuel, 61, 1119-1123.

Nomura, M., Sakashita, H., Miyake, M. and Kikkawa, S.,1983, "Comparison of Hydroliquefaction Catalyzed by $ZnCl_2$-MCl_n (CuCl, $CrCl_3$, and $MoCl_5$) and $ZnCl_2$ Melts", Fuel, 62, 73-77.

Nomura, M., Kawakami, Y. and Miyake, M.,1986, "Specific Catalysis of $SnCl_2$ Melt for Hydroliquefaction of Subbituminous Coal", Fuel Process. Technol, 14, 261-268.

Song, C., Nomura, M., and Miyake, M.,1986, "Coal Hydroliquefaction Using $MoCl_5$-$NiCl_2$ - Containing Salts as Catalysts", Fuel, 65, 922-926.

Zielke, C.W., Struck, R.T., Evans, J.M., Costanza, C.P. and Gorin, E., 1966, "Molten Salt Catalysts for Hydrocracking of Polynuclear Hydrocarbons", Ind. Eng. Chem. Process Des. Dev., 5, 151-157.

TITLE: Counterflow Reactor (CFR) for Upgrading of Heavy Oil and Heavy Oil/Coal Slurries

AUTHORS: Dr. Dietmar J. Berger[1]

Dr. H. Würfel[2]

Dr. Norman Anderson[3]

AFFILIATION: [1]Canadian Energy Developments Inc.
4222 - 97 Street
Edmonton, Alberta

[2]Gesellschaft für Kohleverflüssigung mbH
Kokerelstrasse, 6620
Völklingen-Fürstenhausen
Germany

[3]Kilborn Inc.
2200 Lakeshore Boulevard West
Toronto, Ontario

ABSTRACT: Since 1984, Canadian Energy Developments Inc. (CED) and Gesellschaft für Kohleverflüssigung mbH (GfK) have performed extensive process development to upgrade the sizeable reserves of Alberta heavy oil, bitumen, and subbituminous coal, as well as mixtures of these energy resources.

The experimental work was performed initially on a Bench Scale unit (BSU), and then on Process Development Units (PDU) to develop the upgrading technologies for heavy oil upgrading, heavy oil/coal co-processing and liquefaction of coal. Specifically, the following technologies were developed:

- PYROSOL Technology, which involves hydrogenation plus coking;

- Co-current Upflow Bubble Reactor Technology - as a one- or two-stage hydrogenation technology; and

- Counterflow Reactor (CFR) Technology.

The operation of the PDU demonstrated the key advantages of the CFR technology over the Co-current reactor. These advantages are:

- no concerns about the settling of solids because liquids and solids are removed from the bottom of the reactor;
- lower recycle gas rates, determined only by kinetics;
- optimum internal recovery of the exothermic heat of reaction, resulting in less severe feed preheating; and
- a favourable profile for the hydrogen partial vapour pressure.

In the CFR system, the oil/coal slurry enters the top of the reactor and flows downward countercurrent to the upflowing recycle hydrogen and product vapours. The product vapours and other gases are withdrawn from the top of the reactor, and unconverted coal, "solids" and liquid hydrocarbons are removed from the bottom of the reactor under level control.

At conversion rates ranging from 74 to 94 wt%, distillable oil yields of 50 to 74 wt% were obtained for coal liquefaction and heavy oil upgrading, respectively (using Alberta sub-bituminous coal and Cold Lake heavy oil).

Conceptual commercial-size upgrading facilities were designed on the basis of the test results. For an Alberta location, capital and operating costs were estimated. These, together with the annual revenue, form the basis for economic feasibility studies. For a 20% DCF-ROE (60/40 debt/equity) a "window of opportunity" opens for co-processing once the heavy oil price rises above approximately \$15.50/bbl (\$100/m^3).

TITLE: Coal Bed Methane - An Alberta Opportunity

AUTHORS: B.A. Rottenfusser, D. Nikols, R.J.H. Richardson, G. Mandryk and W. Langenberg

AFFILIATION: Alberta Geological Survey
Alberta Research Council
Box 8330, Station "F"
Edmonton, Alberta
T6H 5X2

ABSTRACT: The Alberta portion of the Western Canadian Sedimentary Basin contains over 7.7 x 10^{12} tons (7 x 10^{12} tonnes) of coals that range in rank from lignite to semi-anthracite. These coals are distributed throughout the mountains, foothills and plains regions, and range from Jurassic to Tertiary in age. Most of the coal in the plains region is subbituminous, while bituminous coals are common in the foothills and mountains. Promising geologic settings for coal bed methane recovery are found in all three regions.

The first test of Alberta coals for methane commenced in 1974. The Alberta petroleum industry has shown renewed interest recently, and companies have begun to test samples and acquire land positions. Resource estimates of the coal in non-traditional mining areas are being revised and improved. An evaluation by the Alberta Geological Survey of the coal bed methane potential in the province was supported by industry, and the provincial and federal governments.

Plains coals to a depth of approximately 400 metres were explored extensively and evaluated by the Alberta Geological Survey in the early-1980s. The deep coal resources have received considerably less attention, despite being penetrated by thousands of petroleum exploration and production wells. The current

resource evaluation has defined the stratigraphic and structural framework of the coals throughout the province. This was accomplished by producing an extensive suite of cross-sections and maps. Regional trends in subsurface coal rank were defined for the entire province for the first time. This was done by collecting hundreds of vitrinite reflectance measurements.

CARBON FIBER AND CHEMICALS FROM COAL DERIVED LIQUIDS

BY

T. Ohuchi, B. Croy and A. Chambers

Coal and Hydrocarbon Processing Department
Alberta Research Council
P.O. Bag #1310
DEVON, Alberta
T0C 1E0

OCTOBER 1991

ACKNOWLEDGEMENTS

The research project was funded by the Alberta Office of Coal Research and Technology. The project staff are indebted to Dr. Y. Maekawa, the Government Industrial Development Laboratory, Hokkaido, Japan for arranging the acquisition of Battle River Coal derived distillate and pitch from the NEDOL pilot plant.

We express our appreciation to Dr. M. Shiraishi at the National Institute for Pollution and Resources (NIPR) for useful discussion, and for arranging to obtain a pitch feed of commercial GPCF production; also Dr. Y. Sato at NIPR for providing quantitative GC data on dealkylation products.

EXECUTIVE SUMMARY

The abundant coal reserves in Alberta have the potential to provide valuable new feedstocks to a world market other than transportation fuels. Both carbon fiber (CF) from coal derived pitch and aromatic chemical intermediates from coal derived distillates are potential products. This paper presents their applications, current market status and the results of recent experimental work at the Alberta Research Council on production of Industrial Grade CF and chemicals from coal derived liquids.

1. INDUSTRIAL GRADE CARBON FIBER FROM COAL DERIVED PITCH - GENERAL PURPOSE CARBON FIBER (GPCF)

1.1 Overview

About thirty years has passed since carbon fibers (CF) was first produced commercially. Activities of R & D on CF application are still continuing. Carbon fiber is mainly used as a unique reinforcing material with a binder to produce specific properties, such as high strength to weight ratio, anti-electrostatic properties, high temperature resistance and so on. Furthermore, due to their flexibility and strength, CF filaments can be interwoven with other fibers to supply enhanced properties for new engineering materials.

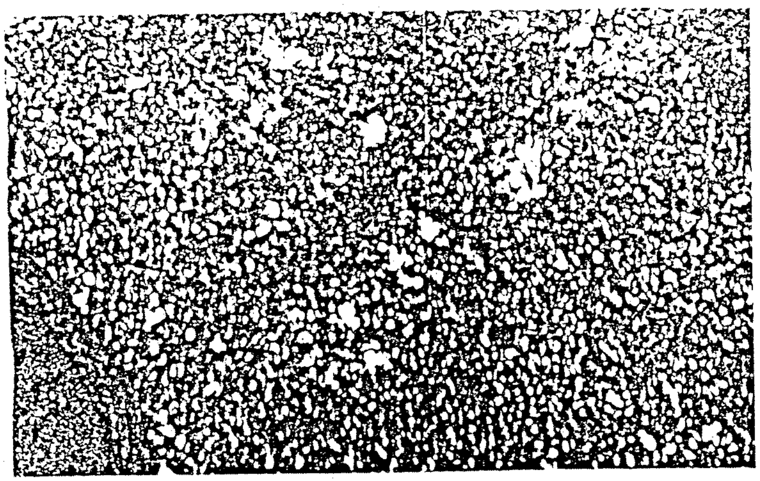
Carbon fiber products are classified according to their strength: the weaker general purpose carbon fiber (GPCF) and the stronger high performance carbon fiber (HPCF). During the thermal treatment of pitch, a liquid crystal material (mesophase/anisotropic), progressively forms. The levels of concentration of mesophase and alignment of crystals provide the final properties of HPCF products. GPCF does not have any mesophase (isotropic), giving a fiber with low strength, but high flexibility. On the contrary, HPCF has high concentration of mesophase accompanied with crystal alignment, giving strong but brittle properties. Temperature of thermal pitch treatment is one parameter controlling final fiber properties. Figure 1 shows polarized light micrographs of mesophase growth of modified Battle River Coal derived pitch. As the temperature rose over about 390°C, optical mesophase (spheres) appears in a isotropic matrix; with the increase of temperature to 420°C the mesophases grow larger. A comparison of the properties of GPCF and HPCF is shown in Table 1, including typical application of present and possible future use.

Three main feedstocks for CF production are polyacrylonitrile (PAN), petroleum and coal derived pitches. Carbon fiber from PAN currently has a larger share

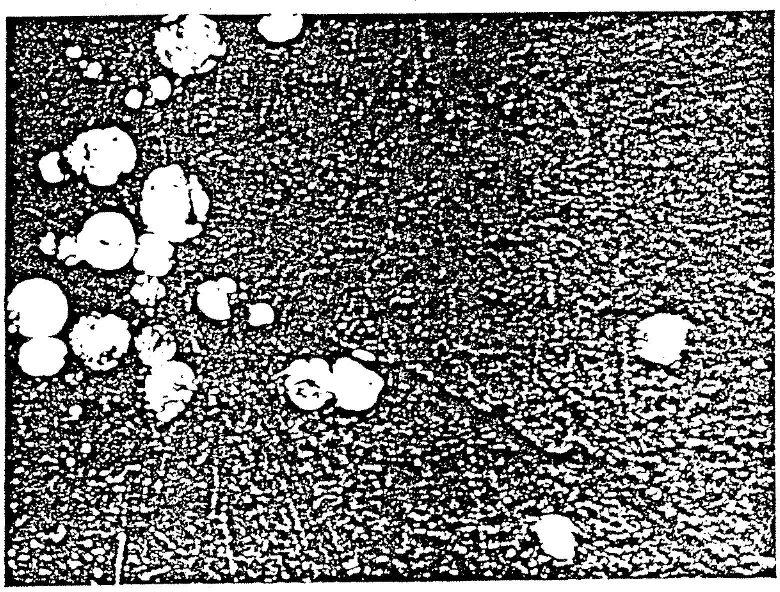
Table 1: Comparison of General Purpose Carbon Fiber and High Performance Carbon Fiber

Properties	GPCF	HFCF
Strength (kg/mm^2)	90 - 85	~ 300
Modulus (10^3 kg/mm^2)	42 - 38	50 - 200
Application at Present	Insulation, gasket filler in plastic, reinforced cement	Reinforcing elements (aerospace, sporting good)
Future Use	Substitution for asbestos, electrode sound absorber, fibrous activated carbon	Heat and chemical resistant pipe and valve, electric conductive resin sheet, automobile parts

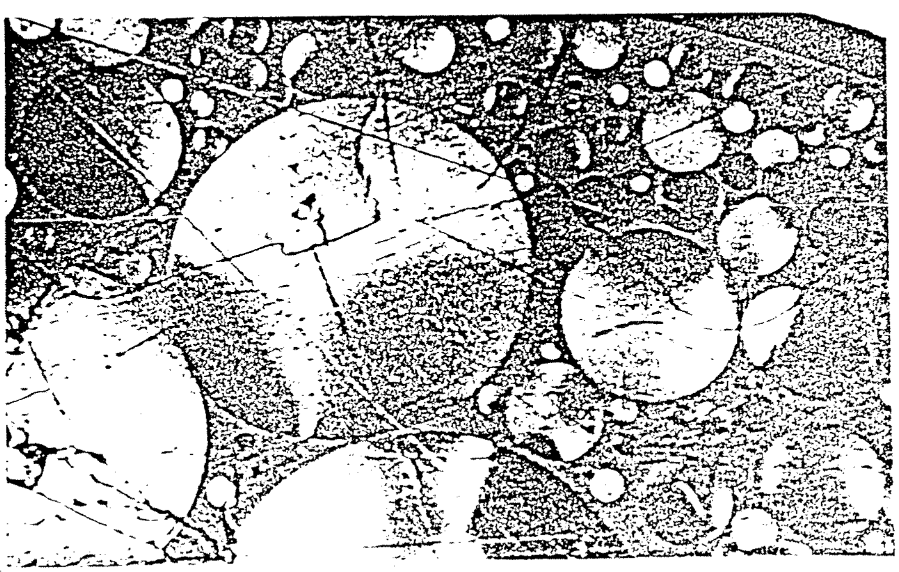
Figure 1: Mesophase Formation from Coal Derived Pitch



Feed
(isotropic)



390^{o}C
6 hr.



420^{o}C
6 hr.

of the market; however, because pitch is a less expensive feedstock and gives a higher yield of CF compared to PAN, pitch derived CF may be expected to increase its share of the market. Pitch yields 85 to 90% of CF, while PAN yields only 40 to 50%. Pitch derived CF generally has higher modules, but CF from PAN has higher tensile strength regarding HPCF in the present market products. Schematically the characteristics are shown in Figure 2 (1). Increasing strength increases stiffness of fiber; causing difficult handling and less resin impregnability. The stiffness of P-100 (product from Amoco P.P. in Figure 2) maybe the maximum limit for CF which can be readily handled as a resin composite (2).

A recent result indicates HPCF having both high modules and tensile strength were made from PAN and pitch in a lab stage (3).

With regard to GPCF, production cost is the major factor in market share. Thus, pitch is more advantageous than PAN as a feedstock to manufacture GPCF. The characteristics of GPCF are relatively high flexibility and low production cost. This allows and opportunity to find new applications, not suitable for HPCF.

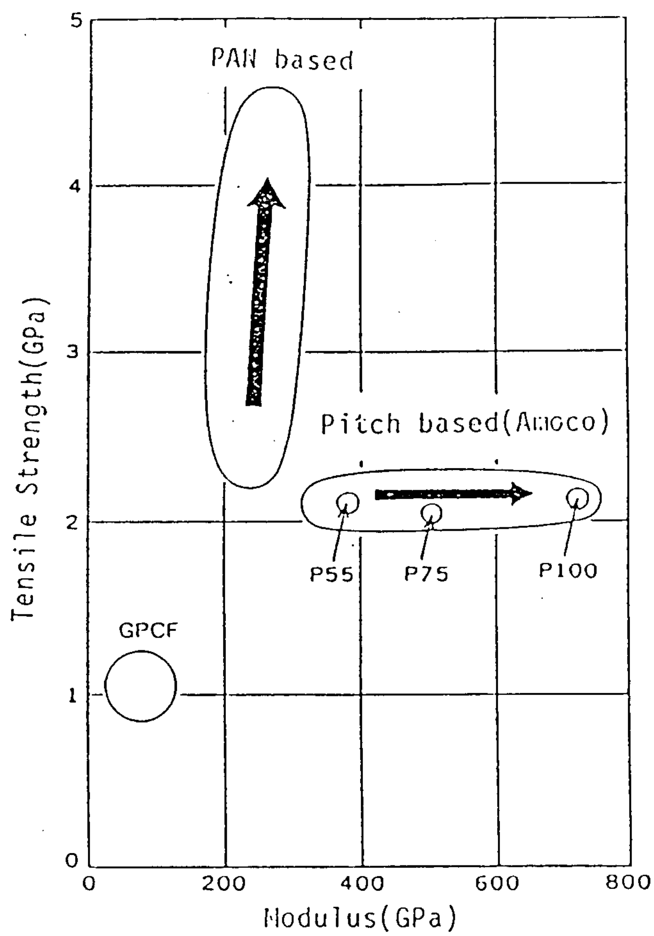
1.2 Applications

Two main sectors of CF applications are the high technology sector, which includes aerospace and nuclear engineering, and the general engineering and transportation sector, which includes engineering components, such as: bearings, gears, cams, fan blades, etc.; and automobile bodies, reinforced cement, asbestos replacement. However, the requirements of two sectors are fundamentally different. For example, the large-scale use of CF in aircraft and aerospace is driven by maximum performance and fuel efficiency, while the cost factor and the production requirements are not critical. The use of CF in general engineering and surface transportation is dominated by cost constraints, high production rate requirements, and generally less critical performance needs. Table 2 shows characteristics and practical applications of CF (4). Table 3

Table 2: Characteristics and Applications of Carbon Fibers

1. Physical strengh, specific toughness, and light weight	Aerospace, road and marine transport, sporting goods
2. High dimentional stability, low coefficient of thermal expansion and low abrasion	Missiles, aircraft brakes, aerospace antenna and support structure, large telescopes, optical benches, waveguides for stable high-frequency(GHz) precision measurement frames
3. Good vibration damping, strength and toughness	Audio equipment, loudspeakers for Hi-fi equipment, pick-up arms and robot arms
4. Electrical conductivity	Automobile hoods, novel tooling, casings and base electronic equipments, EMI and RF shielding, brushes
5. Biological inertness and X-ray permeability	Medical applications in prostheses, surgery and x-ray equipment, implants, tendon/ligament repair
6. Fatigue resistance, self-lubrication, high damping	Textile machinary, general engineering
7. Chemical inertness, high corrosion resistance	Chemical industry; nuclear field; valves, seals, and pump components in process plants
8. Electromagnetic properties	Large generator retaining rings, radiological equipment

Figure 2: Characteristics of Carbon Fiber:
GPCF, HPCF from PAN and Pitch



summarizes applications of GPCF and HPCF in advanced composites (5). This summary indicates GPCF and HPCF are both used in not only a wide range of products, but also overlap in some applications. This suggests that GPCF has a great opportunity to expand more share due to its lower production cost.

The main current and potential applications of GPCF are as follows:

- resin reinforcement (CF reinforced plastic: CFRP)
- cement reinforcement (CF reinforced cement: CFRC)
- asbestos replacement
- anti-electrostatic materials: panel, tile, carpet
- heat insulator
- automobile components
- component in chemical plant
- activated CF: filters for air and water purification
- sound absorber

1.3 Current and Future Markets

Table 4 shows the capacity of CF production in the world, 1988; CF from PAN currently has a larger share of market (6, 7). Table 5 represents major CF manufactures in 1989, using pitch feed (8). The producers of GPCF in 1989 are as follows: Kureha Chemicals, Nippon Carbon, Donac, Ashland, Amoco Performance Products, and Mitsubishi Kasei. They produced 2,000 tons in 1988 (9). Market competition for production of HPCF has been quite severe since 1987/88. R & D activities on HPCF were ceased at many companies; eighteen out of twenty in Japan dropped their plans (3). Nippon Steel, Kawasaki Steel, Nitto Boseki, Indemitsu Kosan, Nippn Oil, Sumitomo Metal and DuPont were active in 1990. Concerning GPCF, the information is not available at this moment. The average annual global growth rate of CF reinforced composites for the 1985-95 decade is projected to be 8-9% world wide, compared to 25-30% in the previous decade (10). This suggests that the growth of HPCF

Table 3: Application of CF in Avanced Composites

(GP-general purpose; HP-high performance)

Matrix	Application(grade)	Related Industry
CF alone	Heat insulator(GP) Activated(GP and HP)	Environmental, Medical Electronics, Automobile Aircraft, Nuclear reactor
	Sealing(GP)	Chemical, Petrochemical, Petroleum, Automoble
Plastic resin	Engineering material: frictionless, electric conductivity, anti-corrosive (GP and HP)	Electric, Electronics, Machinery, Automobile, Aircraft, Chemical
C/C	Structural composite: reduced weight, primary and secondary material of design with strength(HP)	Sporting goods, Medical, Space, Aircraft, Automoble, Electric
	Abrasion material(GP and HP)	Space
Metal	Tribological properties (GP and HP)	Automoble, railway, Aircraft, Machinery
	Carbon, graphite(GP)	Steel, Electric
Inorganic	Material for cell (GP and HP)	Electric, Automoble
	Building(GP and HP)	Construction, Ship, Housing, CFRC, Asbestos replacement

——— in use; - - - - - utilization expected in future

Table 4: Production Capacity of Carbon Fiber in 1988

Feedstock	Manufacture	Trade name	Capacity
Petroleum derived pitch:			
	Kureha Chemicals	Kureca	900
	Amoco Performance Products	Thoreal	230
	Ashland Petroleum Co.	Carboflex	20
	Nippon Oil	Granoc	50
	Kashima Petroleum	Carbonic	12
	Toa Nenryo		12
	Showa-Shell		10
	Idemitsu Kosan		1
	Mitsubishi Petroleum		1
Coal derived pitch:			
	Mitsubishi Kasei	Dialead	500
	HITCO	HITEX	60
	Stackpole Carbon	Panex	40
	Donac	Donacarbo	300
	Nippon Carbon		36
	Nippon Steel		10
	Kawasaki Steel		12
	Nitto Boseki		20
PAN:			
	Japan		3350
	Asia		380
	U.S.A.		2760
	Europe and Others		1250
	Total		9954

Table 5: Carbon Fiber Manufactures Using Pitch in 1989

Name	Feedstock	Grade	Capacity t/y
Amoco P. P.	Petroleum mesophase	HPCF, Yarn	230
Toa Nenryo	Petroleum mesophase	HPCF, Yarn	12
Nippon Oil	Petroleum mesophase	HPCF, Yarn	5
Kashima Petroleum	Petroleum mesophase	HPCF, Yarn	12
Kureha Chemicals	Petroleum isotropic	GPCF, Felt/Chop	900
MItsubishi Kasei	Coal mesophase	HPCF, Yarn	500
Donac	Coal mesophase	HPCF, Yarn	10
Nippon Steel	Coal mesophase	HPCF, Yarn	10
Donac	Coal isotropic	GPCF, Felt/Chop	300
Nippon Carbon	Coal isotropic	GPCF, Felt/Chop	36
Nitto Boseki	Coal isotropic	GPCF, Felt/Chop	20

HPCF- High Performance Carbon Fiber
GPCF- General Purpose Carbon Fiber

consumption including PAN products will be similar.

Although projected growth rate to GPCF is not available, R & D of the application of GPCF is very active, especially in automobile and construction industry.

Regarding GPCF, a key factor is process development to produce less expensive GPCF. Lower cost would potentially have large impact on application, such as CFRC.

1.4 Study of Process Development on GPCF Production at the Alberta Research Council

In the fiscal year 1990/91, samples of GPCF were produced from both Battle River coal derived pitch and coal/heavy oil coprocessing pitch at the Coal and Hydrocarbon Processing Department.

The most important step in GPCF production is the pitch modification prior to production of the carbon fiber. The modified pitch must have two distinct characteristics:

1. A softening/melting point above 180°C.
2. Good 'spinnability' in the filament making step.

Figure 3 gives the flow diagram of the GPCF production. Table 6 summarizes physical properties of GPCF's from coal derived, produced at ARC, and commercial pitches of GPCF production. The extensive study on a scale-up process has been ongoing.

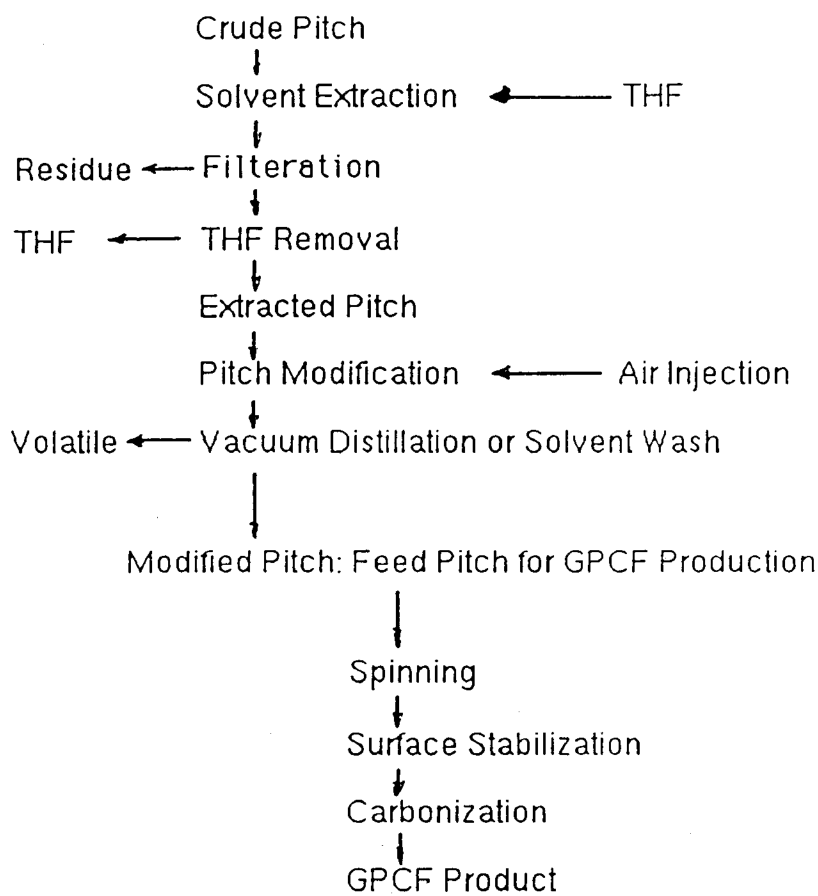
Carbonization was performed at 900°C and 1100°C. Filament diameters, tensile strength, tensile modulus and elongation were similar for the three carbon fibers, indicating that, although there is room for improvement, the GPCF produced in the lab was acceptable. GPCF produced from the commercial pitch did give superior (by about

Table 6: Properties of GPCF Products

Sample	Filament Diameter (um)	Tensile Strength (MPa)	Tensile Modulus (GPa)	Elongation (%)
Coal derived carbonization at 900°C	51.5 ± 18.4	146 ± 42	47.9 ± 7.7	0.74 ± 0.16
Coal derived carbonization at 1100°C	51.1 ± 12.9	180 ± 99	45.8 ± 10.1	0.83 ± 0.21
Commercial pitch carboniztion at 1100°C	52.3 ± 15.0	251 ± 97	51.9 ± 23.5	1.00 ± 0.18
Commercial GPCF *	13 - 18	650 - 800	35 - 40	1.8 - 2.0

* Donacarb S from Osaka Gas

Figure 3: Process for GPCF Production from Battle River Coal Derived Pitch



40%) tensile strength over that produced the coal derived pitch. Compared to a commercial (GPCF) product (Table 6), filament diameters were about four times larger, while tensile strength was about 25% of the commercial GPCF. The tensile modulus was similar. With decreasing diameter tensile strength increases; to date, very small diameter filaments have been produced.

Recommendation from this first study included refinements to the production process used plus scale up to produce sufficient quantities for product application testing. This work is ongoing in FY 1991/92.

2. CHEMICAL INTERMEDIATES FROM COAL DERIVED DISTILLATES - TWO OR MORE CONDENSED AROMATIC RINGS

2.1 Overview and Applications

Recently the fine chemical and plastic industries have been seeking new starting feedstocks for specialty chemicals; focusing primarily on compounds with two or more condensed aromatic rings with functional groups. In the world, about fifteen million tons/year of coal tar is produced and 600 different components of coal liquids have been indentified. However, individual desirable component concentrations are low (11 and 12). Table 7 summarizes the main distillable components above a concentration of 0.3%. These components account for about 35% of coal tars. Typical major components of coal tar are the unsubstituted condensed aromatic hydrocarbons, such as: naphthalene, phenanthrene, anthracene, and the hetrocyclic aromatics like dibenzofuran, carbazole and quinoline. These two types of hydrocarbons account for about 20% of the liquids.

By modern separation techniques, including distillation, crystallization and extraction, only 20 to 30 compounds may be isolated (13). In West Germany chemical industries based on coal were developed extensively (14). Figure 4 illustrates the distribution of 1.4 million tons of coal derived tar products to markets in 1974, supplying

57% of the basic chemicals for the organic chemical industry. The distribution of individual manufacturing is shown in Figure 5; one third was used for dye synthesis and the manufacture of plastics (adhesives), and another third was used for plant protecting agents and pharmaceuticals, derived from the distillable fractions. The most abundant component of distillates is naphthalene, making up about 9% of coal liquids. It is almost exclusively obtained from coal tar with the exception of the quantities produced in the U.S.A. through dealkylation of aromatic petroleum fractions. The current applications of naphthalene are shown in Figure 6 indicating 70% of naphthalene is used for production of phthalic anhydride by air oxidation of one of the aromatic rings. Instead of the destruction of naphthalene rings, industry has investigated the use of naphthalene derivatives in advanced polymer and fine chemical industries. As of November 1988, the utilization of naphthalene is shown in Table 8 (15). Note that carboxylic acids are the main functional groups, suggesting an economical process for production of naphthalene carboxylic acid may help to deveop a new Alberta chemical industry.

Compared to existing products made from naphthalene derivatives, little information is available regarding derivatives of three to five ring condensed aromatics and nitrogen containing products. Figure 7 and 8 give some of the historical utilization of these materials. Increasing condensation of aromatic rings increases the complexity of possible reactions and the rigidity of product molecules as feedstocks. Therefore, as chemicals or feedstocks for advanced polymer production, five condensed aromatics may be the maximum size of polyaromatics which could be readily handled in existing equipment (16).

Recent development of advanced engineering materials, applications, such as specialty monomers, liquid crystals and photosensitive materials, seek greater than two fused aromatics (naphthalene) to add distinct properties to final products.

Table 7: Distillable Components of Coal Tar Above Concentration of 0.3%

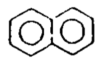
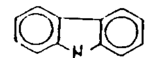
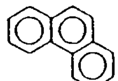
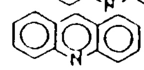
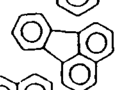
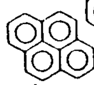
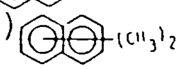
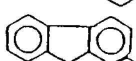
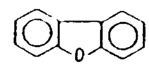
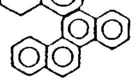
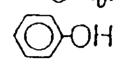
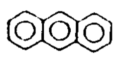
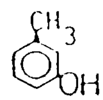
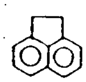
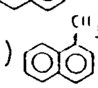
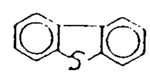
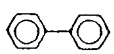
COMPOUND(% CONCENTRATION)	FORMULA	COMPOUND(% CONCENTRATION)	FORMULA
Hydrocarbons		Nitrogen compounds	
naphthalene(9.0)		carbazole(1.5)	
phenanthrene(5.0)		acrydine(0.6)	
fluoranthene(3.3)		quinoline(0.3)	
pyrene(2.1)			
dimethylnaphthalene(2.0)		Oxygen compounds	
fluorene(2.0)		dibenzofuran(1.0)	
chrysene(2.0)		phenol(0.5)	
anthracene(1.8)		m-cresol(0.4)	
2-methylnaphthalene(1.4)			
indene(1.0)		Sulfur compounds	
acenaphthene(0.9)		benzothiophene(0.3)	
1-methylnaphthalene(0.6)		dibenzothiophene(0.3)	
biphenyl(0.4)			
benzene(0.3)			

Table 8: Naphthalene Derivatives and their Applications

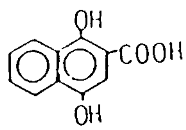
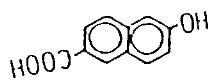
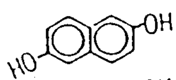
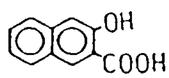
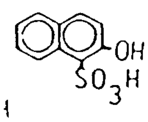
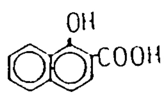
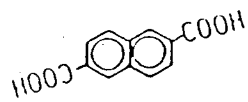
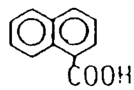
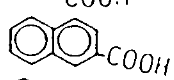
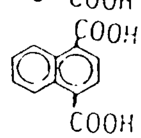
NAME	FORMULA	APPLICATION
1,4-dihydroxy-2-naphthoic acid		photography
2-hydroxy-6-naphthoic acid		advanced material
2,6-dihydroxy-naphthalene		advanced material
β-oxy-naphthoic acid		dye, pigment
β-naphthol sulfonic acid		dye
1-hydroxy-2-naphthonic acid		photosensitive plastic
2,6-naphthalene dicarboxylic acid		advanced material, liquid crystal
α-naphthoic acid		dye for photography
β-naphthoic acid		dye, pigment
2,3-naphthalene dicarboxylic acid		color developer
1,4-naphthalene dicarboxylic acid		advanced materials

Figure 4: Distribution of Coal Tar Products
(1.4 million tons/1974, West Germany) (Collin et al., 1980)

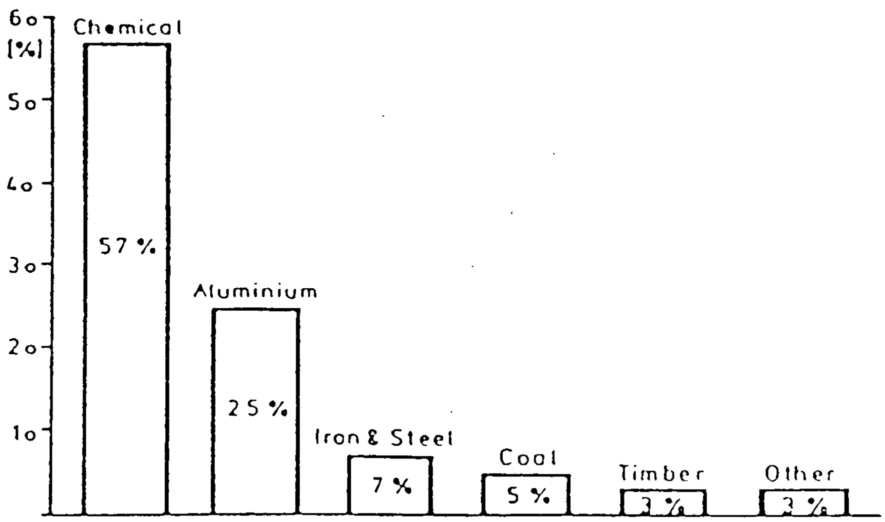


Figure 5: Distribution of Chemical Products from Coal Tar
(Collin et al., 1980)

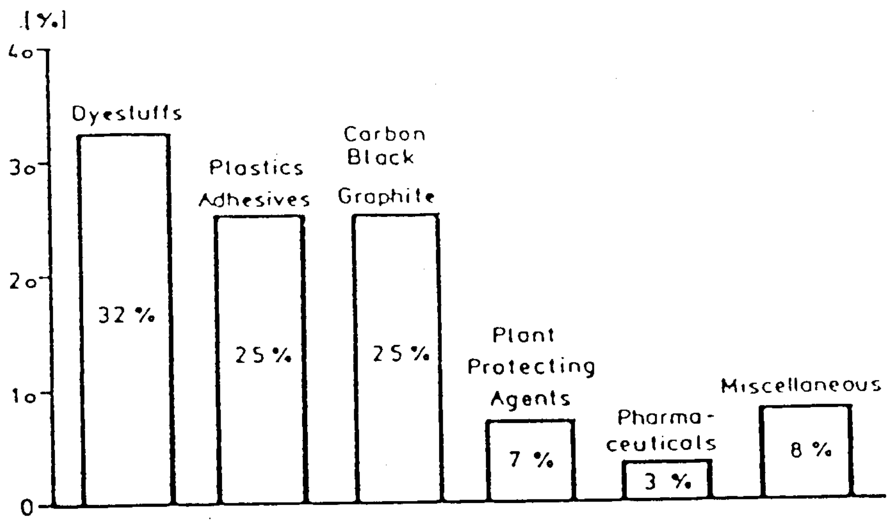


Figure 6: Application of Naphthalene
(Collin et al., 1980)

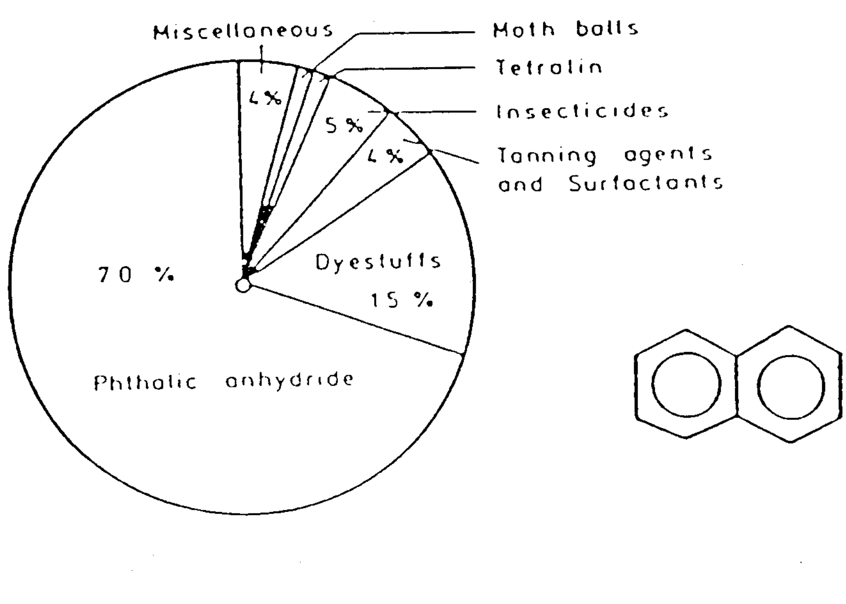


Figure 7: Utilization of Three to Four Condensed Aromatics

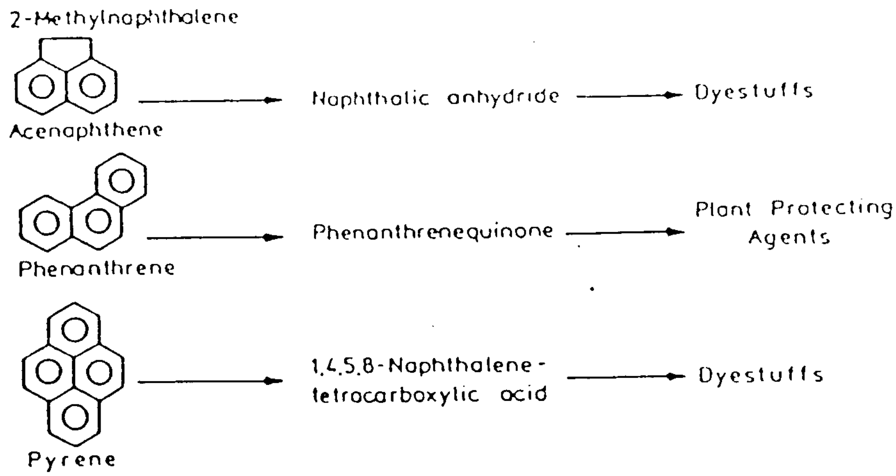
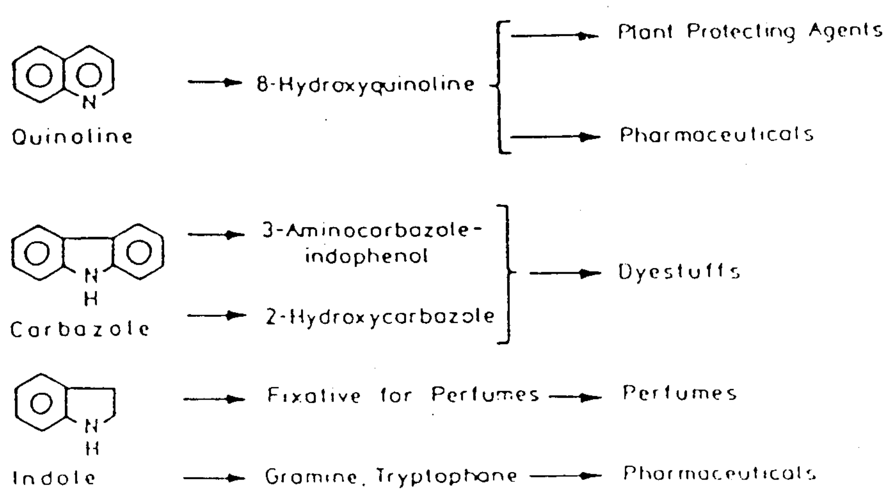


Figure 8: Utilization of Nitrogen Containing Components



2.2. Study of Process Development on Chemical Intermediates Production From Coal Derived Distillates

Feedstocks:

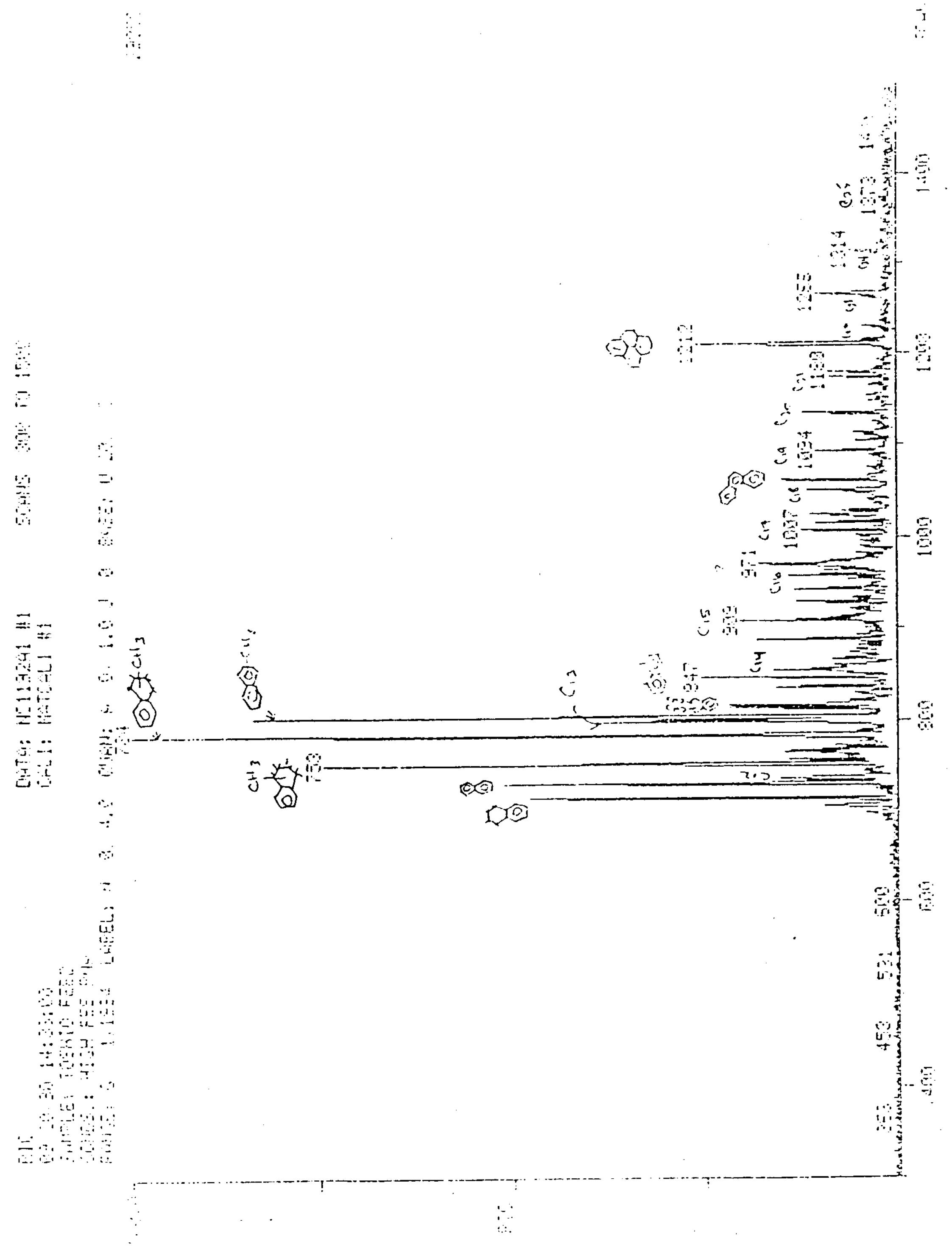
Alberta subbituminous coal derived distillate fractions consist of mainly two to four aromatic conjugated rings, although alkyl side chains on aromatic rings at various positions make variety of isomers. Figure 9 represents a typical gas chromatograph (GC) of feed distillate. This feedstock distillate was obtained from Mitsui Engineering and Shipbuilding Company, Japan. Using a NEDOL plant (one ton/day), Alberta subbituminous coal from the Battle River area had been liquefied with recycled coal derived liquid. The naphtha fraction was removed in the process. The feedstock distribution (boiling point) was as follows: no naphtha, 60.6% of light oil (200-320°), 37.0% of heavy gas oil (321-500°), and 2.4% of +500°C fraction.

Dealkylation:

If side chains are removed, the products can be isolated readily by conventional separation techniques. These components could be intermediates for further synthesis of final products, such as: fine chemicals, monomers for unique polymers and pharmaceuticals. Processes for dealkylation of the coal derived distillates were studied.

The dealkylations were performed above 700°C under hydrogen or methanol flow. Figure 10 shows a GC of products. Compared to the GC in Figure 9, the side chains were successfully removed. One promising condition was determined which gave main products of naphthalene (25.3%) indene (16.5%), 2-methylnaphthalene (11.5%) and acenaphthylene (4.8%). Equipment and procedures have been established. Recommended further work would include: different feedstocks, especially tar process, completion of GC set-up and initial investigation of separation techniques.

Figure 9: Feed Distillate from Battle River Coal Liquefaction



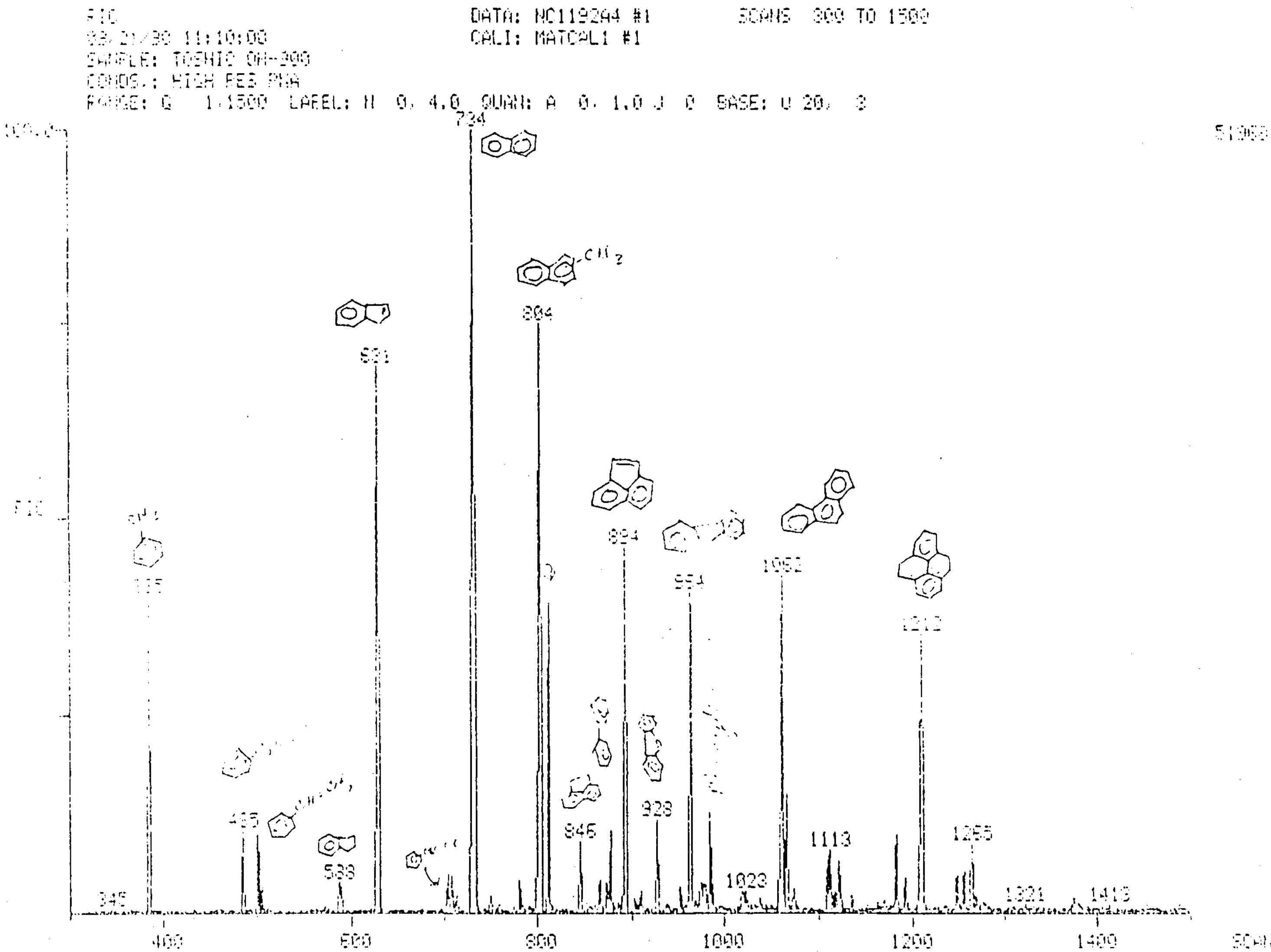


Figure 10: Products from Dealkylation

3. CONCLUSIONS

Industrial grade carbon fiber (GPCF) was produced at a laboratory scale from Alberta coal derived pitch and the property assessments were performed.

In addition, GPCF was produced from coprocessing (coal heavy/oil) pitch. GPCF from Alberta coal could potentially be used in advanced composite production, such as carbon reinforced cement.

By dealkylation with hydrogen or methanol, alkyl side chains on aromatic fractions in Alberta coal derived distillate were removed and concentrations of major components identified. The products could be suitable chemical intermediates for fine chemicals and unique monomer synthesis for advanced materials.

Future Work

Scale up of GPCF production to produce samples for performance/application testing is ongoing. Work is required in specialty chemical production to study a process development and obtain data for economic assessment.

4. REFERENCES

1. Otani, S., Gendai Kagaku, March 1983, p. 14.
2. Personal communication with a researcher at a private company, 1990.
3. Personal communication with Dr. M. Shiraishi, at National Institute for Pollution and Resources, Japan, 1991.
4. Donnet, J.B. and R. Bansal, Carbon Fibers, p. 370, Marcel Dekker Inc., New York, 1990.
5. Okuda, K., CF and Composite Materials, p. 6, Kyonitsu, Tokyo, 1988.
   * extra information added by T. Ohuchi, ARC.
6. Kashihara, K., JETI, Vol. 37, No. 3, 101, 1989.
7. Yamamoto, S., JETI, Vol. 36, No. 7, 67, 1988.
8. Mitsuo, S., JETI, Vol. 37, No. 3, 106, 1989.
9. Nikkan Kogyo Shinbun (Newspaper), March 28, 1989.
10. Fishman, N., Modern Plastics, p. 73, July 1989.
11. Kurata, N., "A View of Polycyclic Aromatic Compounds for Specialty Chemicals", Chemistry and Industry, Vol. 41(1), 147, 1988.
12. Lauer, J.C., D.H. Valles Hernandez and D. Cagniant, "Improved Characterization of Coal Tar Distillation Cuts (200-500°C)", Fuel, Vol. 67, 1273, 1988.
13. Okazaki, H. and M. Nozawa, Fine Chemicals, 6, 1986.
14. Collin, J., "Chemical from Tar Processing", World Conference of Future Sources of Org. Raw Materials, Toronto, Canada, July 1978, Oxford U.K. Pergamon Press, pp. 283-297, 1980.
15. Okui, N. and S. Taketani, "Separation and Application of Bicyclic Aromatics from Coal Tar", Chemical Economics, p. 76, November 1988.
16. Panel Discussion, "Application of Polyaromatics", Petrotech, Vol. 12(2), 78, 1989.

Coal Fired Steam Generation for Heavy Oil Recovery

Presented by

Firmin, K., MSc
Shell Canada Limited

Slide One

In the Province of Alberta, we are fortunate in having substantial reserves of oil, natural gas as well as coal and in particular, contained within the oil resource base, the vast reserves of heavy oil associated with the Athabasca, Cold Lake and Peace River Oil Sand deposits. It is also somewhat fortunate that these heavy oil reserves are in close proximity to large reserves of low cost, low sulphur sub bituminous coal reserves which may be considered as an alternative fuel for steam generation.

Industry produces some 21,000 cubic metres per calendar day of heavy oil and bitumen by in-situ recovery methods involving the injection of high pressure steam into the reservoir for mobilizing and recovering the oil. The actual amount of steam required depends on the nature of the reservoir and the particular recovery technology used.

This steam requirement is currently satisfied through the use of standardized natural gas fired oil field steam generators.

Natural gas has been selected as the preferred fuel for steam generation because of its relatively low cost per unit of energy, the ease with which it can be transported to the operating sites, and the simplicity of gas utilization technology. Gas has the additional benefits of being the 'cleanest' burning fossil fuel while at the same time providing the lowest levels of gaseous emissions per unit of available energy.

As we know, within Alberta, natural gas is in plentiful supply and therefore competitively priced compared to other fuels. Consequently a substantial general increase in the demand for natural gas has occurred. Large increases in demand could lead over time to significant future price increases although at present, customers are enjoying very low natural gas prices. If significant price increases do occur, then the profitability of insitu oil sand operations, with a large portion of their operating costs fuel related, would be adversely effected.

It is for this reason that in 1985 heavy oil producers along with government commissioned a study to look at the opportunities for replacing natural gas used for steam generation with an alternate fuel. The study, carried out by L.A. Smith Consultants concluded that coal would be a viable alternative and would be the next best from a cost standpoint.

Slide 2

Coal is Alberta's most abundant fossil fuel resource accounting for 70 percent of all fossil fuel reserves. It is also the most underutilized resource in terms of its overall reserve base. At current consumption rates the life of Alberta's coal reserve far exceeds the life of all other fuels combined.

Eighty four percent of Alberta's coal reserve, is sub-bituminous and low in sulphur content. Approximately one third of this sub-bituminous coal reserve, is amenable to low cost surface mining techniques: however, its relatively low energy content limits the distance over which it can be transported and remain economically competitive with other fuels.

Slide 3

This next slide provides a typical analysis of sub- bituminous coal presently mined in Alberta for use in electrical power generation. Of particular note is the low sulphur content.

Slide 4

Alberta's major oil sand deposits are located in three distinct geographical areas - Peace River, Athabasca and Cold Lake with insitu bitumen production coming from the Peace River and Cold lake areas.

The reserves of sub-bituminous coal are within reasonable proximity to these oil sand resources. Unfortunately however, the rail infrastructure within Alberta was not put in place with a view to moving coal to the heavy oil areas.

Rail distances from Edmonton are in the order of 350km to Cold Lake and 500km to Peace River. Whereas at modest tonnage levels minimal rail upgrading is required to Peace River, rail access to Cold Lake requires significant upgrading at a cost only supportable at large tonnage movements. Other transportation opportunities such as coal slurry pipelining could be a feasible alternative.

For coal to be considered as a viable alternative to natural gas various technical issues need to be addressed including, environmental emissions control; transportation and coal utilization (combustion and handling). A broad range of technical options are available to deal with these issues.

In order for the technical options to be evaluated economically against the use of natural gas, views have to be taken as to future price projections for both coal and natural gas. It is expected that the future price of sub-bituminous coal in Alberta will be relatively fixed in real terms due to the large resource base and the limit by its energy content for transportation to alternate markets. On the other hand, the future price of natural gas is less certain as it is more transportable and therefore exposed to the international forces of market supply and demand.

However, the conversion to coal once taken can provide the heavy oil producer with a reasonably assured fuel supply over a long time frame together with a relatively stable energy cost.

In considering coal for steam generation, one of the critical factors to its acceptance is its environmental performance. Particularly since natural gas the economic choice is also seen as the most environmentally acceptable fossil fuel.

Coal firing results in the emissions of three major gaseous components; the oxides of sulphur (SO_2), Nitrogen (NOx) and carbon (CO_2). Emphasis to date has been placed on the impacts of SO_2 and NOx as the precursors to acid rain, and strict environmental standards have been put in place to limit their emission.

The current Canadian federal guidelines and Alberta provincial regulations, for maximum emissions of SO_2 and NOx from large coal fired plants, are 0.6 lbs per million BTU (258 ng/J), for both gases. It is expected that coal fired steam generators for heavy oil recovery will also have to meet these limitations.

Slide 5

A review of the technologies available for reduction of SO_2 and NOx from coal combustion as applied to heavy oil recovery steam generators has been carried out by Monenco Consultants Ltd. The review looked at both the removal capabilities and the cost for the various technologies.

This next slide shows the various technologies that were considered. Combustion control technologies were found generally to be less costly than post combustion flue gas clean up.

Other environmental considerations which need to be addressed in the design and operation of coal fired steam generators include the control of particulate emissions, that is fly ash and coal particles, the disposal of boiler and fly ash, the management of the boiler feed water and subsequent disposal of boiler blowdown water.

As I have mentioned previously, steam generation for heavy oil recovery in Alberta is accomplished by the utilization of 190GJ/hr gas fired once-through steam generators. Multiples of these generators are normally clustered in groups and centrally located to provide the total steam requirements for a particular area of the reservoir.

Slide 6

This slide shows a typical arrangement of steam generators clustered in groups which are usually made up of four to six units.

The size of the individual generator is determined by the desire of the heavy oil producers to fabricate the generator in a manufacturing centre and transport the generator, normally in two to three components, by road for final erection at the heavy oil production site. The ultimate size of the generator is therefore determined by the maximum load allowable by road transportation.

Slide 7

The gas fired steam generator consists of a once-through single pass water tube boiler with a feed water heat exchanger, convection section and radiant section. The feed water is typically preheated to prevent external tube corrosion in the convection section where heat is scavenged from the flue gas. From the convection section the water passes to the radiant section where the water tubes are exposed to the flame and the water is vaporized. Approximately 80% of the feedwater is converted to steam leaving 20% of the water to carry off remaining dissolved solids and to prevent solid deposits forming on the tube walls.

Slides 8,9 and 10

The quality of the feed water is a most important factor and dictates the design and operation of the steam generator.

Two sources of water are typically available for use by the heavy oil producer: surface water from lakes and rivers requiring minimal treatment ; and produced water from the reservoir containing a high level of dissolved solids requiring significant pretreatment to soften the water and reduce the level of dissolved solids.

The concern with accessibility to large volumes of surface water for steam generation and the need to subsequently dispose of produced water, generally by injection into a suitable subsurface formation, require that in large scale operations produced water is recycled for use in steam generation. To enable the recycled produced water to be used for steam generation, pretreatment of the produced water, currently with hot lime softeners, is necessary. In addition, the steam generators are operated at low heat flux and 80% steam quality to prevent rapid vaporization of the feed water and consequent deposition of solids on the tube walls.

In the application of coal firing to steam generation for heavy oil recovery, the need for once through generation of 80% quality steam at low heat flux are critical design parameters and differ from the more typical application of pulverized coal firing in boilers associated with electrical power generation.

In 1988, a joint Canadian Industry/Government committee commissioned a study to design and cost a pulverized coal fired steam generator suitable for use in insitu heavy oil applications. The study was carried out by Delta Projects Inc. in conjunction with Combustion Engineering. Following the study it was intended to use the design as the basis for the construction of a prototype steam generator to demonstrate the technology. The prototype generator was to be designed to handle a variety of Alberta coals and would need to meet government guidelines for limiting the gaseous emissions of SO_2 and NOx.

Various steam generator and burner configurations were evaluated in the study. The study also indicated that single rather than multiple burners would be most suitable for this type of application.

Slide 11

This slide shows the various steam generator configurations considered in the study.

Configurations A and B incorporate a single burner firing vertically downward into a square section furnace with tubes laid out in a horizontal or inclined configuration to allow self draining and averaging of heat flux. The gas flow leaving the furnace flows horizontally through a cavity and vertically upward through an evaporator and economizer section. The steam and water mixture flows upward through the water cooled furnace.

The two configurations differ only in the manner of ash collection and removal.

Configurations C and D consist of a horizontal furnace fired with a single burner in the front wall. The waterwalls are comprised of intermeshed tubing forming the floor of the unit and integrated into a welded wall covering the sides, roof as well as front and rear walls. Tubes are laid out essentially horizontal, moving sequentially around the furnace. The welded wall construction provides a gas-tight wall surface and prevents ash migration into the insulation or surrounding environment. The cool, smooth surface of the waterwall also assists in minimizing accumulation of ash deposits.

The gas flow leaves the furnace horizontally and flows upward through an evaporator and economizer section similar to configurations A and B.

All four steam generator configurations incorporate an evaporator section which takes the steam water mixture leaving the furnace at approximately 50% quality and converts it to its final condition of 80% quality steam. This arrangement reduces the heat flux during final evaporation from the furnace radiant zone thereby minimizing the risk of deposition or burnout should uneven flow condition occur. The likelihood of deposition or corrosion is further reduced as a result of moving the circuit region, where the highest quality steam is produced, from a radiant zone to a non radiant zone where the heat flux is approximately one third that of the radiant zone.

Dual Economizers are used to reduce the flue gases to their desired exit gas temperature by heat transfer with the incoming boiler feed water.

All configurations incorporate the ability to add calcium sorbent to control SO_2 emissions.

Of the four steam generator configurations considered, the vertical units provide the best design from an ash collection and removal standpoint. The area forming the furnace bottom is one third the size of the horizontal units and the ash collection zone in these units is expected to have a lower range of operating temperatures.

The conventional water cooled ash hopper utilized in configuration A is most suitable when significant quantities and size ranges of agglomerated ash particles are to be removed. However it requires the greatest overall building height and has the highest cost. In contrast the different ash ash disposal and collection system in B is expected to be less costly and require the lowest maintenance and was selected as the basis for design.

Slide 12

The configuration is shown in more detail in this next slide.

To meet the current Canadian federal government guidelines for SO_2 emissions from coal fired utility boilers, utilizing low sulphur western Canadian coal, it is necessary to remove approximately 40% of the SO_2 from the gas stream.

A review of various methods for accomplishing this reduction was carried out in the associated study mentioned previously. Sorbent injection into the furnace by addition to the coal prior to firing was considered to be the economic choice.

If it is required to reduce SO_2 concentrations in the flue gas to lower levels, or a higher sulphur content coal is used, desulphurization of the flue gas may be required. However, this would add significantly to the cost of the steam generation facilities.

The control of NOx (NO and NO_2) emissions is accomplished by minimizing peak flame temperatures and providing for fuel rich conditions at the burner to achieve off-stochiometric conditions within the furnace.

Slide 13

This single burner shown schematically in this slide was used in the design. It is a turbulent burner incorporating both natural gas and coal firing with approximately 90% of the combustion air passing through the burner throat, the remaining 10% combustion air is introduced as staged air.

In addition to combustion air staging at the burner, NOx emissions are reduced by diluting combustion air with flue gas. Flue gas is removed at an intermediate section of the economizer and mixed with ambient air to supply heat for drying the coal in the pulverizer. After drying, the air-gas mixture is used to convey the coal to the burner. This recirculation of flue gas via the pulverizer to the the burner dilutes the combustion air and assists in minimizing peak flame temperatures.

By these methods it is expected that NOx emissions can be controlled to levels at one half the current Canadian federal government guidelines.

Particulate emissions in the flue gas can be controlled prior to exhaust to the atmosphere utilizing either electrostatic precipitators or a baghouse filtering system. Flyash resistivity is dependent on the type of coal and the sorbent used to reduce the SO_2 concentration. The addition of sorbent will also add to the dust loading in the flue gas.

In a prototype facility where a number of different coals and sorbents may be tested, a baghouse filtering system provides the greatest flexibility and was selected for the design developed in the study. For a commercial installation where the coal and sorbent are known before hand, either method of dust capture is technically acceptable and the choice would result from an operational and economic evaluation.

Slide 14

The coal fired steam generation system shown schematically here incorporates the various favoured design aspects. It represents a viable design concept utilizing conventional technology modified to suite the particular steam generating requirements for heavy oil production in Alberta, while at the same time meeting the çurrent Canadian guidelines for the emissions of SO_2 and NOx. The design also provides the basis from which economic evaluations of coal fired versus natural gas fired steam generation may be carried out.

The LNS Burner

At the same time as this design for a coal fired steam generator based on conventional pulverized fuel technology was being developed, Transalta Resources Corporation has been developing a multi-stage slagging type coal fired combuster, the LNS Low NOx SOX Burner suitable for application with oil field steam generators.

The LNS Burner ,

Slide 15

and the demonstration project at Cold Lake has been covered previously in the conference.

Slide 16

This next slide shows the emission targets for the demonstration project.

Slide 17.

These are one half the federal government guidelines for SO_2 emissions and one third the guidelines for NOx emissions. It is in this direction of emmision levels that coal firing technology must progress if it is to meet future requirements, as well as answering the critics of coal as an environmentally acceptable fuel.

Economics

In addition to the availability of viable technologies for coal fired steam generation must be the economic incentive for the heavy oil producers to utilize coal as an alternative fuel to natural gas. To evaluate this incentive a joint industry/government study was carried out in 1989 to evaluate the differential capital and operating costs of burning coal rather than natural gas using as a basis the conventional pulverized coal fired steam generator design previously outlined.

The study, carried out by Delta Projects Inc., developed costs of transporting coal to the Peace River and Cold Lake heavy oil sites by rail from three different coal resource areas in Alberta. Two configurations of clustered steam generators at the heavy oil sites were considered, a single 'six pack' (six steam generators) design and a three by six pack design for determining different levels of fuel requirement.

Higher initial capital costs (three to five times that for gas) were indicated for the coal fired plant. Also, higher operating costs, excluding delivered fuel costs , were expected for the coal option. In addition, given the existing rail infrastructure, significant investments in rail bed upgrading and access to the heavy oil sites would be required for coal transportation, particularly at the higher levels of fuel requirement. However, sufficient differential in energy cost were predicted to occur over time between coal and natural gas to overcome these additional costs.

Project economics were shown to be most sensitive to natural gas prices, coal transportation costs and coal plant capital costs.

The natural gas and coal price forecasts used in the study are based on government projections and indicate a significant gas price increase over the next few years. It should be noted that these projections although just a little over one year old were seen as realistic at the time but can now be considered somewhat optimistic given todays experience with natural gas price contracts and the present outlook for a delay in any rebound to the market.

The study concluded for a project timing in the mid 1990's, that based on the fuel price projections and the design and cost premises used in the study, the use of coal for steam generation in heavy oil in-situ projects in the Peace River and Cold lake areas of Alberta is economic when compared to natural gas.

Slide 18

In conclusion, it can be said that viable technologies are being developed which will allow the utilization of coal rather than natural gas for steam generation in in-situ heavy oil operations and meet or exceed the current Canadian environmental guidelines for acid gas emissions . There is a significant additional capital cost for the coal fired option, however if the predicted future increases in the differential cost of energy between natural gas and coal occur this additional cost can be overcome and coal will be the economic fuel of choice for steam generation in Canada's insitu heavy oil operations.

Looking at what must be the next step,

DELEGATE LIST

ALBERTA COAL RESEARCH CONTRACTOR'S CONFERENCE

Delegate List (Alphabetical by Company) as of October 25, 1991

Luscar Ltd.	Craig	Acott	Director, Technical Services	Edmonton
University of Calgary - Chem. & Petroleum Eng. Dept.	Amir	Badakhshan	Professor	Calgary
TransAlta Utilities Corporation	Cam	Bateman	Reclamation Specialist	Calgary
Canadian Energy Development Inc	D.	Berger	Vice President Development	Edmonton
CANMET Energy Research Laboratories	Alan	Bowles	Manager, Business Development	Ottawa
Alberta Research Council	Yeugenia	Briker	Coal and Hydrocarbon Processing	Devon
The Coal Association of Canada	G.	Capobianco	President & CEO	
University of Calgary Dept of Chem. & Pet. Eng	A.	Chakma		Calgary
Mining Association of B.C.	J.	Chamberlin	Coordinator of Applied Research	Vancouver
Alberta Energy	Kathryn	Cochrane	Director-Minerals Research	Edmonton
Associated Mining Consultants Ltd.	Alan	Craven	Office Manager	Calgary
CANMET - Energy, Mines & Resources Canada	John	Donini	Coal Research Laboratories	Devon
Energy, Mines and Resources Canada	M.	Everell	Assistant Deputy Minister	
Fawcett Mining Consultants Inc	David	Fawcett		Vancouver
Shell Canada Limited	Keith	Firmin	Senior Staff Engineer	Calgary
Unocal Corporation - Science & Tech. Divn	T.	Frankiewicz	Senior Research Associate	Brea
TransAlta Utilities Corporation	Tim	Godfrey	Manager Fuel Supply Planning	Calgary
Alberta Research Council	Arvid	Hardin	Department Head-Coal & Hydrocarbon Proc. Dept.	Devon
Edmonton Power	Doug	Heaton	Environmental Project Engineer	Edmonton
TransAlta Utilities Corporation	Andrew	Hickinbotam	Geologist	Calgary
G. Hoffman Consulting Ltd.	Georgia	Hoffman	Consulting Geologist	Calgary
Smoky River Coal Limited	Edgar	Horner	Manager Plant & Site Services	Grande Cache
Gwil Industries Birtley Coal & Minerals Testing Div.	Frank	Horvat	General Manager	Calgary
University of Calgary	R.	Joshi	Department of Civil Engineering	Calgary
Alberta Energy	Juergen	Kleta	Executive Director, Research & Technology Branch	Edmonton
Loring Laboratories Ltd.	David	Ko	General Manager/Dirc.	Calgary
Alberta Research Council	G.	Kovacik	Research Officer	Nisku
Smoky River Coal Limited	John	Kulach	Senior Mining Engineer	Grande Cache
CN Rail Coal, Sulpher & Potash Marketing	Andrew	Lane	Marketing Officer	Calgary
Alberta Research Council	Willem	Langenberg	Research Geologist	Edmonton
Alberta Energy	Don	Macdonald	Director, Clean Coal Technology - Development	Edmonton
Alberta Energy	Robert	MacKenzie	Technical Writer	Edmonton
Alberta Energy	Norman	MacMurchy	Assistant Deputy Minister	Edmonton
TransAlta Utilities Corporation	M.	McDonald	Manager Research	Calgary
Fording Coal Ltd.	Colin	McKenny	Manager Energy Resource Planning	Calgary
Idemitsu Kosan Co. Ltd. -Canada Office	Masao	Nagao	General Manager	Calgary

ALBERTA COAL RESEARCH CONTRACTOR'S CONFERENCE

Delegate List (Alphabetical by Company) as of October 25, 1991

Petroleum Recovery Institute	H.	Nasr-El-Din	Staff Research Engineer	Calgary
Alberta Geological Survey	Dennis	Nikols	Section Head	Edmonton
Alberta Research Council Coal & Hydrocarbon Recovery	T.	Ohuchi	Associate Research Officer	Nisku
Alberta Energy	Garnet	Page	Coal Development Advisor	Edmonton
Byron Creek Collieries	A.	Peach	Coal Mountain Coal	Sparwood
The Coal Association of Canada	Brian	Raymond	Project Manager, IGCC	Calgary
The University of Calgary	Edward	Rhodes	Dean-Faculty of Engineering	Calgary
Alberta Research Council	R.	Richardson	Alberta Geological Survey	Edmonton
Alberta Research Council	B.	Rottenfusser	Alberta Geological Survey	Edmonton
Smoky River Coal Limited	H.	Sandkuhler	Manager Underground Mines	Grande Cache
Fluor Daniel Canada Inc.	Sujit	Sarkar	Technical Director (Mechanical)	Calgary
Garritty & Baker Drilling 1979 Ltd.	Chris	Schulthess	Consultant	Calgary
Smoky River Coal Limited	Ken	Shinya	Manager, Finance & Planning	Calgary
TransAlta Resources Investment Corporation	B.	Simonson	Manager, LNS Burner Projects	Calgary
CN Rail	Larry	Stevenson	Marketing Officer	Calgary
Manalta Coal Ltd.	John	Taylor	Vice President Technical Services	Calgary
Komex International	M.	Thompson	Engineer	Calgary
Alberta Energy	Al	Turak	Director, Clean Coal Technology - Utilization	Edmonton
Monenco Consultants Ltd	Neil	Wade	Assistant Chief Geotechnical Engineer	Calgary
University of Calgary	M.	Ward	Vice President, Research	
Consultant	William	Wilson	Consultant	Calgary
Energy Resources Conservation Board	R.	Yenter	Staff Geologist	Calgary